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DEPARTMENT OF THE INTERIOR

ALBERT B. FALL, SECRETARY

BUREAU OF MINES

H. FOSTER BAIN, DIRECTOR

ELECTRIC BRASS FURNACE PRACTICE

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Bulletin 202

DEPARTMENT OF THE INTERIOR

ALBERT B. FALL, SECRETARY

BUREAU OF MINES

H. FOSTER BAIN, DIRECTOR

**ELECTRIC BRASS FURNACE
PRACTICE**

BY

H. W. GILLETT and E. L. MACK



**WASHINGTON
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PREFACE.

The Federal Bureau of Mines began its work on the reduction of metal losses in melting nonferrous metals and the improvement of conditions affecting the health and safety of the workmen in the nonferrous industries in 1911.

The electric furnace at once suggested itself as a promising means of reducing metal loss and improving working conditions. No electric furnace suitable for melting brass was then in existence. The bureau promptly began to study the problem experimentally and to encourage brass melters, electric-furnace designers, and electric generating stations to study it also.

Since that time a great amount of experimental work on, and commercial development of, electric brass furnaces has been done by different groups commercially interested, as well as by the bureau, with the result that electric melting has been found to give the advantages expected, and to be industrially desirable; it is acknowledged to-day to be usually the best and cheapest way to melt brass and many other nonferrous alloys. Apparently it is only a matter of a few years before more brass will be melted by electricity than by fuel. The electric furnace bids fair to become the standard type for brass melting.

The present report is published to record the progress so far made in melting brass electrically; to aid the plants which have not yet taken up such melting by pointing out the types of furnaces available, describing their performance and indicating their possibilities and their limitations; and to encourage further experimentation with and the development and installation of electric brass furnaces.

ELECTRIC BRASS FURNACE PRACTICE.

By H. W. GILLETT and E. L. MACK.

INTRODUCTION.

HISTORICAL REVIEW.

Prior to 1911 the literature on melting brass by electricity consisted entirely—save for some suggestions made in patent literature but not actually worked out—of a few observations by farseeing men¹ on the theoretical advantages of melting brass and other nonferrous alloys in electric furnaces.

Roeber was the first to see and clearly state the possible advantages of the electric furnace in brass melting. The same factors that make it desirable to manufacture special steels in the electric furnace rather than in crucibles, he declared, apply to brass as well. He pointed out the possibility of using larger melting units, of eliminating crucible cost, of readily controlling the purity of the product, and of nearly eliminating the loss of zinc. "The replacement of the old crucible process is sure to come," he said; "perhaps it will come in two years, but it will, necessarily, in ten years. . . . The inevitable advance along this industrial line will have great influence in the metal world and will also provide, in large cities, a new kind of load for the central station."

The years 1905 to 1910 may be termed the purely theoretical period of the development of the electric brass furnace, though the Conley furnace was tried out experimentally about 1910 and a few heats were made in an induction furnace.²

In December, 1911, the American Chemical Society held a symposium on mineral wastes, at which Bassett³ discussed the zinc losses that attend making brass in fuel-fired furnaces.

¹ Roeber, E. F., *Manufacture of brass in the electric furnace*: *Electrochem. and Met. Ind.*, vol. 3, 1905, p. 4; Moldenke, R., *Electric furnaces for the foundry*: *Electrochem. and Met. Ind.*, vol. 1, 1909, p. 160; Krom, L. J., *Development of melting furnaces*: *Metal Ind.*, vol. 7, 1909, p. 287; Bragg, C. T., *Modern brass-foundry progress*: *Trans. Am. Brass Founders' Assn.*, vol. 4, 1910, p. 43; Richards, J. W., *Electric furnaces in nonferrous metallurgy*: *Met. and Chem. Eng.*, vol. 8, 1910, p. 233.

² Rowlands, T., *Induction-furnace progress*: *Trans. Am. Electrochem. Soc.*, vol. 17, 1910, p. 103.

³ Bassett, W. H., *Zinc losses*: *Jour. Ind. and Eng. Chem.*, vol. 4, 1912, p. 164.

In the discussion of this paper, Parsons⁴ pointed out that in a closed electric furnace zinc losses should be minimized, and said that the Bureau of Mines was planning to take up the question of electric furnaces for melting nonferrous alloys.

About this time several electric furnaces⁵ were advertised, advocated, or suggested for melting brass, seemingly either on the basis of tests with zinc, on the basis of a run or two on copper, or on even less experience, except for the Hering furnace.

None of these furnaces are now prominent in the brass industry, although the final outgrowth of Clamer's work with the Hering furnace was the Ajax-Wyatt furnace, a type of great commercial importance.

Bensel⁶ gave in 1912 the results on a few runs on brass in a small experimental laboratory furnace.

Various small-scale tests were made with different types of furnaces by different experimenters in 1913 without any notable progress resulting. The period 1911 to 1913 may be termed the early, small-scale experimental period of the electric brass furnace.

Late in 1913, in 1914 and 1915, experiments began with fair-sized furnaces under foundry conditions, the Rennerfelt being tested in Sweden, the Helberger, Hering, and two early forms of the Baily, the Ajax-Wyatt, a Snyder crucible lift-out, and a Hoskins crucible lift-out, in the United States. Of these only the Rennerfelt and Ajax-Wyatt survived. The years 1913 to 1915 comprised the period of early larger-scale experiments.

Early in 1916 the Ajax-Wyatt reached a close approximation to its present form and began to melt brass commercially. At the same time a small Snyder direct-arc furnace gave promising results on leaded bearing-bronze and was the precursor of an installation of two 1-ton Snyder furnaces which began commercial production later in the year.

The present form of the Baily furnace began to melt commercially "Lumen" metal, an alloy containing 85 per cent zinc, with occasional test runs on brass. The General Electric furnace was

⁴Parsons, C. L., Discussion: Metal Ind., vol. 10, 1912, pp. 240-241; see also Parsons, C. L., Notes on mineral wastes: Bull. 43, Bureau of Mines, 1912, p. 20.

⁵Anonymous, The Helberger furnace: Metal Ind., vol. 9, 1911, p. 212; advertisements; see also Wile, R. S., An electric furnace for treatment of tin dross, concentrates, and other metallurgical work: Met. and Chem. Eng., vol. 10, 1912, p. 495; Weeks, C. A., Melting nonferrous metals in an electric furnace: Met. and Chem. Eng., vol. 9, 1911, p. 363; see also Hansen, C. A., Electric melting of copper and brass: Trans. Am. Inst. of Metals, vol. 6, 1912, p. 110; Clamer, G. H., and Hering, C., The electric furnace for brass melting: Trans. Am. Inst. Metals, vol. 6, 1912, p. 99; Thomson-Fitzgerald, F. A. J., A new electric resistance furnace: Trans. Am. Electrochem. Soc., vol. 19, 1911, p. 283.

⁶Bensel, F., Versuche zur Verminderung der Metall-verluste beim Messingschmelzen: Metallurgie, vol. 9, 1912, p. 533. Abstracted in Chem. Abs., vol. 6, 1912, p. 3256; Jour. Soc. Chem. Ind., vol. 31, 1912, p. 879; Journal Inst. Metals (British), vol. 8, 1912, p. 353; Jour. Franklin Inst., vol. 175, 1913, p. 150.

melting on a semicommercial scale at Schenectady, and experimental melting was done with the Foley furnace.

In 1917 came the installation in this country of commercial Rennerfelt furnaces for brass and bronze, marked extension of the use of the Ajax-Wyatt and the Baily, a trial of the General Electric at a Chicago plant, and a foundry-scale test of a rocking furnace.

By the end of 1918 three types of furnaces—the Ajax-Wyatt, the Baily, and the rocking furnace—had forged ahead and were doing the bulk of the commercial electric-furnace melting.

Table 1 shows the number of active furnaces installed or being installed for melting brass and bronze on January 1, 1922.

TABLE 1.—*Electric furnaces in active commercial use or being installed for nonferrous alloys in the United States, Jan. 1, 1922.*

DIRECT-ARC FURNACES.

Make.	No.	User.	Alloy.	Capacity tons per heat per furnace.	Total capacity tons.	Total kw. rating.	Total kw.
Heroult e.....	2	Driver-Harris Co., Harrison, N. J.	Nickel alloys....	2	4	800	b 1,600
Greaves - Etchell.....	1	Hoskins Manufacturing Co., Detroit, Mich.	do.....	$\frac{1}{2}$	$\frac{1}{2}$	300	b 300
Snyder.....	1	Monel Metal Products Co., Bayonne, N. J. c	Monel.....	$\frac{1}{2}$	$\frac{1}{2}$	100	d 100
	1	Chrobalitic Tool Co., Chicago, Ill.	Cobalt-chromium.....	$\frac{1}{2}$	$\frac{1}{2}$	60	d 60
	3	Haynes Stellite Co., Kokomo, Ind.	do.....	$\frac{1}{2}$	$\frac{1}{2}$	60	d 240
	2	Chicago Bearing Metal Co., Chicago, Ill.	Leaded bronze..	1	2	400	e 800
Total.....	10	Number of users, 6.			7.5		2,100

STATIONARY INDIRECT-ARC FURNACES.

Rennerfelt f.....	1	Chicago Bearing Metal Co., Chicago, Ill.	Leaded bronze..	1	1	300	A 300
	1	U. S. Mint, Philadelphia, Pa.	Bronze and silver	$\frac{1}{2}$	$\frac{1}{2}$	150	d 150
	1	do.....	do.....	$\frac{1}{2}$	$\frac{1}{2}$	300	d 300
	1	U. S. Mint, San Francisco, Calif.	do.....	$\frac{1}{2}$	$\frac{1}{2}$	125	125
	1	Bausch Machine Tool Co., Springfield, Mass.	Hardeners for aluminum.	$\frac{1}{2}$	$\frac{1}{2}$	100	100
	1	Hardite Metals Co., Long Island City, N. Y.	Nickel-chromium.	$\frac{1}{2}$	$\frac{1}{2}$	150	150
	1	Gorham Manufacturing Co., Providence, R. I.	Silver.....	$\frac{1}{2}$	$\frac{1}{2}$	100	A 100
	1	Skagit Iron & Steel Co., Sedro Wooley, Calif.	Bronze.....	$\frac{1}{2}$	$\frac{1}{2}$	300	A 300

e Another Heroult furnace melting nickel alloys is installed at Hiram Walker Metal Products Co., Walkerville, Ontario, Canada.

b Three-phase automatic control.

c According to an editorial in *Foundry* (vol. 49, 1921, p. 583), a Moore furnace (size not given) has recently been ordered for the International Nickel Co., Huntington, W. Va., for melting Monel metal.

d Single phase.

e Single phase, automatic control.

f A 1-ton 100-kw. Rennerfelt for melting Monel, bronze, and brass up to 22 per cent zinc is installed at the Gerling Brass Foundry Co., Kalamazoo, Mich., but has not been in service for some time on account of power shortage.

g A second 1-ton Rennerfelt at this plant has been made over into a Re-pel-arc furnace.

h Two phase.

i Two phase; nose tilting and automatic control.

TABLE 1.—*Electric furnaces in active commercial use, etc.*—Continued.

STATIONARY INDIRECT-ARC FURNACES—Continued.

Make.	No.	User.	Alloy.	Capacity tons per heat per furnace.	Total capacity tons.	Total kw. rating.	Total kw.
Re-pel-arc /	1	Chicago Bearing Metal Co., Chicago, Ill.	Leaded bronze..	1	1	450	± 450
	2	Crane Co., Chicago, Ill.	Monel and bronze	$\frac{1}{2}$	$\frac{1}{2}$	50	± 100
	1	Sandusky Foundry & Machine Co., Sandusky, Ohio.	Bronze.....	$\frac{1}{2}$	$\frac{1}{2}$	50	± 50
	1	Manning, Maxwell & Moore, Inc., Bridgeport, Conn.	do.....	$\frac{1}{2}$	$\frac{1}{2}$	50	± 50
	1	Bridgeport Brass Co., Bridgeport, Conn.	do.....	$\frac{1}{2}$	$\frac{1}{2}$	70	± 70
Total.....	14	Number of users, 11.			7.1		2,245

MOVING INDIRECT-ARC FURNACES—SINGLE PHASE.¹

Detroit rocking.	2	Aluminum Manufacturing Co., Inc., Detroit, Mich.	Brass and bronze	1	2	300	600
	2	American Bushing Corp. Marysville, Mich.	do.....	1	2	300	600
	2	American Manganese Bronze Co., Holmesburg, Pa.	do.....	1	2	300	600
	1	American Metal Products Co., Milwaukee, Wis.	Bronze.....	$\frac{1}{2}$	$\frac{1}{2}$	150	± 150
	1	Bethlehem Shipbuilding Co., Redington, Pa.	Brass and bronze	1	1	300	300
	5	C. B. Bohn Foundry Co., Detroit, Mich.	Brass, bronze, and aluminum	1	5	300	1,500
	2	Bound Brook Oilless Bearing Co., Bound Brook, N. J.	Bronze.....	$\frac{1}{2}$	1	150	300
	1	Bridgeport Brass Co., Bridgeport, Conn.	Bronze and copper.	1	1	300	300
	1	Chapman Valve Mfg. Co., Indian Orchard, Mass.	Brass and bronze	$\frac{1}{2}$	$\frac{1}{2}$	125	125
	4	Chase Metal Works, Waterbury, Conn.	Brass.....	1	4	300	1,200
	1	Cleveland Brass & Copper Rolling Mills, Cleveland, Ohio.	do.....	1	1	300	300
	8 ^a	Detroit Copper and Brass Rolling Mills, Detroit, Mich.	do.....	1	8	300	± 2,400
	1	Detroit Lubricator Co., Detroit, Mich.	Brass and bronze	1	1	300	± 300
	1	Ford Motor Co., Detroit, Mich.	do.....	1	1	300	300
	1	Ford Motor Co., River Rouge, Mich.	do.....	1	1	300	300
	2	General Aluminum & Brass Manufacturing Co., Detroit, Mich.	do.....	1	2	300	600
	1	General American Tank Car Corp., Chicago, Ill.	do.....	$\frac{1}{2}$	1	150	150
	1	Lumen Bearing Co., Buffalo, N. Y.	do.....	$\frac{1}{2}$	$\frac{1}{2}$	150	± 150
	1	Hills, McCanna Co., Chicago, Ill.	do.....	$\frac{1}{2}$	$\frac{1}{2}$	75	75
	2	Michigan Lubricator Co., Detroit, Mich.	do.....	1	2	300	600
	4	Michigan Smelting & Refining Co., Detroit, Mich.	do.....	1	4	300	1,200
	2	Mueller Metals Co., Port Huron Mich.	Brass.....	1	2	300	600
	1	Oregon Brass Works, Portland, Oreg.	Brass and bronze	1	1	300	300
	1	do.....	do.....	$\frac{1}{2}$	$\frac{1}{2}$	150	150
	3	Parish Pool Co., Cleveland, Ohio.	do.....	1	3	300	± 900

¹ The repelling-arc furnace may be operated either as direct or indirect arc furnace.² Three phase.³ A three-phase oscillating indirect-arc furnace, the Volta, has been tried out in a 1-ton 360-kw. size at the Titanium Bronze Co., Niagara Falls, but that firm has ceased operations. A demonstration furnace is installed at The Fitzgerald Laboratories, Inc., Niagara Falls, N. Y. Three furnaces, of 1, $\frac{1}{2}$, and $\frac{1}{4}$ ton capacities have been ordered, but none are to be installed at once.^a Being installed.^b Six installed, two being installed.^c Direct pouring furnaces.^d Automatic control.^e Two direct pouring.

TABLE 1.—Electric furnaces in active commercial use, etc.—Continued.

MOVING INDIRECT—ARC FURNACES—SINGLE PHASE—Continued.

Make.	No.	User.	Alloy.	Capacity tons per heat per furnace.	Total capacity tons.	Total kw. rating.	Total kw.
Detroitrocking.	1	Standard Underground Cable Co., Perth Amboy, N. J.	Brass and copper.	1½	1½	300	300
	1	Storm Peak Potash Co., Denver, Colo.	Aluminum.....	½	½	150	150
	3	Sherwood Brass Works, Detroit, Mich.	Brass and bronze	1	3	300	900
	1	Sarachan & Rosenthal, Inc., Rochester, N. Y.	Brass.....	1	1	300	300
	1	M. Smolensky Manufacturing Co., Cleveland, Ohio.	do.....	½	½	75	75
	1	White & Bros., Philadelphia, Pa.	Brass and bronze	1	1	300	30
	1	Wheeler Condenser & Engineering Co., Carteret, N. J.	do.....	1	1	300	300
Total re.....	60	Number of users, 30.			54.4		16,325
Booth.....	3			1	3	300	900
	4			½	2	180	720
	25			½	6½	125	9,125
	8			½	2	75	600
Total.....	40	Number of users of Booth furnaces, 35 to 40.			13½		5,345
Total for type..	100	Number of users of type, about 65.			67.6		21,670

REFLECTED-HEAT FURNACES.

Baily.....	1	Acheson Manufacturing Co., Rankin, Pa.	Brass and bronze	½	½	105	105
	1	Akron Bronze & Aluminum Co., Akron, Ohio.	do.....	½	½	105	105
	1	do.....	do.....	½	½	50	50
	1	Allis Manufacturing Co., Milwaukee, Wis.	do.....	½	½	50	50
	1	Alliance Brass & Bronze Co., Alliance, Ohio.	do.....	½	½	75	75
	1	American Hardware Corp., New Britain, Conn.	do.....	½	½	105	105
	1	Anaconda Copper Mining Co., Butte, Mont.	do.....	½	½	105	105
	1	Atlas Brass Works, Columbus, Ohio.	do.....	½	½	105	105
	1	Bagley & Sewall Co., Watertown, N. Y.	do.....	½	½	50	50
	2	Buick Motor Car Co., Flint, Mich.	do.....	½	1½	105	210
	2	Burlington Brass Works, Burlington, Wis.	do.....	½	1½	105	210
	2	Capitol Brass Co., Detroit, Mich.	do.....	½	1½	105	210
	1	Chapman Valve Co., Springfield, Mass.	do.....	½	½	105	105
	1	Coppus Engineering & Equipment Co., Worcester, Mass.	do.....	½	½	105	105
	1	Claus Auto Gas Cock Co., Milwaukee, Wis.	do.....	½	½	75	75

* Direct pouring, standard furnace, lined for larger charge.

re Since this table was put in type the Ford Motor Co. is reported to have ordered five more 1-ton 300 kw. Detroit rocking furnaces, all having automatic control.

* The number of furnaces of each size is given according to information supplied by the makers of the furnace. The number of active furnaces may be less, as it is impossible to check up the full list of users, because it is not given out by the makers. Among the users are Bridgeport Brass Co., Bridgeport, Conn.; Cleveland Brass Co., Cleveland, Ohio; Dearborn Brass Co., Cedar Rapids, Iowa; Dallas City Foundry Co., Dallas City, Ill.; Fulton-Harwood Brass Works, South Bend, Ind.; National Bronze and Aluminum Foundry, Cleveland, Ohio; Michigan Smelting and Refining Co., Detroit, Mich.; Muskegon Aluminum Foundry Co., Muskegon, Mich.; Yale & Towne, Stamford, Conn.; Hill Pump Valve Co., Chicago, Ill.; Nordyke & Marmon Co., Indianapolis, Ind.; Prime Manufacturing Co., Milwaukee, Wis. Some of the Booth furnaces have automatic control.

TABLE 1.—*Electric furnaces in active commercial use, etc.*—Continued.

REFLECTED-HEAT FURNACES—Continued.

Make.	No.	User.	Alloy.	Capacity tons per heat per furnace.	Total capacity tons.	Total kw. rating.	Total kw.
Baily.....	1	Dayton Engineering Laboratories Co., Dayton, Ohio.	Aluminum.....	‡	‡	105	105
	2	do.....	Brass and bronze	‡	‡	50	100
	1	Deming Co., Salem, Ohio.	do.....	‡	‡	50	50
	3	Detroit Brass Works, Detroit, Mich.	Brass.....	‡	2‡	105	315
	1	Drew Electric & Manufacturing Co., Cleveland, Ohio.	Brass and bronze	‡	‡	105	105
	1	Evinrude Motor Co., Milwaukee, Wis.	do.....	‡	‡	75	75
	1	Flood City Manufacturing Co., Johnstown, Pa.	do.....	‡	‡	75	75
	1	Hays Manufacturing Co., Erie, Pa.	Brass.....	‡	‡	105	105
	1	Kayline Co., Cleveland, Ohio.	Brass and bronze	‡	‡	50	50
	1	Oscar Krenz Copper & Brass Works, San Francisco, Calif.	do.....	‡	‡	75	* 75
	1	Kennedy Valve Co., Elmira, N. Y.	do.....	‡	‡	105	105
	1	Landers, Frary & Clark, New Britain, Conn.	Aluminum.....	‡	‡	50	50
	2	Lumen Bearing Co., Buffalo, N. Y.	Brass and bronze	‡	1‡	105	210
	1	do.....	Lumen metal...	‡	‡	105	105
	1	Miller Pasteurizing Machine Co., Canton, Ohio.	Brass and bronze	‡	‡	50	50
	4	Michigan Smelting & Refining Co., Detroit, Mich.	Brass.....	‡	3	105	420
	1	A. Y. McDonald, Co., Dubuque, Iowa.	Brass and bronze	‡	‡	105	105
	1	McKenna Brass Works, Pittsburgh, Pa.	do.....	‡	‡	105	105
	1	do.....	do.....	‡	‡	50	50
	1	Metric Metals Co., Erie, Pa.	do.....	‡	‡	75	75
	3	McRae-Roberts Co., Detroit, Mich.	do.....	‡	2‡	105	315
	1	W. A. Mills, Port Chester, N. Y.	do.....	‡	‡	75	75
	3	H. Mueller Manufacturing Co., Decatur, Ill.	do.....	‡	2‡	105	* 315
	3	Nolte Brass Co., Springfield, Ohio.	do.....	‡	2‡	105	315
	1	Oro Metals Co., Los Angeles, Calif.	do.....	‡	‡	105	105
	1	H. Orne & Sons, St. Paul, Minn.	do.....	‡	‡	75	75
	1	Penberthy Injector Co., Detroit, Mich.	do.....	‡	‡	105	105
	1	Pittsburgh Valve & Fixture Co., Barberton, Ohio.	do.....	‡	‡	105	105
	1	Chas. D. Reeve Co., Grand Rapids, Mich.	do.....	‡	‡	50	50
	1	Regent Brass Works, Marysville, Ohio.	do.....	‡	‡	105	105
	1	Rundle Manufacturing Co., Milwaukee, Wis.	do.....	‡	‡	75	75
	1	Springfield Brass Works, Springfield, Ohio.	do.....	‡	‡	105	105
	5	Standard Sanitary Co., Louisville, Ky.	do.....	‡	3‡	105	525
	1	Sperry Gyroscope Co., New York, N. Y.	do.....	‡	‡	75	75
	1	S. H. Thomson Manufacturing Co., Dayton, Ohio.	do.....	‡	‡	50	50
	2	Union Brass & Metals Manufacturing Co., St. Paul, Minn.	do.....	‡	1‡	105	210
	1	Union Screen Plate Co., Fitchburg, Mass.	do.....	‡	‡	75	75
	1	U. S. Copper Products Co., Cleveland, Ohio.	Brass.....	‡	‡	105	* 105
	1	U. S. Navy Yard, Washington, D. C.	Brass and bronze	‡	‡	105	* 105
	1	U. S. Smelt., Ref., and Mining Co., Boston, Mass.	do.....	‡	‡	75	75

‡ Being installed.

* Not tilting.

TABLE 1.—*Electric furnaces in active commercial use, etc.*—Continued.

REFLECTED-HEAT FURNACES—Continued.

Make.	No.	User.	Alloy.	Capacity tons per heat per furnace.	Total capacity tons.	Total kw. rating.	Total kw.
Bally.....	1	Utica Valve & Fixture Co., Utica, N. Y.	Brass and bronze	$\frac{1}{2}$	$\frac{1}{2}$	105	105
	1	Westinghouse Electric Manufacturing Co., East Pittsburgh, Pa.	do.....	$\frac{1}{2}$	$\frac{1}{2}$	50	50
	2	West Virginia Metal Products Co., Fairmont, W. Va.	do.....	$\frac{1}{2}$	$1\frac{1}{2}$	105	210
	1	White & Bros., Philadelphia, Pa.	do.....	$\frac{1}{2}$	$\frac{1}{2}$	105	105
	1	J. B. Wise, Inc., Watertown, N. Y.	do.....	$\frac{1}{2}$	$\frac{1}{2}$	75	75
	1	Wolverine Brass Works, Grand Rapids, Mich.	do.....	$\frac{1}{2}$	$\frac{1}{2}$	50	50
Total *	83	Number of users, 57			51.7		7,455
General Electric.	1	General Electric Co., Schenectady, N. Y.	Aluminum.....	$\frac{1}{2}$	$\frac{1}{2}$	75	75
	1	do.....	do.....	$\frac{1}{2}$	$\frac{1}{2}$	150	150
	1	do.....	Brass and bronze	$\frac{1}{2}$	$\frac{1}{2}$	250	250
	1	McNab & Harlin Manufacturing Co., Paterson, N. J.	"Aterite".....	$\frac{1}{2}$	$\frac{1}{2}$	300	300
	1	do.....	do.....	$1\frac{1}{2}$	$1\frac{1}{2}$	400	400
	1	U. S. Copper Products Co., Cleveland, Ohio.	Brass and copper.	$1\frac{1}{2}$	$1\frac{1}{2}$	400	400
	1	Ohio Brass Co., Mansfield, Ohio.	Brass and bronze	$1\frac{1}{2}$	$1\frac{1}{2}$	400	400
	1	U. S. Navy Yard, Washington, D. C.	do.....	$1\frac{1}{2}$	$1\frac{1}{2}$	400	400
Total	8	Number of users, 5			5.75		2,375
Rennerfelt reverberatory.	1	Chicago Faucet Co., Chicago, Ill.		$\frac{1}{2}$	$\frac{1}{2}$	75	75
Total for type..	92	Number of users, 62			57		9,730

VERTICAL-RING INDUCTION FURNACES—SINGLE PHASE.

Ajax-Wyatt....	3	Ajax Metal Co., Philadelphia, Pa.	Yellow brass....	$\frac{1}{2}$	$\frac{1}{2}$	30	90
	1	do.....	do.....	$\frac{1}{2}$	$\frac{1}{2}$	60	60
	32	American Brass Co., Waterbury, Conn.	do.....	$\frac{1}{2}$	16	60	1920
	24	American Brass Co., Kenosha, Wis.	do.....	$\frac{1}{2}$	12	60	1,440
	24	American Brass Co., Torrington, Conn.	do.....	$\frac{1}{2}$	12	60	1,440
	1	American Hardware Co., New Britain, Conn.	Yellow and red brass.	$\frac{1}{2}$	$\frac{1}{2}$	30	30
	6	Baltimore Tube Co., Baltimore, Md.	Yellow brass....	$\frac{1}{2}$	3	60	360
	17	Bridgeport Brass Co., Bridgeport, Conn. "	do.....	$\frac{1}{2}$	$5\frac{1}{2}$	50	850
	6	do.....	do.....	$\frac{1}{2}$	$4\frac{1}{2}$	80	480
	1	Buick Motor Car Co., Flint, Mich.	do.....	$\frac{1}{2}$	$\frac{1}{2}$	60	60
	26	Chase Rolling Mill Co., Waterbury, Conn.	do.....	$\frac{1}{2}$	13	60	1,440
	1	Crane Co., Chicago, Ill.	Red brass.....	$\frac{1}{2}$	$\frac{1}{2}$	60	60
	1	Erie-Buffalo Tube Co., Erie, Pa.	Yellow brass....	$\frac{1}{2}$	$\frac{1}{2}$	60	60
	1	General Electric Co., Schenectady, N. Y.	do.....	$\frac{1}{2}$	$\frac{1}{2}$	60	60
	1	Haines, Jones & Cadbury, Philadelphia, Pa.	Red brass.....	$\frac{1}{2}$	$\frac{1}{2}$	60	60
	6	National Conduit & Cable Co., Hastings, N. Y.	Yellow brass....	$\frac{1}{2}$	3	60	360

* Nose tilting.

* Two 105-kw. furnaces are also used by Metal & Thermit Co., Chicago, for tin recovery. Other Bally furnaces are installed in Canada and Mexico as follows: Canada, Empire Brass Works, one 105-kw.; Monarch Metal Co., one 50-kw.; Dominion Steel Products Co., one 50-kw.; Mexico, G. Amsinck Corp., two 105-kw.

* This firm originally used mostly the 30-kw. size, but has changed to a smaller number of higher-powered furnaces.

TABLE 1.—*Electric furnaces in active commercial use, etc.*—Continued.

VERTICAL—RING INDUCTION FURNACES—SINGLE PHASE—Continued.

Make.	No.	User.	Alloy.	Capacity tons per heat per furnace.	Total capacity tons.	Total kw. rating.	Total kw.
Ajax-Wyatt.....	1	Raritan Copper Works, Raritan, N. J.	Yellow brass....	$\frac{1}{2}$	$\frac{1}{2}$	60	60
	1	Parish Pool Co., Cleveland, Ohio.	do.....	$\frac{1}{2}$	$\frac{1}{2}$	60	60
	3	Rome Brass & Copper Co., Rome, N. Y.	do.....	$\frac{1}{2}$	1 $\frac{1}{2}$	60	180
	12	Scoville Manufacturing Co., Waterbury, Conn.	do.....	$\frac{1}{2}$	6	60	720
	1	Speakman Co., Wilmington, Del.	do.....	$\frac{1}{2}$	$\frac{1}{2}$	60	60
	1	Specialty Brass Co., Kenosha, Wis.	do.....	$\frac{1}{2}$	$\frac{1}{2}$	60	60
	1	Stamford Rolling Mills Co., Stamford, Conn.	do.....	$\frac{1}{2}$	$\frac{1}{2}$	60	60
	1	Standard Sanitary Manufacturing Co., Louisville, Ky.	do.....	$\frac{1}{2}$	$\frac{1}{2}$	75	75
	2	Standard Underground Cable Co., Perth Amboy, N. J.	do.....	$\frac{1}{2}$	$\frac{1}{2}$	60	120
	6	West Virginia Metal Products Co., Fairmont, W. Va.	do.....	$\frac{1}{2}$	3	60	360
	1	Western Cartridge Co., W. Alton, Ill.	do.....	$\frac{1}{2}$	$\frac{1}{2}$	60	60
	1	U. S. Navy Yard, Washington, D. C.	do.....	$\frac{1}{2}$	$\frac{1}{2}$	60	60
Total.....	176	Number of users, 24.....			84.3		14,085

HORIZONTAL-RING INDUCTION FURNACES—SINGLE PHASE.

General Electric.	1	Hoskins Manufacturing Co., Detroit, Mich.	Nickel-chromium	$\frac{1}{2}$	$\frac{1}{2}$	100	100
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CONTACT-RESISTANCE FURNACES—THREE PHASE.

Bennett.....	1	Scoville Manufacturing Co., Waterbury, Conn.	Brass and copper.	5	5	500	500
	6	do.....	do.....	1	6	150	900
Total.....	7	Number of users, 1.....			11		1,400

HIGH-FREQUENCY EDDY-CURRENT FURNACES.

Ajax-Northrup.	1	Baker & Co., Newark, N. J.	Platinum.....			18	≈ 18
	1	H. A. Wilson Co., Newark, N. J.	do.....			20	≈ 20
	1	D. Belais, New York City.....	White gold.....			20	≈ 20
	1	Goldsmith Bros. Smelting & Refining Co., Chicago, Ill.	Precious metals.....			20	≈ 20
	2	U. S. Mint, Philadelphia, Pa.	Silver and gold.....			8	≈ 8
	2	do.....	do.....			16	≈ 32
	1	Handy & Harman, Bridgeport, Conn.	Silver.....	$\frac{1}{2}$	$\frac{1}{2}$	60	≈ 60
Total.....	9	Number of users, 6.....			$\frac{1}{2}+$		188

$\approx$ Single phase, crucible lift-out.

$\approx$ Two phase, crucible lift-out.

$\approx$ Three phase, nose tilting.

∞ The number of Bennett furnaces in use by the Scoville Mfg. Co. has been increased by about a dozen since this table was prepared.

TABLE 1.—*Electric furnaces in active commercial use, etc.*—Continued.
OTHER CRUCIBLE LIFT-OUT FURNACES.

Make.	No.	User.	Alloy.	Capacity tons per heat per furnace.	Total capacity tons.	Total kw. rating.	Total kw.
Baily.....	1	W. A. Rogers, Ltd., Niagara Falls, N. Y.	Silver.....	½	½	40	cc 40
	1	U. S. Smelt, Ref. & Mining Co., Boston, Mass.					
Marsh.....	2	bb Hoskins Manufacturing Co., Detroit, Mich.	Nickel-chromium	½	½	50	cc 100
Total.....	4	Number of users, 3.....			½		140

SUMMARY.

Make.	Number of furnaces.	Total capacity per heat tons.	Total kilowatts.
Ajax-Wyatt.....	176	84.2	14,085
Detroit rocking.....	60	54.4	16,325
Baily.....	83	51.7	7,455
Booth.....	40±	13.2	5,345
Other furnaces.....	54	30.1	7,661
Total.....	413	233.7	50,861

aa Single phase, two crucibles.

bb These furnaces are used less than formerly on account of the installation of larger furnaces of other types.

cc Single phase, one or two crucibles per furnace.

Various other types are still under experimental development.

After the year 1916 began the period of commercial introduction and development of electric brass furnaces. The status of the industry, which was still in its infancy, was thus given in a review⁷ of electrometallurgy in 1919:

In the nonferrous metal industries, 1919 was characterized by the rapid introduction of electric melting furnaces in place of melting in crucibles or open-combustion furnaces. The electric furnace had a great opportunity and it was grasped. Fifty or more electric furnaces were erected in a single brass-producing city, and all the other brass makers in the country are either looking on with interest or have determined to follow suit. America is leading the world in this development, and the brass and other related industries of other countries can not do otherwise than imitate our methods and follow our lead.

Richards^a says: "It will be but a short time before all the brass in this country will be melted by electric furnaces."

The Metal Industry^b said, in May, 1919: "We can now repeat with more emphasis what we said in an editorial last October: 'The electric furnace has arrived.'"

The importance of the development of electric brass furnaces will be made clearer by considering the savings possible by their use.

⁷ Electrometallurgy in 1919: Chem. and Met. Eng., vol. 22, 1920, p. 2.

^a Richards, J. W., Electrochemistry: Elect. Rev., vol. 74, 1919, p. 992; compare also Parsons, F. W., Everybody's business, electric heat to-morrow: Sat. Even. Post., vol. 192, 1920, Jan. 24, pp. 39, 73.

^b Electric-furnace progress: Metal Ind. vol. 17, 1919, p. 230; see also vol. 18, 1920, p. 65; see also Cone, E. F., Electric melting in nonferrous industry: Iron Age, vol. 105, 1920, p. 655.

MONETARY SAVING POSSIBLE BY ADOPTION OF ELECTRIC BRASS MELTING.

The saving due to the adoption of electric brass melting will, of course, vary with the conditions and the location of the plant, but suppose that electric melting had progressed so far that all melting had been done electrically during the war. In 1917, according to the War Industries Board,¹⁰ 1,072,508,000 pounds of brass and bronze were made, mostly yellow brass. To produce this, at least a third as much more scrap and crop ends had to be melted. The total melt may be estimated at 1,430,000,000 pounds.

Metal losses were high during the rush of war production, and it seems very probable that the melting loss with electric furnaces would have been 2 per cent less than with crucibles, so that the saving would have been about 28,500,000 pounds of metal. This lost metal was mainly zinc, having an average price, for rolling-mill spelter, of 11 cents,¹¹ but partly copper, with an average of 31 cents.¹¹ On the assumption that this 2 per cent loss was $1\frac{1}{2}$ per cent zinc and $\frac{1}{2}$ per cent copper, the average value of the lost metal was then 16 cents and the possible saving in metal alone was \$4,500,000. As crucibles and labor were both higher than ever before, an equal amount, at least, should be allowed for savings through not using crucibles and approximately \$1,000,000 through savings in labor, or a total of \$10,000,000.

The domestic consumption of copper rose from 1,316,463,754 pounds in 1917 to 1,800,000,000 pounds, or thereabout, in 1918, and the brass production doubtless increased in about the same proportion, or enough to offset the slightly lower prices of zinc, about 10 cents, and of copper, about 25 cents.¹² Crucibles and labor still remained high, so it is reasonable to figure that if all brass made in the country could have been melted electrically in 1917 and 1918 the war expenses of the country would have been reduced some \$20,000,000.

In 1919, when the domestic consumption of copper was 800,000,000 pounds, that of brass and bronze was, say, 600,000,000 pounds, and the capacity¹³ of the copper-using plants was over twice that figure, including that from secondary metals. The total melt to produce that amount of brass and bronze may be assumed as 800,000,000 pounds. With the average 1919 price of copper as $19\frac{1}{2}$ and that of zinc as 8 cents per pound, and the metal saving to be made by electric melt-

¹⁰ Mitchell, W. C., Aldrich, H. R., and Schmuckler, J., *History of prices during the war: War Industries Board publication, 1919; Foundry, vol. 47, 1919, p. 862; Metal Industry (London), vol. 15, 1919, p. 455.*

¹¹ Anonymous, *Waterbury average: Metal Ind., vol. 16, 1918, p. 582.*

¹² Anonymous, *Waterbury average: Metal Ind., vol. 17, 1919, p. 108; vol. 18, 1920, p. 110.*

¹³ See Bregman, A., *The distribution of copper; Metal Ind., vol. 18, 1920, p. 220.*

ing taken as $1\frac{1}{2}$ per cent, or about 6,000 tons (say 4,000 tons of zinc and 2,000 tons of copper), the preventable loss of metal figures out as more than a third of a million dollars. This figure is based only on the net saving of metal and does not include the money saved by the avoidance of the need of recovering from the slag and ashes much metal that has thus to be recovered in fuel-fired practice. The saving due to reduction of gross loss is not far from that due to reduction of net loss. Add to these savings those due to savings in crucibles, fuel, and labor and it appears that, roughly speaking, under peacetime conditions the country will save annually two or three million dollars, perhaps more, by going completely over to electric brass melting. Under the conditions of low production and low metal prices prevailing in the latter part of 1920 and during 1921 the saving would be less, but over a term of years it would average at least the figure given. Whether any particular plant is so situated that it can save part of that sum for itself by using electric furnaces or whether it is one of the small minority so situated that electric melting would cost more than clinging to the older methods is a question to be answered only by careful consideration of the location of each plant and the conditions there. It is hoped that this report may aid in answering the question.

SAVINGS DUE TO MORE HEALTHFUL CONDITIONS.

Besides the readily recognizable saving in dollars and cents due to electric melting there is another saving less easily calculated, but no less real. The electric furnace is cooler, cleaner, and gives less zinc fume than older methods of melting. One rolling mill that has changed completely to electric melting has had no case of "brass shakes" since the fuel-fired furnaces were abandoned, and the number of severe burns has been greatly reduced. On account of the cooler and cleaner working conditions, the elimination of much hard manual labor, and the avoidance of zinc fume the electric furnace is no small factor in improving the comfort, health, and safety of the workmen. It was this point, no less than the possibilities of the electric brass furnace as a factor in the conservation of our natural resources, that caused the Bureau of Mines to undertake its investigation of melting brass in electric furnaces.

WORK OF THE BUREAU OF MINES.

The actual work of the Bureau of Mines on electric brass melting began in September, 1912, at a field office established at Morse Hall, Cornell University, Ithaca, N. Y., where, by the courtesy of the uni-

versity, the excellent facilities of the electric-furnace laboratory and other laboratories of the department of chemistry were, through a cooperative agreement, made available for the work of the bureau.

The work has been in three phases: (1) Collection of data, for comparison, on the performance of fuel-fired brass furnaces, which have been published as Bulletin 73¹⁴; (2) experimental work to show whether zinc losses could be reduced by electric melting, without regard to the question of whether or not the type of furnace used was likely to be commercially useful, which showed that the losses could be materially reduced, and finally (3) attempts to find out, through laboratory tests, inspection of commercial furnaces, attendance at commercial tests, and collection of data, what different types of electric furnaces have done or may be expected to do under various conditions.

The preliminary work on zinc losses was done mainly in crucible furnaces—first, because they were easier to handle on an experimental basis for the purpose in view, and, second, because it seemed desirable to start with an electrically heated furnace of a general type with which, in fuel-fired form, all foundrymen are familiar. Some of the preliminary work was reported¹⁵ at the 1914 meeting of the American Institute of Metals. The early work was confined mainly to brasses high in zinc, because the possibilities of metal saving were greatest for those alloys and because, at the prices then prevailing for coke, coal, fuel oil, metals, and crucibles, it was clear that the cost of electric power being higher than that of fuel made necessary the development of electric melting to a high degree of perfection before it could compete, on a cost basis, with fuel where the alloy melted was low in zinc and little saving of metal could be expected.

The outbreak of the European war, with the subsequent rise in cost of metals, fuel, and crucibles—the latter accompanied by a marked decrease in quality—altered conditions materially and gave experimenters greater incentive to carry on and complete their work. As the cost of crucibles had risen to unprecedented heights, the elimination of crucibles made possible so great a saving that electric melting in noncrucible furnaces became applicable to red brass and bronze, irrespective of the metal savings.

At the same time users of fuel-fired furnaces made every effort to limit the use of crucibles to alloys that absolutely required them. Crucibles were still generally used in melting yellow brass, but open-

¹⁴ Gillett, H. W., Brass-furnace practice in the United States; Bull. 73, Bureau of Mines, 1914, 298 pp.

¹⁵ Gillett, H. W., and Lohr, J. M., Melting losses in electric brass furnaces. This paper was preprinted and read, but as the data were to be included in a bulletin of the Bureau of Mines it was not printed in the transactions of the Institute. See Trans. Am. Inst. Metals, vol. 8, 1914, p. 11; Met. and Chem. Eng., vol. 12, 1914, p. 647.

flame oil furnaces very largely displaced crucibles for melting red brass and bronze, the determining factor being, of course, the zinc content of the alloy.

When the United States entered the war these conditions became even more altered from those prevailing in 1912 to 1914. Economy in materials, labor, and money outlay was not only commercially desirable, but became a patriotic duty. As a result during the war the development of electric brass melting received great impetus. The bureau's interest in the work increased as the problem from being mainly one of conserving zinc and promoting health and safety in foundries came to include the imperative need of producing munitions and essential war materials more cheaply and rapidly.

The bureau's work included the laboratory development and semi-commercial testing of the rocking type of furnace, which has been brought into commercial use by the Detroit Electric Furnace Co. that is licensed under Bureau of Mines patents to manufacture and sell it. The work on the rocking furnace has been related fully,¹⁶ and reference to that furnace in this report will be confined mainly to the performance of the commercial installation of this type made since the bureau's experiments were finished.

The Bureau of Mines, besides doing experimental work, has been privileged, through the cordial cooperation of inventors and makers of electric furnaces, of their users, and of central stations, to collect much data—by attendance at tests, by inspection of furnaces in commercial operation, as well as by correspondence—on the operation of most of the various types of electric brass furnaces now in use, on many of the discarded types, and on those still being developed. Some data already have been reported briefly.¹⁷

ACKNOWLEDGMENTS.

Grateful acknowledgment is made for his active interest and support to Dr. C. L. Parsons, formerly chief chemist of the Bureau of Mines to whose initiative the bureau's work on this problem is due, and under whose direction it was mainly carried out; and to Dr. J. M. Lohr and Mr. A. E. Rhoads, formerly of the bureau, for efficient aid in experimental work.

Thanks are due Cornell University for the use, under cooperative agreement, of its electric furnace laboratory and other equipment.

¹⁶ Gillett, H. W., and Rhoads, A. E., Melting brass in a rocking electric furnace: Bull. 171, Bureau of Mines, 1918, 131 pp.; A rocking electric brass furnace: Jour. Ind. Eng. Chem., vol. 10, 1918, p. 459; Met. and Chem. Eng., vol. 18, 1918, p. 583.

¹⁷ Gillett, H. W., Utilization of electric brass furnaces: Jour. Ind. Eng. Chem., vol. 11, 1919, p. 664; Chem. and Met. Eng., vol. 21, 1919, pp. 6, 116; Electrical melting of alloys (serial): Foundry, vol. 48, 1920, pp. 177, 229, 275, 319, 362, 400, 445, 486, 531, 571, 612, 656, 693; Electric furnaces for nonferrous alloys: Trans. Am. Electrochem. Soc., vol. 39, 1921, p. 277.

utilized in the experimental work, as well as to Professors W. D. Bancroft and T. R. Briggs, in charge of electrochemical work at Cornell University, for many courtesies.

So many makers and users of electric brass furnaces have cooperated by furnishing data and information that a complete acknowledgment would list practically everyone interested in the problem. The bureau's thanks are due to all these firms and individuals, and particularly to those who permitted representatives of the bureau to watch their furnaces in operation. Special mention should be made of the constructive cooperation of the Detroit Edison Co., through Mr. E. L. Crosby, and of the Commonwealth Edison Co., Chicago, through Mr. H. M. St. John.

DIFFERENT TYPES OF ELECTRIC BRASS FURNACES.

Not less than 80 different types or different makes of the same type of furnaces have been used, tried, or suggested for melting copper, brass or bronze, aluminum, or nickel alloys. These are listed in Table 2 in the order in which they are described in the text. Mention is made in the text of types as well suited to melting these materials, as are many of those in the list, many of which are purely "paper furnaces" that probably never existed except in patent specifications. If the patent specifications, however, definitely state that the furnace is designed for melting brass instead of merely mentioning the melting of metals in general, it has been included. If one furnace made by the same person or firm differs from another only in size and not in type or design, but one entry is made. On the other hand, the same type or design as made by different persons or firms is given a separate entry for each make. If it were possible to include all the types of furnaces on which no information is available and that have been tried out and abandoned by various workers, the list would be increased materially—probably to at least a hundred.

TABLE 2.—*Electric furnaces designed or tested for melting nonferrous metals.*

CRUCIBLE FURNACES.

No.	Name of designer or builder.	Used or tested by or at—	Type.	Distinguishing features.	Tests of use on nonferrous alloys.		
					Has had laboratory test.	Has had foundry test.	Present commercial use in United States, Jan. 1, 1922.
1	Fitzgerald.....	Titanium Alloys Manufacturing Co.	Crucible lift-out.	Granular resistor.	Yes.....	No.....	None.
2	Baily.....	W. A. Rogers (Ltd.).	Crucible lift-out, 2 pots.	do.....			One furnace melting silver.
3	do.....	Detroit Copper & Brass Rolling Mills.	Crucible lift-out, 8 pots.	do.....		Yes.....	None.
4	Snyder.....	Commonwealth Edison Co. English test.	Crucible lift-out.	Solid molded resistor.		Yes.....	None.
5	Greenwood & Hutton.	do.....	do.....	Solid grid resistor.	Yes.....	No.....	None.
6	Hoskins (Marsh).	Commonwealth Edison Co. and Bureau of Mines.	do.....	Carbon-plate resistor.	Yes, 16 sizes and modifications.	Yes.....	One or two furnaces melting nickel-chromium alloys.
7	Bureau of Mines.	Bureau of Mines.	Crucible lift-out, 3 pots.	Carbon-plate resistor between pots.	Yes.....	No.....	None.
8	Cadwell.....		Crucible lift-out, 2 pots.	Granular resistor below arcs at side of pots.			None.
9	Rennerfelt reverberatory crucible.		Crucible lift-out, several pots.	Arc resistance.			None.
10	General Electric Co.	General Electric Co.	Crucible tilting.	do.....	Yes.....	?.....	None.
11	Stansfield.....		do.....	Granular resistor surrounding crucible.	?.....		None.
12	Baily.....		do.....	do.....	Yes.....		None.
13	Bureau of Mines.	Bureau of Mines.	do.....	do.....	Yes.....		None.
14	National Carbon Co.		do.....	do.....			None.
15	General Electric Co.		do.....	do.....			None.
16	Baily.....	Detroit Copper & Brass Rolling Mills.	do.....	Granular resistor below crucible.		Yes.....	None.
17	Weintraub.....	General Electric Co.	do.....	Solid silicon-grid resistor.		Yes.....	None.
18	Helberger.....	National Cash Register Co. English test.	do.....	Solid resistor.		Yes.....	None.
19	Electro-Metals Co. (Ltd.).		do.....	do.....	Yes.....	No.....	None.
20	Morgan.....	do.....	do.....	do.....	?.....	?.....	Advertised for sale, but probably not used.
21	Conley.....	Electrical Testing Laboratory.	do.....	do.....	Yes.....	No.....	None.

TABLE 2.—*Electric furnaces designed or tested for melting nonferrous metal—Continued.*

HEARTH FURNACES.

No.	Name of designer or builder.	Used or tested by or at—	Type.	Distinguishing features.	Tests of use on nonferrous alloys.		
					Has had laboratory test.	Has had foundry test.	Present commercial use in United States, Jan. 1, 1922.
22	Rockwell.....		Resistor below hearth.		Yes.....	No.....	None.
23	Bristol.....		Carbon-rod resistor.	Steam.....	?.....	No.....	None.
24	Clingman.....		do.....		?.....	?.....	None.
25	Johnson.....		Lattice resistor.	Pump for stirring.	No.....	No.....	None.
26	Fitzgerald-Thomson.		Resistor layer of rods.		Yes.....	No.....	None.
27	do.....		Self-supporting resistor.		Yes.....	No.....	None.
28	Thomson.....		Zigzag resistor.	Gyrating.....	?.....	?.....	None.
29	do.....		Tubular zigzag resistor.	Rotating.....	?.....	?.....	None.
30	Bensel.....	German test..	Carbon-rod resistor.	Rocking.....	Yes.....	No.....	None.
31	Harvey.....		Resistor.....	Rotating.....	?.....	No.....	None.
32	Hoskins.....	Bureau of Mines.	Carbon-plate resistor overhead.		Yes.....	No.....	None.
33	Thornton.....		Carborundum resistor.		?.....	No.....	None.
34	Bureau of Mines.	Bureau of Mines.	Carborundum resistor between two hearths.		Yes.....	No.....	None.
35	Weindenthal..		Carborundum resistors at sides.		?.....	No.....	None.
36	Erichsen.....		Carborundum resistors overhead.		?.....	No.....	None.
37	Dirzuweit....	Baltimore Tube Co.	Granular resistor in troughs.			Yes.....	None.
38	Baily.....	Many users..	Granular resistor in circular trough.			Yes.....	Wide use, 82 furnaces.
39	do.....	Several trials.	Granular resistor in straight troughs at sides, large tapping furnace.		?.....	Yes, on aluminum and zinc.	None. Successfully used in annealing furnaces.
40	do.....	do.....	Granular resistor in straight trough over hearth.		?.....	Yes.....	None.
41	General Electric.	Several users..	Arces and resistance, rectangular.		Yes.....	Yes.....	9 furnaces of both types.
42	do.....	General Electric Co.	Arces and resistance, round.		Yes.....	Yes.....	
43	Rennerfelt reverberatory.	Chicago Faucet Co.	Arces and resistance.	Heat source between 2 hearths.	Yes.....	Yes.....	One.
44	Heroult type..	Several users..	Direct arc...		Yes.....	Yes.....	2 on nickel alloys, none on brass or bronze.

TABLE 2.—*Electric furnaces designed or tested for melting nonferrous metal—Continued.*

HEARTH FURNACES—Continued.

No.	Name of designer or builder.	Used or tested by or at—	Type.	Distinguishing features.	Tests of use on nonferrous alloys.		
					Has had laboratory test.	Has had foundry test.	Present commercial use in United States, Jan. 1, 1922.
45	Snyder.....	Several users..	Direct arc...		Yes.....	Yes.....	6 on nickel or cobalt alloys, 2 on leaded bronze, 1 on nickel alloys.
46	Greaves-Etchells.	Hoskins Manufacturing Co.	do.....			Yes.....	
47	Bureau of Mines.	Bureau of Mines.	do.....		Yes.....	No.....	None.
48	Wile.....	Several trials..	Slag resistance.	Glass slag...	Yes.....	Yes.....	None.
49	Bennett.....	Scovill Manufacturing Co.	Contact resistance.	Low voltage.	Yes.....	Yes.....	7 large furnaces.
50	De Nolly-Grammont.	French test...	Direct arc...	Large electrodes.		Yes.....	None in United States.
51	Von Schlegel-Fletcher.		Direct arc, then indirect arc.		?.....	No.....	None.
52	Moldenke.....		Indirect arc.	Charge preheated by fuel.	No.....	No.....	None.
53	Weeks.....	Weeks and Hansen.	do.....		Yes, on copper only.	No.....	None.
54	Shipton.....		do.....	Shelf over arc.	No.....	No.....	None.
55	Indianapolis...	(?)	do.....		Yes.....	(?)	None.
56	Stassano type.	General Electric Co.	do.....		Yes.....	(?)	None.
57	Stassano-Pettinot.	(?)	do.....		(?)	(?)	None.
58	Bureau of Mines.	Bureau of Mines.	do.....		Yes.....	No.....	None.
59	Bennerfelt.....	Several users..	do.....		Yes.....	Yes.....	8 furnaces.
60	Repel-arc (von Schlegel).	do.....	do.....	Repelling arc.	Yes.....	Yes.....	6 furnaces.
61	Bureau of Mines.	Bureau of Mines and Michigan Smelting and Refining Co.	do.....	Rocking....	Yes.....	Yes.....	See No. 62.
62	Detroit.....	Many users..	do.....	do.....		Yes.....	Wide use 60 furnaces, mostly large.
63	Universal.....	Jos. Brenner & Co.	do.....	do.....		Yes.....	None.
64	Tooth.....	Many users..	do.....	Rotating...		Yes.....	Considerable use, 40 furnaces, mostly small.
65	American.....		do.....	Revolving...	(?)	No.....	None.
66	Moore.....	(?)	do.....	do.....	(?)	None.....	None.
67	Reardon.....		do.....	Rocking....		No.....	None.
68	Volta.....	Titanium Bronze Co.	do.....	Gyrating...	Yes.....	Yes.....	None.
69	Colby-Kjellin type.	Several trials..	Induction...	Horizontal ring.	Yes.....	No.....	None except 1 on nickel alloys.
70	Spencer.....	Pierce Arrow Motor Car Co.	do.....	do.....	Yes.....	No.....	None.
71	Kjellin.....	English test...	do.....	do.....	Yes.....	No.....	None.

TABLE 2.—*Electric furnaces designed or tested for melting nonferrous metal—Continued.*

HEARTH FURNACES—Continued.

No.	Name of designer or builder.	Used or tested by or at—	Type.	Distinguishing features.	Tests of use on nonferrous alloys.		
					Has had laboratory test.	Has had foundry test.	Present commercial use in United States, Jan. 1, 1922.
72	Reechling-Rodenhauser.	Baltimore Copper Smelting and Rolling Co.	Induction...	Horizontal ring.		Yes.....	None.
73	Harden.....		do.....	Horizontal ring with modifications.	(?)	No.....	None.
74	Gehrkins.....		do.....	do.....	(?)	(?)	None.
75	Greene.....		do.....	do.....	(?)	No.....	None.
76	Howard.....		do.....	do.....	(?)	No.....	None.
77	Hering.....	Several trials.	Pinch effect.	Resistor tubes.	Yes.....	Yes.....	None.
78	Foley.....	Bristol Brass Co.	Induction...	Vertical ring.	Yes.....	Yes.....	None.
79	Brophy.....		do.....	Vertical ring with modifications.	(?)	No.....	None.
80	Ajax-Wyatt...	Many users...	do.....	Vertical ring.	Yes.....	Yes.....	98 furnaces
81	Clark.....		do.....	Vertical ring with modifications.	(?)	No.....	None.
82	Ajax-Northrup	Handy and Harmon, etc.	Eddy current, tilting.	High frequency current.	Yes.....	Yes.....	9 on silver, gold, and platinum.
83	General Electric.	General Electric Co.	Nickel-chromium resistor.	Special for aluminum.	Yes.....		None.

Note. Another electrical brass furnace, made by Wm. Swindell & Bros., Pittsburgh, Pa., was advertised as ready for commercial use late in 1921, but no description is yet available.

The same idea held by different people or different ideas held by the same person have produced 80 or more forms of electric brass furnaces, of which at least 50 have been actually tested in melting copper, brass, or bronze. Some of these never completed the first heat, whereas others in commercial use by scores of firms have melted huge tonnages of metal.

The surviving types of furnaces, therefore, represent the fruit of a vast amount of thought, of experiment, and of commercial elimination of the unfit. This process of development and elimination will continue, and the perfect electric brass furnace, though it may be evolved in the future, is still nonexistent. Crude as all types may seem in the light of future developments, there are several types that are of present commercial value and are ready for foundry use.

OBJECT OF THIS REPORT.

The object of this report is twofold: First, to collect all necessary and pertinent information available on what commercial furnaces are

doing and can do, in order that the prospective user may have data for forming a decision as to whether to install any electric furnace and for selecting a type of furnace fitted to his requirements; second, to show, by description and by data on experimental tests, the gradual evolution of the present commercial types, and by including information on failures as well as successes to save future workers in this field the labor of following the steps of previous workers.

Another object is to point out some of the fundamental principles in the design, construction, and operation of electric brass furnaces. Part of the information in this report has been published before, but it was written chiefly from one viewpoint or in connection with some particular furnace. This report considers the subject broadly.

THEORETICAL ADVANTAGES OF ELECTRIC MELTING.

The study of the various furnaces will be facilitated by first considering the theoretical advantages of electric brass furnaces. These advantages are as follows:

1. Melting may take place in a neutral or reducing atmosphere, thus minimizing loss of metal by oxidation and improving the quality of the product through freedom from oxides.

2. Metal of crucible quality may be obtained without the use of crucibles.

3. Melting may take place in a tightly closed chamber, or at least in one free from the constant passage of the products of combustion of fuel, and thus losses of volatile metals such as zinc and lead may be reduced. Contamination by sulphur from fuel is avoided.

4. In some types of electric furnaces the temperature may be more readily controlled than in fuel-fired furnaces.

5. In some types of furnaces the molten metal is thoroughly stirred, thus giving a uniform product, even with large heats.

6. There is no handling or storage of fuel, such as coke, coal, or oil, and no ashes have to be removed. The cost of power can be accurately predicted over longer periods than the cost of fuel.

7. Working conditions about the furnaces are less dangerous to health and safety of workmen, provided suitable types of furnaces are chosen.

8. The above advantages may be obtained in furnaces of larger capacity than can be used satisfactorily in the fuel-fired crucible types, with resulting greater uniformity of product, lower labor cost, and increased production.

There is no occult or mysterious effect on the metal from its having been "under the influence of electricity." Whatever improvement in quality results is largely due to the fact that electric heat is "clean" and permits the avoidance of oxidation, etc. If one could find a brick that would absorb enough heat to do the work, heat it, drop it into a crucible charged with brass, and cover the crucible,

thus melting the metal by the heat stored in the brick, he would melt the brass under about the same conditions as with an electrically heated crucible. The action of an electric furnace is no more mysterious than that of the hypothetical brick. The electric current is used merely as a means of producing heat, and no subtle alchemistic influence is exerted on the metal.

The general opinion of users of electric brass furnaces seems to be that the average product is no better and no worse than the metal from a well-operated fuel-fired crucible and that, owing to melting in larger units and to the use of furnaces that mix the charge thoroughly, the product is, in general, more uniform. On account of the uniformity of electrically melted metal the Bridgeport Brass Co. declares that electric furnaces make better condenser tubing and permit closer working to specification as to analysis. As the uniformity from perfect mixing in the electric furnaces used by this firm would naturally decrease the danger of electrolytic attack, the claim of superior quality seems justified, though it is probably a superior average quality rather than an improvement over the best crucible product. Claims are sometimes made¹⁸ that the physical properties of electrically melted cast brass are slightly superior to those of brass melted by fuel, but the advantage claimed is so slight that it might as readily be due to such factors as differences in pouring temperatures as to actually improved quality of the metal.

At any rate, it is not evident that anyone is relying enough on the added strength of electrically melted metal to have the courage to cut down the cross section and weight of castings made from it.

The basic electric steel furnace can take cheap, low-quality scrap and, by the use of refining slags, can make a high-grade product. Usually no analogous refining of nonferrous metals can be done in electric brass furnaces, though the ability of the electric furnace to handle borings and turnings with less metal loss than crucibles may sometimes permit the use of cheaper materials by allowing an increase in the proportion of borings used. There seems, however, to be a general impression that electric steel of a given analysis is better than even crucible steel of the same analysis, though this point is still open to discussion. A superiority similar to that alleged for electric steel may exist for electric brass, but its definite proof is even farther from being complete. In fact, there have been cases where electrically melted metal has not seemed to be as satisfactory as that from crucibles, though in other very similar circumstances it has been found to be just as good.

Various other alleged reasons for the superiority of electric brass furnaces have been brought up by furnace makers, such as the

¹⁸ Compare Collins, E. F., *Electric furnace for melting nonferrous metals*: Jour. Cleveland Eng. Soc., vol. 11, 1919, p. 293.

possibility of carrying the furnace to the mold and pouring from it direct, which might be practical in some cases, but is of extremely slight importance in any discussion of the general advantages of electric brass melting.

PRACTICAL ADVANTAGES OF ELECTRIC BRASS FURNACES.

The real reason for the rapidly growing use of electric brass furnaces is a highly practical one. Such furnaces properly chosen for the work in hand, in a large majority of cases, can produce metal of satisfactory quality at a lower total melting cost than fuel-fired furnaces. They save metal and labor, they increase production, they eliminate crucibles. The value of these factors overbalances the interest charge on the high first cost of the electric furnace and the higher cost, compared with fuel, under most conditions, of the electric current required.

Not every foundry is so situated that electric melting is cheaper than melting by fuel, and neither electric furnaces in general, nor, still less, any particular type or make of electric furnace, can be advocated as a panacea for all nonferrous melting problems. A careful consideration of plant conditions and of the performance of all types of electric furnaces operating under similar conditions, as far as such operation can be found, is essential before deciding to change from fuel-fired to electric furnaces.

When such a careful consideration of conditions is made, however, it is evident that, taking the brass industry as a whole, the economic conditions so strongly favor electric melting that it is probably only a matter of time, and not a very long time, before such melting will become standard practice for brass and bronze. Already the use of electric furnaces is growing at a more rapid rate in the brass industry than during the analogous period in the history of the steel industry, and the ratio of the tonnage of electric brass to the total tonnage is already much greater than that of electric steel to the total output of steel. There are almost, if not quite, as many electric furnaces installed for melting nonferrous alloys in the United States to-day as there are for electric steel.

DIFFERENCES BETWEEN BRASS MELTING AND STEEL MELTING.

Consideration of theoretical advantages and of the high state of perfection that the electric furnace has attained for melting steel might give the impression that the application of the electric steel furnace to the melting of nonferrous alloys is a simple matter, and that because of the lower melting points of brasses and bronzes their melting should be simpler than that of steel. If certain physical properties of molten steel and of molten brass be considered, however, it will be seen that steel and brass offer two distinct sets of conditions.

TYPES OF STEEL FURNACES.

According to Miller,^{18a} on January 1, 1917, out of 155 electric steel furnaces in the United States and out of 471 in the world the proportions of the different types were as follows:

Proportions of different types of electric steel furnaces in the United States and in the world.

	Direct arc.	Indirect arc.	Induc- tion.
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
World.....	69½	20	10½
United States.....	89	9	2

Out of some 400 electric steel furnaces in the United States and Canada¹⁹ only four, or 1 per cent, are induction furnaces. About 5 per cent are indirect arc, the other 94 per cent being direct-arc furnaces. Out of some 150 furnaces in England only four are induction furnaces.

DRAWBACKS OF STEEL FURNACES FOR BRASS MELTING.

Induction furnaces of the ordinary horizontal-ring type will not melt brass, bronze, or similar copper alloys; their electrical resistance is so low that in order to introduce enough power for melting the current must be so high that the "pinch force"²⁰ comes into play and the conducting ring of metal no longer stays in place in the furnace. Hence the only types of steel furnaces that might be adapted to brass without material change are arc furnaces.

But the temperature of the carbon arc is about 3,500° C.,²¹ and the boiling points of various pure metals are given as follows:

Boiling points of various pure metals.

Metal.	Boiling point.			Metal.	Boiling point.		
	°C.	°C.			°C.	°C.	
Iron.....	a 2,450	b 2,450		Lead.....	1,525	1,640	1,555
Copper.....	2,310	2,350	c 2,305	Antimony.....	1,440	1,440	1,380
Tin.....	2,270	2,260	2,270	Zinc.....	d 920	920	930
Manganese.....	1,900	1,900					

^a Greenwood, H. C., The boiling points of metals: Trans. Faraday Soc., vol. 7, pt. 2, 1911, p. 151.

^b Johnston, J., The vapor pressure and volatility of several high-boiling metals: Jour. Ind. Eng. Chem., vol. 9, 1917, p. 876; Watts, O. P., The metals in order of their boiling points: Trans. Am. Electrochem. Soc., vol. 12, 1907, p. 141.

^c Ruff, O. and Bergdall, B., Researches at high temperatures: Ztschr. anorg. Chem., vol. 108, 1919, p. 16; Chem. Abstr. vol. 13, 1919, pp. 304, 3054.

^d Burgess, G. K., work cited, p. 438; compare Dewey, F. P., Volatilization in assaying: Min. and Met., No. 158, sec. 31, Feb., 1920; Met. and Chem. Eng., vol. 22, 1920, p. 797.

^{18a} Miller, D. D., Industrial heating as a central-station load: Supplement Society for Electrical Development, 1917.

¹⁹ Cone, E. F., The status of the electric steel industry: Iron Age, vol. 107, 1921, p. 69.

²⁰ Hering, C., The pinch phenomenon: Trans. Am. Electrochem. Soc., vol. 11, 1907, p. 329.

²¹ Burgess, G. K., Measurement of high temperatures: 1912, p. 454.

In a direct-arc steel furnace the arc is in contact with metal or with a comparatively thin layer of slag. Probably there is very little volatilization of iron in such a steel furnace, as the conduction of heat through the molten metal probably suffices to keep all save a very small portion of the bath below the boiling point; nevertheless, when ferromanganese is remelted in a furnace of the direct-arc type there is a high loss of manganese, unless the voltage of the furnace is so low that it runs more as a resistance furnace than a true arc furnace.²² Hansen²³ found a notable volatilization of copper in a direct-arc steel furnace for melting pure copper, and similar results have been reported by other observers; but lower-melting copper alloys have been melted in direct-arc furnaces with no trouble from copper vapor.

The approximate boiling points of copper-zinc alloys are shown in figure 1.²⁴ It seems probable that if a direct-arc steel furnace were used for melting a true copper-tin bronze there would be little loss by volatilization, because the boiling points of both metals are high. For alloys containing much lead or antimony some loss might be expected, and for alloys of copper with much zinc, such as yellow brass, the loss would be decided.

This is brought out still more clearly by Johnston,²⁵ who shows that the partial pressure of zinc vapor at 1,200° C. is about two and one-half atmospheres for 60:40 brass, two for 65:35, and one and one-half for 70:30.

For nickel or cobalt alloys not containing zinc, such as Monel, nichrome, chromel, and Stellite, the types of furnaces used for steel—direct arc, indirect arc, and horizontal-ring induction—are suitable, and all these types are used. Nickel brass, or German silver, however, more closely resembles brass and is not melted in the steel-making types.

Indirect-arc furnaces, in which the arc is struck over the metal, but in fairly close proximity to the bath, should show less loss by volatilization than direct-arc furnaces. However, as hot molten metal is lighter than colder molten metal, and as the arc must be above the bath, there is a possibility of the production of a superheated surface layer of metal on top of the bath, directly under the arc, so that the boiling point of the alloy may be exceeded in this layer and volatilization losses ensue.

²² Rodenhauser, W., *Ferromangan als Desoxydationsmittel*, 1915, p. 86.

²³ Hansen, C. A., Copper poisoning: *Met. Chem. Eng.*, vol. 9, 1911, p. 67; *Electric melting of copper and brass*: *Trans. Am. Inst. Metals*, vol. 6, 1912, pp. 122-125.

²⁴ Reprinted from Bull. 73, Bureau of Mines, figure 2, p. 129.

²⁵ Johnston, J., The volatility of the constituents of brass: *Jour. Am. Inst. Metals*, vol. 12, 1918, p. 20.

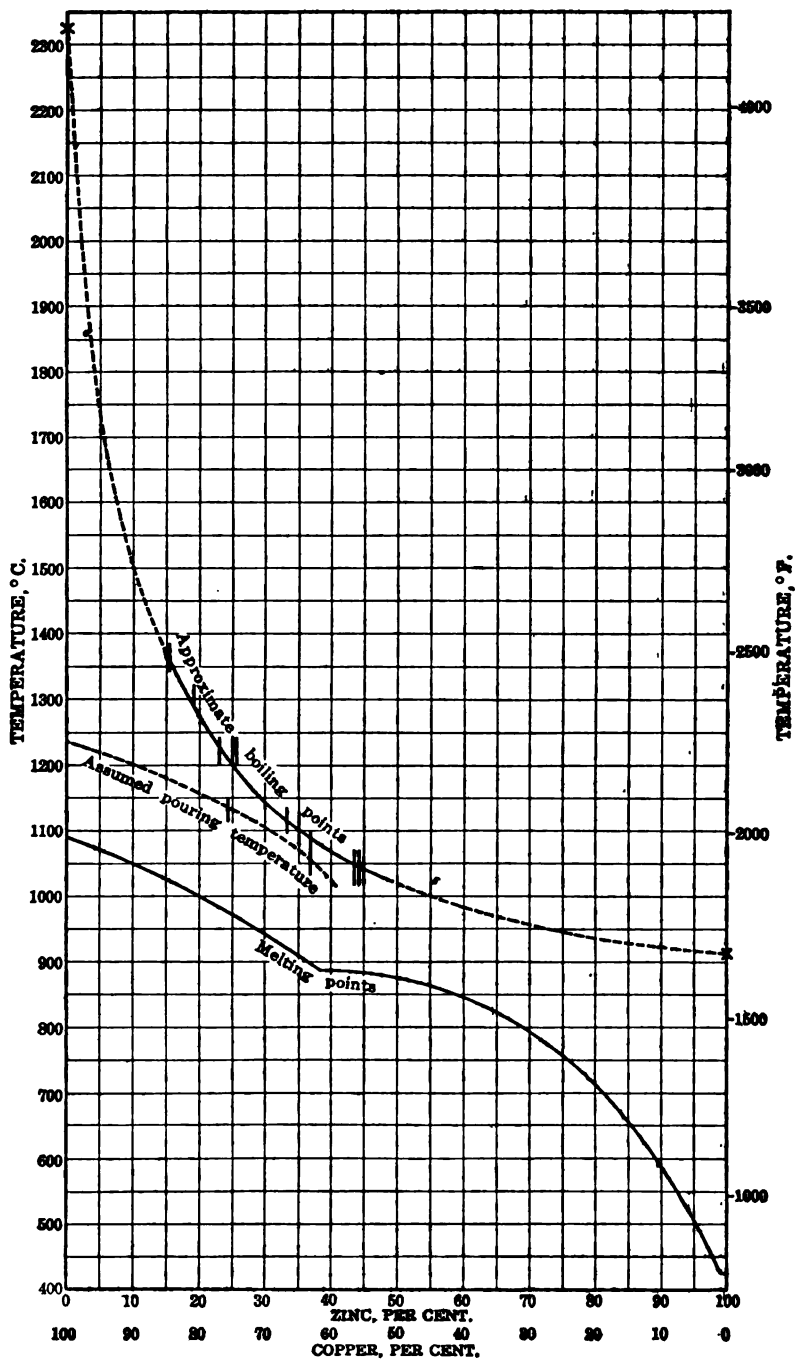


FIGURE 1.—Approximate boiling points and assumed pouring temperatures of copper-zinc alloys.

It will be shown later that the applicability of the direct-arc and indirect-arc types of steel-melting furnaces to brass is limited for alloys with volatile constituents—just as the theory would indicate—and that to extend electric melting to such alloys it has been necessary to develop types of furnaces radically different from those in use for steel melting.

TYPES OF COMMERCIAL ELECTRIC MELTING FURNACES FOR STEEL.

All electric furnaces generate heat by the flow of current through a resistor, which may be solid, liquid, or gaseous. The gaseous-resistance furnace is, however, an arc furnace, which is the first subdivision usually made in the classification of electric furnaces.

If the arcs are drawn from carbon electrodes directly to the material to be heated, the furnace is a direct-arc furnace. If the arcs are drawn to a slag or to a material that has a relatively high resistance, the direct-arc furnace may become a combination arc-resistance furnace. In general, the arc heating so overbalances the resistance heating that the latter is negligible.²⁸

On the other hand, if the arc voltage is very low the resistance of the bath may be considerable in comparison with that of the arc. In the extreme case the voltage may be so low that there is no true arc, but merely a sort of contact resistance between the electrodes and the bath.

In practically all the arc, arc-resistance, and contact-resistance furnaces at least one electrode hangs vertically and its lower end is in close proximity to the bath.

Direct-arc furnaces may be (a) single phase with one top electrode and one bottom electrode making direct contact with the bath (Siemens,²⁷ Snyder,²⁸ Massip^{28a}); (b) single phase with two top electrodes (Heroult²⁹); (c) polyphase with two or more top electrodes and one bottom electrode, which may be in one piece or in several (Girod,³⁰ Mathusius,³¹ Snyder,³² Gronwall-Dixon,³³ Greaves-Etch-

²⁸ Rodenhauser, W., and Schoenawa, J., translated by vom Baur, C. H. [Electric furnaces in the iron and steel industries], 1917, p. 80.

²⁷ Siemens, W., English Patent 2,110 of 1879.

²⁸ Snyder, F. T., U. S. Patent 1,167,026, Jan. 4, 1916.

^{28a} Massip, G., U. S. Patent 1,080,840, Dec. 9, 1913.

²⁹ Heroult, P. L. T., U. S. Patent 707,776, Aug. 26, 1902.

³⁰ Girod, P., U. S. Patent 921,228, May 11, 1909; Buck, C. C., The Bethlehem 10-ton Girod steel furnace: Trans. Am. Electrochem. Soc., vol. 31, 1917, p. 81.

³¹ See Rodenhauser, W., Ferromangan als Desoxydationsmittel, 1915, p. 50.

³² Industrial Electric Furnace Co., Electric Furnace Bulletin, October, 1917; Snyder, F. T., U. S. Patent 1,327,174, Jan. 6, 1920.

³³ Crowley, J. A., The Gronwall-Dixon electric furnace: Foundry, vol. 44, 1916, p. 497; Dixon, J. L., U. S. Patent 1,198,626, Sept. 19, 1916.

ells,³⁴ Greene,³⁵ Booth-Hall,³⁶ Moore,³⁷ Webb,³⁸ Keller³⁹); (d) poly-phase, three or more top electrodes, no bottom electrode (Heriout,⁴⁰ Burke,⁴¹ Ludlum,⁴² vom Baur,⁴³ Price-Dixon,⁴⁴ Snyder,⁴⁵ Masera,⁴⁶ and Dryssen⁴⁷).

In all types with a bottom electrode that electrode may be of conducting material, such as carbon or metal, or may be of a second-class resistor; that is, one that conducts only when hot, as magnesite.

Of arc-resistance furnaces, or resistance furnaces structurally similar to direct-arc furnaces, the examples are fewer. They include the Wile, shaft,⁴⁸ and tilting,⁴⁹ and the Schemman and Bronn.⁵⁰ Of indirect-arc furnaces, with the arc struck between carbon electrodes over the metal bath, the commercial examples are the Rennerfelt,⁵¹ two-phase, and Stassano,⁵² three-phase.

A résumé of present electric steel practice is given by Diller.⁵³

The various induction furnaces that have had commercial use (Colby, Kjellin, Frick, Roehling-Rodenhauser) are described by Rodenhauser and Schoenawa⁵⁴ and by vom Baur.⁵⁵

³⁴ See Rodenhauser, W., and Schoenawa, J., translated by vom Baur, C. H., work cited, p. 26.

³⁵ Rolling cylinder electric steel furnace: *Foundry*, vol. 44, 1918, p. 339.

³⁶ Booth, W. K., The Booth-Hall electric furnace: *Trans. Am. Electrochem. Soc.*, vol. 33, 1918, p. 95.

³⁷ Acheson Graphite Co., Electric furnaces made in the United States, September, 1919, pp. 10, 14; Moore, W. E., U. S. Patent 1,304,350, May 20, 1919.

³⁸ Industrial Electric Furnace Co., Electric Furnace Bulletin No. 40, 1917, p. 10.

³⁹ Stansfield, A., The electric furnace 1914 ed. p. 228.

⁴⁰ See Rodenhauser, W., and Schoenawa, J., translated by vom Baur, work cited.

⁴¹ Burke, J., U. S. Patent 1,082,459, Dec. 23, 1913.

⁴² Sawhill, R. V., Melting all scrap in the electric furnace: *Foundry*, vol. 45, 1917, p. 339.

⁴³ vom Baur, C. H., The vom Baur arc furnace: *Trans. Am. Electrochem. Soc.*, vol. 33, 1918, p. 137; U. S. Patents 1,252,633, Jan. 8, 1918, and 1,385,411, July 26, 1921.

⁴⁴ Acheson-Graphite Co., Electric furnaces made in the United States, p. 11.

⁴⁵ Industrial Electric Furnace Co., advertisement: *Foundry*, vol. 48, 1920, Feb. 15, p. 61.

⁴⁶ Masera, C., U. S. Patent 1,320,884, Nov. 4, 1919.

⁴⁷ Dryssen, W., U. S. Patent 1,317,911, Oct. 7, 1919.

⁴⁸ Wile, R. S., U. S. Patent 947,723, Jan. 25, 1919.

⁴⁹ Wile, R. S., U. S. Patent 1,070,568, Aug. 19, 1913.

⁵⁰ Schemmann, W., and Bronn, J., U. S. Patent 1,056,456, Mar. 18, 1913; see also Rodenhauser, W., Ferromangan als Desoxydationsmittel. 1915, p. 26.

⁵¹ Rennerfelt, I., U. S. Patent 1,076,518, Oct. 21, 1913; vom Baur, C. H., The Rennerfelt electric-arc furnace: *Trans. Am. Electrochem. Soc.*, vol. 29, 1916, p. 497; Rennerfelt electric-furnace operation: *Trans. Am. Electrochem. Soc.*, vol. 31, 1917, p. 87; Rodenhauser, W., and Schoenawa, J., translated by vom Baur, C. H., Electric furnaces in iron and steel industry. 1917. p. 161.

⁵² Stassano, E., U. S. Patents 799,105, Sept. 12, 1905; 1,024,657, Apr. 30, 1912; 1,059,490, Apr. 22, 1913; and 1,105,859, Aug. 4, 1914; see also Rodenhauser, W., and Schoenawa, J., work cited, p. 106.

⁵³ Diller H. E., Pointers on electric steel furnace practice: *Foundry*, vol. 47, 1919, p. 239; see also Seede, J. A., Electric furnaces for production of steel and ferro alloys: *Gen. Elec. Rev.*, vol. 21, 1918, p. 767; Moore, W. E., Electric furnaces for steel foundries: *Elec. Jour.*, vol. 16, 1919, p. 360; Lindemuth, L. B., The position of the electric furnace in iron and steel metallurgy: *Trans. Am. Electrochem. Soc.*, vol. 37, 1920, p. 611.

⁵⁴ Rodenhauser, W., and Schoenawa, J., work cited, p. 173 ff.

⁵⁵ Vom Baur, C. H., Electric induction furnaces for steel: *Trans. Am. Electrochem. Soc.*, vol. 22, 1912, p. 117.

A recent modification made by the General Electric Co. is described by Unger and Scharschu.⁶⁶

Of all these various types and makes of steel furnaces only two—the Snyder and the Rennerfelt—have had appreciable commercial application to brass or bronze, and the use of these two is strictly limited to alloys low in zinc. Other types and makes are used in melting nickel alloys.

TYPES OF BRASS FURNACES.

A great variety of designs, many of them markedly different from steel-furnace designs, have therefore been tried or suggested for brass and bronze, as has been shown in Table 2.

Electric brass furnaces should be considered first as brass furnaces and second as electric furnaces; that is, the user is concerned with quality, rate of production, and cost of production rather than with kilowatt hours, power factor, load factor, volts, and amperes.

The electrical characteristics of the furnace, however, affect its reliability, its efficiency, and thus its operating cost, as well as its desirability as a load for the power plant or the central station, and therefore should be considered. For this reason electric brass furnaces have to be classified not only in regard to their behavior as brass furnaces but also in regard to the means by which they generate heat from the electric current. They will be discussed under the following heads:

- A. Crucible furnaces (with few exceptions, of the carbon resistance type).
 - a. Lift-out.
 - b. Tilting.
- B. Hearth or noncrucible furnaces.
 - a. Carbon resistance, radiated-heat type.
 - b. Arc furnaces.
 - I. Smothered arc.
 - II. Direct-arc type.
 - a. Stationary furnace.
 - 1. Open arc.
 - 2. Arc to slag or slag resistance.
 - 3. Low-voltage arc or contact resistance.
 - β. Moving furnaces.
 - III. Indirect-arc type.
 - a. Stationary furnaces.
 - ε. Moving furnaces.

⁶⁶ Unger, M., and Scharschu, C. A., New type of induction electric furnace: *Iron Age*, vol. 108, 1921, p. 344.

c. Internal metallic resistance furnaces.

α. Horizontal-ring induction.

β. Pinch-force resistor.

γ. Vertical-ring induction.

δ. High-frequency, eddy-current induction.

OPERATION OF ELECTRIC BRASS FURNACES DOES NOT REQUIRE AN ELECTRICAL ENGINEER.

Although constant reference has to be made to electrical terms in describing electric brass furnaces, and these terms are defined in this report, there should be no misconception as to the amount of electrical knowledge necessary for the operation of electric brass furnaces.

One does not have to be an electrical engineer to understand the operation of an electric furnace any more than one has to be an automotive engineer to understand the operation of an automobile. Several types of commercial electric brass furnaces have been operated regularly and satisfactorily by negro crews. Even the superintendent of a brass foundry using electric furnaces need concern himself but slightly with the purely electrical side of the installation. An electric-furnace load is, generally speaking, so desirable to a central electric generating station that the station, if consulted in the choice and during the installation of the furnaces, will gladly supply all needed electrical advice. Central stations on the lookout for desirable loads have contributed much to the development of the commercial use of electric brass furnaces.

If the electric furnace is built and installed by a furnace maker who has the proper combination of electrical engineering knowledge, electric furnace experience, and acquaintanceship with brass melting and brass foundry problems—the last factor is by no means the least important—he should be able, during the time he spends in teaching the foundry force to run the furnace, to impart to them all the electrical knowledge necessary. With the engineers of the local central station⁵⁷ eager to assist in the solution of problems that might arise

⁵⁷ Compare St. John, H. M., Commercial testing of metallurgical electric furnaces: Chem. and Met. Eng., vol. 21, 1919, p. 377; The present status of electric brass melting; Elec. World, vol. 1, 1918, pp. 1129, 1216; Electric brass melting from the central-station viewpoint: Trans. Am. Inst. Metals, vol. 9, 1915, p. 395; Crosby, E. L., The present status of the electric furnace: paper at Central Station Sales Managers' Assn. Meeting, August, 1914; Electric furnaces and the central station: Elec. Rev., vol. 74, 1919, p. 799; Electric furnaces and industrial foundries: Elec. Rev., vol. 75, 1919, p. 1022; Smith, A. C., The electric-arc melting furnace and the central station electric company: Trans. Am. Electrochem. Soc., vol. 37, 1920, p. 701; Wilcox, E. A., Electric power from the standpoint of the central station: Trans. Am. Electrochem. Soc., vol. 37, 1920, p. 589; Wirme, H. A., The electric melting furnace as a central station load: General Electric Rev., vol. 24, 1921, p. 510.

in their field, no foundry need refrain from installing an electric brass furnace merely because its force does not include an expert in electrical theory.

TERMINOLOGY AND FUNDAMENTAL ELECTRIC PRINCIPLES OF ELECTRIC FURNACES.

In describing and discussing electric furnaces it is, however, necessary to use certain electrical terms, and in order to comprehend the action of the furnaces some of their governing principles should first be understood. These terms and principles are given below, briefly; rigid definitions are not attempted. For further details standard works should be consulted.⁵⁸

An *electric furnace* is a device in which electrical energy is changed into heat.

An *electric current* is a flow of electricity. It can not be exactly defined. It is measured in *amperes*, which give the rate of flow of the current.

Direct current, or D. C., is current that flows in one direction only, as in a storage battery. It is not used in any electric brass furnace.

Alternating current, or A. C., is a current that flows first in one direction and then in the other. All electric brass furnaces except one utilize alternating current. The rate at which alternating current reverses its direction is the *frequency* or *periodicity* of the current. Usually alternating current is 60-cycle; that is, it reverses its direction 60 times per second; 25-cycle is sometimes, and 40-cycle rarely, used. *High frequency* current, as used in only one furnace, may go up to 25,000 cycles. This one furnace may also use *oscillatory current* in which a discharge gap is used, through which current flows at short intervals. While the current flows it oscillates back and forth. The oscillatory-current system for this one furnace requires *condensers*, which are devices for the storage of one type of electric energy.

An *electric circuit* is the path of the current. The unit of electric current is the *ampere*.

⁵⁸ Stansfield, A., *Electric furnaces as applied to nonferrous metallurgy*: Jour. Inst. Metals (British), vol. 15, 1916, p. 301; The electric furnace, 1914 ed.; Rodenhauer, W., and Schoenawa, J., translated by vom Baur, C. H., *Electric furnaces in the iron and steel industry*, 1917 ed.; Roland, A. L., *Applied electricity for practical men*, 1916 ed.; Fowle, F. F., *Standard handbook for electrical engineers*, sect. 19; Kent, R. T., *Mechanical engineers' pocketbook*; Marks, L. S., *Mechanical engineers' handbook*, sec. 14; Lyon, D. A., and Keeney, R. M., *Electric furnaces for making iron and steel*: Bull. 67, Bureau of Mines, 1916, 144 pp.; Lyon, D. A., Keeney, R. M., and Cullen, J. F., *The electric furnace in metallurgical work*: Bull. 77, Bureau of Mines, 1916, 216 pp.; Seede, J. A., *Electric furnaces*: Gen. Elec. Rev., vol. 11, 1914, p. 653; *Electric-furnace control*, vol. 13, 1916, p. 501; Society for Electrical Development, *Industrial heating as a central-station load*, pt. 1, *Electric furnace*: Bull., 1917; St. John, H. M., *Electric brass melting, its progress and present importance*: Elec. Jour., vol. 16, 1919, p. 373; The evolution of the electric brass furnace: *Iron Age*, vol. 105, 1920, p. 1228; *Trans. Am. Electrochem. Soc.*, vol. 37, 1920, p. 161; The influence of the electric furnace on the metallurgy of the non-ferrous metals: *Trans. Am. Electrochem. Soc.*, vol. 40, 1921 (not yet published); Snyder, E. T., *A B C of iron and steel*, Chapt. XVIII: *Electric steel*, 1915, p. 253; West, C. J., *The electric furnace as applied to metallurgy (Bibliography, 1900-1919)*: *Trans. Am. Electrochem. Soc.*, vol. 37, 1920, p. 461; Acheson Graphite Co., *Brass melting* (pamphlet): Aug., 1920; B. M. J., *Electric furnaces for brass melting*: *Mech. World* (Manchester, England), vol. 67, 1920, pp. 5, 105, 198.

The *volt* is the unit of electrical pressure and is the electrical analogue of hydrostatic head in fluids. The *voltage* is the measure of the force that urges the current to flow.

Electric power is transmitted with the least loss at high voltages, hence much long-distance transmission is done at voltages as high as 66,000 volts or more.

Transmission voltages from near-by central stations are usually 6,600, 4,400, or 2,200 volts. The voltage used on the electric brass furnaces themselves is much lower, being usually not over 120.

Electrical power is the rate of use of electrical energy and is measured in *watts*. One thousand watts equals 1 kilowatt (kw.). A furnace is rated in kilowatts as to its ability to absorb power. *Electrical energy* is that which is transformed into heat in the electric furnace. It is electrical power used for a definite time and is measured in watt hours, or in commercial furnace work in kilowatt hours (kw. h.). A kilowatt hour is the unit by which electric energy is paid for. One kilowatt hour, for furnace use, usually costs between 1 and 2 cents. One kilowatt hour equals 3,412 B. t. u.

Power factor (P. F.) is a number not greater than 1, by which the *apparent* power (volts $\times$ amperes) has to be multiplied to give the *real* power (watts).

Where the power factor is unity, volts $\times$ amperes $\times$ 1,000=kilovolt amperes (kv. a.)=kilowatts. Where it is less than unity, volts $\times$ amperes $\times$ 1,000 $\times$ power factor=kilovolt amperes $\times$ power factor=kilowatts. The power factor is always unity with direct current; it is nearly unity in most resistance furnaces, seldom unity in electric-furnace leads, is usually around 0.80 to 0.90 (usually written 80 to 90 per cent or 80 to 90, 100 being taken as unity power factor) in arc furnaces, and is likely to fall lower in induction furnaces. It is sometimes denoted by $\cos \phi$. Central stations usually charge a "penalty" in their power bills when the power factor falls below 70. Most electric induction-motor loads have a power factor not much above 70, so that electric-furnace loads are usually better for the central station than many motor loads. A low power factor does not mean that more water must flow through a turbine or more coal be used in the generation of the power, but it means that more amperes flow through the circuit. Larger wires and bigger generating machinery are required to carry this excess current that does not become changed into heat in the furnace.

Alternating current is usually so generated and transmitted that more than one current flows in the same system. Each branch of these currents is called a *phase*. Common commercial alternating current is three phase, carried on three wires. Two-phase systems, carried on either three or four wires, are sometimes used, but the three-phase three-wire system is most common. In it each of the three, currents travels on two of the wires, or, when connected in another way, on all three wires.

One current only—that is, single-phase power—may be taken off of two wires. Unless an equal amount is taken off of the other two phases at the same time the full capacity of the three-phase generator will not be utilized.

If a large variable load is taken off of one phase of a small generator, the generator will not be able to maintain proper conditions on the other two phases. The permissible single-phase load that can be taken off a three-phase circuit depends on the capacity of the generating or distributing system. In a large generating system a 300-kw. single phase load is not often objectionable. Where a plant generates its own power from one small generator, a much smaller single-phase load might be impossible to handle. Most plants that melt brass, however, take their power from large generating systems that can handle large single-phase loads; and since electric brass furnaces seldom draw over 300 kv. a., the great majority of such furnaces are single phase, as they

are electrically simpler. A few two-phase furnaces are used, but very few three-phase brass furnaces are now in commercial operation. In steel furnaces the situation is just the reverse, the vast majority being three phase, only a few being run on two-phase or single-phase systems.

The power used in a balanced three-phase three-wire system is not $3 \times \text{volts} \times \text{amperes} \times \text{power factor}$, but $\sqrt{3} \times \text{volts} \times \text{amperes} \times \text{power factor}$. Three-phase power can be transformed to two-phase by a special connection of the transformers known as the *Scott connection* and the three-phase load still kept sufficiently balanced. Three-phase power can not be transformed into single phase and keep the three-phase load balanced save by the use of a three-phase motor, single-phase generator set.

A *transformer* is a device having primary and secondary coils by which power at one voltage and amperage is changed without much loss into power at another voltage and amperage; for example, a transformer may take 6,600-volt current at 84 amperes on the primary and deliver it to a furnace at 110 volts and 2,000 amperes on the secondary. The transformer may have several *taps*, from each of which a different voltage may be taken. An *autotransformer* is a type adapted for use when little change of voltage is needed.

Current is carried on *conductors*, where its conversion into heat is not desired, and carried through *resistors* where it is to be so converted.

Conductors, such as copper, allow current to pass readily; *resistors*, like crushed carbon, allow it to pass with difficulty and generation of heat results; and *insulators*, such as cold porcelain, practically do not allow current to pass at all, though, strictly speaking, there is no perfect insulator.

The *resistance* of a material through which current flows is the opposition it presents to that flow. Resistance is measured in *ohms*. $\text{Amperes} = \frac{\text{volts}}{\text{ohms}}$ (Ohm's law).

The resistance of metals increases with increase of temperature, while that of the nonmetallic conductors, as carbon, graphite, carborundum, and most liquids other than molten metals, decreases with increase of temperature. Some insulators become resistors or even conductors (sometimes called second-class conductors) when heated, as magnesite. It is usually necessary to heat to redness or above. The resistance of these materials falls with increasing temperature; they belong to the class of nonmetallic conductors.

Conductors used to carry the current to electric furnaces may be *wires*, *cables*, or *bus bars*, according to their size and shape. *Leads* is a general term for large conductors carrying heavy currents to furnaces.

The current is brought into and taken out from the furnace itself by *electrodes*, which make connection between the leads and the resistor. The latter may be gaseous—that is, an arc—a solid, or a liquid. The end of an electrode where it joins the resistor is hot; where it joins the lead it is cool. Electrodes for commercial brass furnaces are usually graphite or carbon, rarely copper; if the electrodes are copper they are molten where they join the resistor and are water cooled to keep them solid where they join the leads. Graphite electrodes are also usually water cooled at this point. All electric furnaces except induction furnaces require electrodes.

Resistors may be solid, liquid, or gaseous. The filament in an incandescent lamp and the vapor in a mercury-vapor lamp are resistors. A furnace using a vapor as a resistor is an *arc furnace*. In a *direct-arc* furnace the arc is drawn from an electrode directly to the material to be melted or to other material on top of it. In an *indirect-arc* furnace the arc is drawn from one electrode to another, but not to the material to be heated. In either type, with carbon or

graphite electrodes, the gaseous resistor is largely carbon vapor, derived from the electrode, which is therefore slowly used up. It may also contain the vapor from any other sufficiently volatile material in the neighborhood.

A strictly *resistance furnace* has a solid or liquid resistor. Furnaces with liquid resistors usually fall into other classes as well, so that a resistance furnace usually means one with a solid resistor. The resistor may be divided into parts, and the contact resistance between the parts may be much greater than the resistance of the solid material, as granular resistors and resistors made of piles of carbon plates. The resistance of a conductor or resistor increases as the length is increased or as the cross section is decreased. The heat generated in a resistor (or in a conductor) is proportional to the square of the current and directly proportional to the resistance, as well as to the time the current passes: $\text{Heat} = (\text{current})^2 \times \text{resistance} \times \text{time}$ (Joule's law).

As (power factor being disregarded) power is the product of current times voltage, a 200-kilowatt furnace could take 2,000 amperes at 100 volts or 4,000 amperes at 50 volts. There are always some losses from heat generated by the current outside the furnace in leads and electrodes. As the heat thus generated is proportional to the square of the current, the higher the current and the lower the voltage the harder it is to keep these losses low. Other things being equal, a furnace with a high voltage and a low current has an advantage over one of low voltage and high current. On the one hand, the voltage must be kept low enough not to be dangerous to life, preferably not more than 220 volts. On the other hand, the current must be low enough not to necessitate large and expensive leads and electrodes for holding the current density sufficiently low outside the furnace to prevent loss of power by heating.

Current density is the amperes per unit (square inch) of cross section of a resistor or conductor. Current density is usually around 500 amperes per square inch in copper conductors and 100 to 200 amperes per square inch in graphite electrodes. At very high current densities in large leads the current tends to flow more on the outer parts of the lead and less in the center. This is called the "skin effect." If the current density is high enough, as 5,000 amperes per square inch, in an induction furnace, a conductor, like brass, may become a resistor. An *induction furnace* is a type of transformer in which the secondary consists of one coil only, this being the charge itself. The secondary is also a resistor, so that an induction furnace is a form of resistance furnace.

The generation of current in the secondary of an induction furnace or of a transformer is due to the presence of a *magnetic field* containing *lines of force* in the vicinity of a conductor; that is, the primary. With each change of direction of an alternating current this magnetic field ceases to exist and is then reformed. This change or *flux* in the magnetic field makes the lines of force pass across the secondary, and in passing they *induce* or generate current in the secondary. The magnetic field is strongest—that is, the lines of force closest together—at points nearest the primary. The secondary, therefore, must be close to the primary and so placed as to cut all possible lines of force.

If any lines of force escape without being cut, these escaping lines do no work, so that the apparent power taken from the primary is less than the real power sent to the secondary; that is, a transformer or induction furnace that allows magnetic leakage has a low-power factor, less than unity, and sometimes, in a furnace, the power factor is very low.

In order to force or coerce the lines of force to take a regulated path an iron *core* is provided to interlink the primary and secondary, since the magnetic flux can travel more readily in iron than in air.

Proximity of leads carrying heavy currents to structural ironwork or other iron tends to cause loss of energy through magnetic and other losses, may affect adversely the power factor of the whole furnace installation, and may even make impossible the transmission of the proper amount of power to the furnace. The mutual effect of the currents in different leads may have a similar effect, which can be minimized by interlacing the leads properly.

Eddy currents or *Foucault currents* are set up in magnetizable conductors, such as iron, which are subjected to cyclic magnetization by being in the field of force of a conductor carrying alternating current. They flow more readily in solid conductors than in laminated ones, so the cores of transformers and induction furnaces are made up of piles of thin iron sheets. Magnetic eddy currents cause heating just as alternating current does. They are small in magnitude when the frequency of magnetization is low, so eddy-current losses are usually not great with ordinary 60 or 25 cycle power. With high-frequency current, around 12,000 to 25,000 cycles, the heating may be great, and it is utilized in one type of electric brass furnace. The magnetization set up in iron and steel or in another conductor near a conductor carrying heavy currents may be considerable, so that in the design of a furnace and in the installation of the leads care must be taken to avoid this.

When the power factor is low, whatever the immediate cause may be, the underlying cause is the presence in the circuit of *reactance*. Reactance is analogous to resistance; it impedes or tends to hold back the flow of current and makes the waves of alternating current reach their peak at a different instant from the waves of voltage. Reactance may be intentionally added to the circuit and, if properly chosen, may be so proportioned to the resistance of the circuit as not to interfere materially with the flow of current of a certain strength; that is, it does not greatly reduce the power factor at that current strength, but interferes more and more as the current is increased, until finally the current can no longer increase. This prevents too great surges of current in the circuit. In arc furnaces the arc must be started by barely making connection between the electrodes, with a poor contact. Heat is generated at this poor contact until carbon enough is vaporized to form a gaseous resistor; then an electrode is pulled out of contact with the other electrode and the current is carried thereafter by the vapor. Excessive current may flow while the electrodes are in contact if the reactance of the circuit is low. In a certain arc furnace, normally operating at about 1,700 amperes, a current of 13,000 amperes flowed when the electrodes were touched together. After the insertion of a little reactance in the circuit the maximum short-circuit current was only 4,000 amperes.

Properly chosen current-limiting reactances are useful in arc furnaces, but undesirable reactance should be avoided in all furnaces and especially in the leads. This problem is, in general, one for the attention of the furnace maker when he builds and installs the furnace rather than for the operator.

A *short circuit* is the joining, by a path having much lower resistance, of two parts of a circuit which should have considerable resistance between them, as would result from dropping an iron bar against two electrodes of an arc or resistance furnace. The bar would allow excessive current to flow and perhaps burn out the transformer or cause other damage. Furnaces should be so designed that accidental short circuits can not readily occur.

Circuit breakers usually are installed to protect the line against short-circuits or other overloads.

When two or more things that can carry current, whether conductors, electrodes, or resistors, are one after the other in the same circuit, they are said

to be in series. When they are side by side or so arranged that the current passes through two or more branches, they are said to be in *parallel*.

Current (amperes), voltage (volts), power (kilowatts), energy (kilowatt-hours), and power factor may all be measured by *meters* (ammeters, voltmeters, etc.), which, as well as the switches by which a circuit is made or broken, the circuit breakers, etc., are generally mounted on a *switchboard*. The meters required vary with the type of furnace, but some meters are needed for the operation of any furnace.

Demand meters, as well as kilowatt-hour meters, may be installed by the central station on the readings of which the power bill is made out, but the former are not usually a part of the equipment of the furnace. The central station's kw.-h. meter usually reads all the energy for lights, motors, and furnaces that is used in a plant, being placed on the main feed line to the plant and thus registering the power used on the high-tension side of the transformers and including the losses in them. They can seldom be read as closely as is required for use in furnace operation. Each furnace should therefore have its individual kw. h. meter on its own switchboard. It may be that this meter can not conveniently be installed to record the power used on the high-tension side of the transformer and thus to include the losses of power in transformer and leads, then it does not show the total amount of power to be paid for. The furnace maker or the central station in that case should be asked to determine the ratio between the readings of the furnace meter and the total energy required by the furnace plus the transformer and lead losses.

Some new furnaces are provided with *automatic electrode control* in order to reduce the labor of operating the furnace and to maintain the proper power input. These controls are purely electrical devices and their consideration in this paper would lead the reader too far afield.

Other electrical terms used are either self-explanatory or will be explained when used.

The fundamental principles of furnace design for thermal efficiency are the same for electric as for fuel-fired furnaces. They will be brought out in the discussion of individual furnaces.

FACTORS IN THE APPLICABILITY OF ELECTRIC FURNACES.

As has already been pointed out⁵⁹ in regard to fuel-fired furnaces, no one type or size of furnace can meet properly all the various conditions under which nonferrous alloys are melted. It is therefore necessary, by the study of data, to determine for a given set of conditions, (1) whether any electric furnace is preferable to fuel-fired furnaces; (2) if so, what type will best serve; and (3) how it should be operated to produce the best results.

The main points to consider are⁶⁰ metallurgical results, quality of product, electrical characteristics, reliability and rate of production, thermal efficiency, and costs of operating and maintenance. These phases of the subject will be discussed for each type of furnace.

⁵⁹ Gillett, H. W., Brass-furnace practice in the United States: Bull. 73, Bureau of Mines, 1914, p. 16; Eastick, T. H. A., Melting furnaces: Metal Ind., vol. 18, 1920, p. 319.

⁶⁰ Compare St. John, H. M., Commercial testing of metallurgical electric furnaces: Chem. and Met. Eng., vol. 21, 1919, p. 377.

CRUCIBLE FURNACES IN GENERAL.

No carbon-resistor crucible furnace has had any real success in brass melting; all the furnaces commercially used are of the non-crucible or hearth type. The necessary requirements for a successful hearth-type furnace will be given when that type is discussed.

Electric crucible furnaces will be described first, as they were the first type to be tried out. The furnaces are not discussed in chronological order, but in the order of their classification in Table 2.

In nonferrous melting with fuel-fired furnaces, crucible furnaces have been used because they gave a high-grade product. There is no contact with fuel, as compared to a cupola, and little contact with products of combustion, as compared to the reverberatory or direct flame (oil or gas) furnaces. Little surface is exposed as compared to the reverberatory type, and usually the passage of products of combustion is slower over the exposed surface. Avoidance of an oxidizing atmosphere is fairly easy, and as the heating is mainly from the bottom upward a thermal circulation which does some mixing is started by the rising of the hot metal. Crucibles have been used for producing alloys of high quality, free from oxygen and sulphur, and for melting brasses high in zinc in order to avoid volatilization of that element. Metal losses in melting are lower and the quality of the product higher, generally speaking, than in the reverberatory type of furnace.

These advantages are gained by sacrificing speed of production, by increasing the labor cost, because the capacity of the average crucible is small, and by making the fuel cost high because of the extremely low thermal efficiency of a crucible furnace.

It is improbable that electrically heated crucible furnaces will be used much commercially (1) because the advantage of crucible melting in quality of product can usually be obtained in electric furnaces without using crucibles; (2) because crucibles are fragile and costly; and (3) because more energy is required to melt metal in a crucible than is required to melt it by applying the heat directly, not passing it through the crucible wall.

One of the fundamental principles^a in the design of brass furnaces, especially electric brass furnaces, is that unless some strong reason forbids, the source of heat should be as close to the material to be heated as possible and should be separated from it (if it can not be in contact with it) only by a material that will least resist the flow of heat. For the temperature in question a gaseous atmosphere offers less resistance than any solid or liquid.

^a Compare Johnson, W. McA., and Sieger, G. N., *Electric furnaces, their design, characteristics, and commercial application*: Met. and Chem. Eng., vol. 11, 1918, p. 505; Ray, W. T., and Kreislinger, H., *The flow of heat through furnace walls*: Bull. 81, Bureau of Mines, 1911, 80 pp.

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It is about as reasonable to heat a brick house by building a fire against the walls outside instead of within the house as to use an externally heated crucible for brass melting if a more direct application of heat is possible.

Another principle of furnace design is that, as some of the heat supplied is lost by radiation from the walls or in similar ways, the wall area should be as small as possible in comparison to the volume of metal to be melted. This object can be accomplished by avoiding waste space within the furnace or by increasing the size, and hence the ratio of capacity, to the wall area in a given style of furnace. Furnaces to take a single crucible have small capacity in comparison to wall area, and those to take a number of crucibles usually have much waste space inside. It is not surprising, then, that no electric crucible furnace for brass has proved commercially valuable. This will become obvious as different crucible furnaces are considered.

DESCRIPTION OF CRUCIBLE FURNACES—LIFT-OUT TYPE AND THEIR PERFORMANCE.

FITZGERALD FURNACE.

Fitzgerald²² in 1905 described a crucible furnace for experimental work and in 1915 constructed one for the Titanium Bronze Manufacturing Co., of Niagara Falls, which is described by Vickers,²³ with detailed drawings.

This furnace was made to take two No. 50 crucibles, but as the electrodes projected above the furnace and interfered with pulling the pots much smaller pots were used, the heats cited being of but 25 pounds. The furnace was set in the ground like an ordinary pit furnace. There were two cylindrical melting holes, each 15 inches in diameter by 20 inches high, in which the crucibles were set; these holes had walls made of carborundum fire sand and a bond rammed into place, that were $1\frac{1}{2}$ inches thick and were surrounded by 4 inches of crushed coke to serve as resistor. Carborundum fire sand was placed between the annular resistor space and the fire-brick walls of the furnace. The two annular spaces were joined, so that the current entered the furnace by one electrode at each end, ran in parallel around both sides of the resistor about one melting hole, then around the other resistor, and out the other electrode at the opposite end of the furnace. Suitable heat-insulating bricks were placed around the sides, top, and bottom of the furnace, and pits were provided below to catch the metal if a crucible broke.

²² Fitzgerald, F. A. J., Resistance furnace for crucibles: *Electrochem. and Met. Ind.*, vol. 3, 1905, p. 55.

²³ Vickers, C., Electric furnace as a copper-melting medium: *Foundry*, vol. 45, 1917, p. 280.



A. BAILY 8-POT CRUCIBLE LIFT-OUT FURNACE.



B. SMALL BAILY CRUCIBLE LIFT-OUT FURNACE USED FOR MELTING SILVER.

An incomplete log of one day's run is given. The power input averaged about 60 kw., being 22 kw. (120 volts, 190 amperes) at the start and rising to 86 kw. (120 volts, 720 amperes) as the resistor became heated. The voltage was then reduced from time to time as the furnace became hotter and the resistance of the resistor fell.

After two and one-half hours, in which nearly 150 kw. h. had been used, 25 pounds of copper was charged and was melted in 25 minutes, using about 28 kw. h. The power was kept on between heats. After 25 minutes, in which about 25 kw. h. was used, another 25 pounds of copper was charged and poured at 1,300° C. in 32 minutes, having used about 30 kw. h. The furnace was then heated empty for 1 hour and 15 minutes and a heat of nickel (weight not given) charged; this charge was poured 1 hour and 45 minutes later. Apparently but one of the melting holes is represented in the log. The furnace could not show its true efficiency under such a test, only 25 pounds of copper being charged in a furnace made to take two No. 50 crucibles. Seven and one-half hours and over 400 kw. h. were required to melt 50 pounds of copper and an unspecified weight of nickel.

The resistor developed hot spots, and the resistance of the coke resistor changed from day to day. Vickers says: "The furnace is not particularly efficient from a thermal standpoint and it is doubtful whether a granular-resistance furnace can be improved to any great extent. Some other form should be devised."

GREAVES-ETCHELLS CRUCIBLE FURNACE.

Greaves and Etchells⁴⁴ suggest a crucible lift-out furnace for three-phase current. In this furnace a granular resistor surrounds a refractory cylinder, within which a single crucible is set. Six electrodes, suitably connected to the different phases and separated by blocks of nonconducting material, so that a uniform flow of current in the resistor is produced, project downward into the annular ring of granular resistor. It is not known whether or not this furnace has been constructed.

BAILY CRUCIBLE LIFT-OUT FURNACE.

A small Baily furnace, made by the Electric Furnace Co., has been in commercial use in the melting of silver at the plant of W. A. Rogers, Ltd., Niagara Falls, since 1914. This furnace is a rectangular fire-brick box with two resistor troughs, connected in series, along the sides. (See Pl. I, B.) Two small crucibles are set on small fire-brick pedestals between the resistors. The furnace is rated at 40 kw. and is said to produce 1,000 ounces of silver per hour after the furnace is fully heated up, or eight heats (16 pots) in a

⁴⁴ Greaves, H. A., and Etchells, H., British Patent 115,866 of 1917.

10-hour day. According to Miller,⁶⁵ the power consumption is about 960 kw. h. per ton, and there are practically no metal losses.

FEATURES OF FURNACE.

In February and March, 1915, at the plant of the Detroit Copper and Brass Rolling Mills, Detroit, a Baily furnace was tested, which was the largest electric brass furnace built up to that time. (See Pl. I, A.) One of the writers was present during this test.

The primary current, which was single-phase alternating-current, 440-volt, 60-cycle, was led to a transformer having a series of oil switches for changing the transformer ratios. The transformer voltages were 242, 231, 220, 209, 198, 154, 143, 132, 121.

The furnace, which was rated at 200 kw., was set in an excavation, with its top a little above the floor level. A rectangular steel shell measuring approximately 10 by $8\frac{1}{2}$ by 6 feet and having suitable stiffening ribs inclosed the whole furnace.

The furnace floor was laid on two courses of hollow building tile, the spaces in which were filled with infusorial earth. On this were two $2\frac{1}{2}$ -inch courses of fire brick. The side walls were of 9-inch fire brick, backed by 6 inches of loose infusorial earth, and the end walls of $4\frac{1}{2}$ -inch fire brick, backed by 6 inches of loose infusorial earth. The straight side walls extended about $1\frac{1}{2}$ feet above the floor proper, and the roof was an arch 9 inches thick. In the bottom of the furnace were laid eight large fire-brick slabs about 4 inches thick, on which the crucibles were set.

The resistor troughs were rammed in place in wooden forms of carborundum fire sand and water glass, and the wooden forms were later burned out. The troughs were 9 by 6 by 94 inches, inside dimensions, and their walls were about 2 inches thick. They were supported at about 1-foot intervals by silica brick, by which the bottom of the trough was raised 5 or 6 inches from the floor in order to allow free radiation from the bottom of the resistor. A row of fire brick was laid at an angle across the joint between floor and side wall, near the lower edge of the resistor trough, to reflect the heat backward into the chamber. The arched roof, 9 inches thick, had its bottom $3\frac{1}{2}$ feet above the floor proper at the center of the span. Four oval holes in the roof about 15 inches wide by $2\frac{1}{2}$ feet long gave access to the crucibles. Aside from the resistor troughs and the silica brick supports for them the whole furnace lining was ordinary fire brick set in red clay.

The resistor walls were about 3 inches from the fire-brick side walls. The crucibles were placed about 4 inches from the resistor,

⁶⁵ Miller, D. D., The electric furnace for heating nonferrous metals: Jour. Am. Inst. Metals, vol. 11, 1917, p. 283.

about 12 inches from the end walls, and were 6 to 9 inches apart the short way of the furnace and 9 to 12 inches apart the long way.

The tops of the crucibles were about $1\frac{1}{2}$ feet from the bottom of the roof arch at the center of the span. The bottoms of the crucibles were about 4 feet below the upper steel shell of the furnace. The furnace bottom was built slightly sloping toward one end, and a tap hole, plugged by a brick, was provided with the idea of draining off spilled metal.

This furnace had a volume of 440 cubic feet in the walls, roof, and bottom, with a surface area of 390 square feet. The crucibles and contents occupied 10 cubic feet, and there was 60 cubic feet of vacant space in the furnace.

ELECTRODES.

The electrodes were 4 by 8 inch (made up of two 4 by 4 inch) carbon, extended at each end into the resistor troughs, and extended outside the furnace into asbestos millboard boxes in which was carborundum fire sand. Flexible copper strips were bolted to the electrodes and cable or busbars to the strips. There was no water cooling.

RESISTORS.

The resistors were each 99 inches long by 6 inches deep by 9 inches wide (dimensions inside trough), giving in series 188 inches of length and 54 square inches cross section. The highest voltage used was 242 at 205 kw.—a maximum of about $1\frac{1}{2}$ volts per inch length and $15\frac{1}{2}$ amperes per square inch of cross section. The resistor material was granular (chip) Acheson graphite of 3 to 6 mesh. It was heaped over the electrodes. The resistor was replenished when necessary through a long iron spout.

ROOFS AND COVERS.

The four oval openings in the roof were covered by four rectangular covers, about 17 inches wide by 30 inches long by 8 inches high, consisting of fire brick, topped by kieselguhr, all held in a five-sided steel box, with the open side down. These heavy covers were mounted on 6-inch car wheels running on five heavy I-beams placed across the furnace as tracks. The steel box projected downward about an inch below the bricks of the cover and fitted into a rectangular groove surrounding the oval opening, into which sand was placed to form a sand seal. By means of long handles and a system of toggle joints the cover could be lifted far enough to clear the roof, and an automatic catch, actuated by a spring, held the cover up so that it was free to roll on the wheels. When the cover was to be replaced, it was rolled into the proper position over the sand seal, and by

pulling a chain on the handle the catch was thrown out and the cover would drop into place. The handle could be thrown back out of the way in order to get into the furnace on the handle side of the cover. Each cover thus served one opening, or two crucibles, as it could be rolled either way.

CRUCIBLES.

The crucibles were No. 70's, of the same form and size as those used in coke furnaces of the plant. Like crucibles for crucible steel, they were more nearly cylindrical in form than are ordinary foundry crucibles. At first only the tongs used on coke furnaces were provided; the depth of the electric furnace, however, was so great that the workmen's hands were but a few inches above the furnace roof when using them, and longer tongs had to be used. As the roof openings were only about 15 inches the short way and the crucible plus the tongs had an equal width, the tongs to clear the openings had to be run across the furnace.

This furnace was designed to melt 1 ton of yellow brass with 500 kw. h. or less after temperature equilibrium was reached; that is, under continuous 24-hour operation.

PRELIMINARY DRYING AND HEATING OF FURNACE.

The power was first thrown on the new furnace at 4.30 p. m., February 19, 1915, the resistors being in multiple. The wooden form for the resistor troughs was slowly burned out and the empty furnace heated off and on at rather low power. At 3.08 p. m., February 24, the chamber was at 1,975° F. (1,080° C.), 4,110 kw. h. having been used in drying and heating the furnace.

Various attempts to melt were made in which numerous difficulties cropped up, the most serious of which was due to the presence of so many crucibles in one large inclosure. When a cover was opened to lift a pot the outrush of hot gas was so great as to make working over the furnace very hot and arduous.

Moreover, the furnace was full of zinc fume, which made it hard to see the location of the pot, so that it was difficult to place the tongs so as to avoid spilling the metal. Spilled metal, especially if it happened to drop into the resistor, caused even worse fume, and it was decided to add a flue to the furnace to take off the fumes.

After the flue was finished a test was made, first preheating the furnace chamber to 1,300° C.

The alloy to be melted was 58 per cent copper and 42 per cent zinc, this alloy having about the lowest melting temperature and the lowest pouring temperature, estimated at 1,025 to 1,050° C. (1,875 to 1,925° F.), of any used in the shop. The charges weighed 209.5

pounds each and consisted of 75 pounds scrap brass (usually heavy pieces), 75.5 pounds copper, and 59 pounds zinc.

The test was started at 8 a. m. March 2, when the first pot was charged. The eighth pot was charged at 8.37 a. m. The 242-volt tap was used till 8.15 a. m., when 185 kw. were being drawn. The 231-volt tap was then used till 8.45 a. m., when 192 kw. were being drawn. The 220-volt tap then was used till 9.08 a. m., when 168 kw. were being drawn, power dropping on this tap. The 231-volt tap then was used till 9.49 a. m., when 205 kw. were being drawn and a fuse blew. At 9.55 a. m. the fuse was replaced and power again put on, using the 220-volt tap, which gave 187 kw. At 10.07 a. m. the 198-volt tap was used and 145 kw. drawn. The power dropped on this tap to 143 kw. at 10.15 a. m. From 10.17 to 11.07 a. m. the power was thrown off during a wait for a new sledge for bending tongs into place; then the 231-volt tap was put on and 110 kw. drawn, which rose to 122 at 11.15 a. m. At 11.22 a. m. the power was thrown off, as it was decided not to charge any more metal till good double-prong tongs were available.

The resistor had burned away somewhat, which may partly account for the trouble in maintaining the desired power. On too high a tap the power would shoot up too much, and on too low a tap it would drop too much. The attempt in this test was to use as much power as the furnace would stand.

The temperature of the chamber dropped from $1,300^{\circ}$ to $1,200^{\circ}$ C. at 9.15 a. m., as the cold charges were put in, rose to $1,340^{\circ}$ C. at 10.15 a. m., fell when the power was off to $1,320^{\circ}$ C., then held about constant till 11.30 a. m., the power being off at 11.22 a. m. At 11.45 a. m. it had dropped to $1,305^{\circ}$ C. The pots in the furnace after 11.22 a. m. were taken out on stored heat, the last coming out at 12.50 p. m., when the temperature was $1,920^{\circ}$ F. ($1,050^{\circ}$ C.), the covers having been open to let fume escape. The covers were then opened a great deal to allow replacing the displaced bottom bricks and to replenish the resistor, and at 3.45 p. m. the temperature of the chamber was 815° C.

TROUBLES.

At 8.50 a. m. the caster burned his clothing from a flare of gas and fume over his feet and objected to the excessive heat. He declared the conditions were far worse for the melter than in handling crucible steel and could not be endured in summer.

The roof in the center of the furnace showed signs of settling, was red-hot to the steel shell, and the middle rail had warped so that one or two covers had to be moved by a bar, as they could no longer be rolled. Because the covers were warped the sand seal was no longer effective, and zinc fumes came out of the furnace even when all

covers were closed. The fumes were very bad after the first pots were splattered, and the new flue seemed to have no effect, possibly because the old annealing furnace, into which it opened, was not tight enough.

Much trouble developed with the tongs, partly because the single-prong tongs did not get a firm grip, but chiefly because the fumes made the pots invisible and the heat was hard even for gloved hands at several feet above the openings. No asbestos gloves were provided.

A still worse trouble was caused by the incipient slagging of the fire-brick blocks on which the crucible rested, so that the crucibles adhered to the blocks. Even if the tongs were placed properly, the crucible could not be pried loose. Sometimes the bottom blocks came up and had to be barred loose from the hanging crucible, a dangerous proceeding with tongs softened by heat; sometimes one man placed the tongs, fastened them tightly, and forced them from side to side, while another took hold of the crucible top with a long pair of splattering tongs and did the same. After many attempts—in the case of two crucibles, seven each—the pots were lifted, but the bottom blocks were so much disarranged that the crucibles could not be replaced. Three crucibles were ruined by breaking pieces out of the top and by scoring with crucible tongs. In efforts to dislodge the crucibles much metal was spilled, and much ran out when the tongs broke pieces out of the top of the crucibles. All this spilled metal caused heavy fume, and as the tong trouble caused delay several pots became overheated, adding to the fume.

Carborundum fire sand was then placed under each crucible to prevent sticking.

On account of the delays through crucibles sticking to the bottom this run did not go one full round of eight pots before it was necessary to throw off the power. Consequently the figures of power consumption again are only approximate. They approach 500 kw. h. per ton under conditions of continuous operation. On account of the spilling into the furnace, the figures for weight of metal lost would have been of no value and were not taken.

At 3.45 p. m., after the bottom bricks were replaced and carborundum fire-sand was put under all crucibles to prevent sticking, when the temperature was 815° C., power was turned on again to heat the furnace for the following day's run. At 7.40 a. m. the next morning, March 3, the temperature was 1,330° C. About 1,500 kw. h. were used in 16 hours in heating the empty furnace.

A new pair of long double-prong tongs was made, so that the crucible could be grasped more securely. Another full-capacity test was then run.

SECOND FULL-CAPACITY TEST.

The test started at 7.40 a. m. On 242 volts the power was 168 kw., and crept up to 192 kw. at 8.02 a. m., when the 231-volt tap was used; the power then started at 183 kw. and reached 195 kw. at 8.30 a. m.; the 220-volt tap then gave 174 kw., but the power fell on this voltage to 156 kw. at 9.30 a. m. As the temperature was still as high as the refractories would stand, a lower tap (198 volts) was thrown in, and the power fell from 135 kw. to 101 kw. at 10.30, when the 220-volt tap was again used, the furnace taking 132 kw.

The temperature in the chamber fell, as cold metal was charged, from 1,330° C. at 7.40 a. m. to 1,230° C. at 8.30 a. m., and then rose to 1,345° C. at 9.45, falling to 1,205° C. at 10.30 and rising to 1,275° C. at 10.58 a. m. The temperature near the resistor was not so much affected by charging and rose from 1,355° C. at 7.40 a. m. to 1,445° C. at 10.00 a. m., and then fell to 1,405° C. at 10.58 a. m.

ADDITIONAL TROUBLES.

The condition of the roof, covers, and warped rail in the middle of the furnace became worse during this run. Some bricks from one of the end covers fell into one pot of metal. The pots did not stick to the bottom bricks, and the new tongs worked well as long as the caster could see where to place them. The first five pots came out on the first attempt to lift them, a record far better than in the previous tests.

The working conditions were still bad. The third pot to be recharged and put back in the furnace was not set in exactly the proper position, as the fumes made it impossible to see what was being done. The bottom was not replaced exactly level after being torn up by the sticking of the pots on the previous day. The pot had been charged with all the scrap brass and small pieces of copper it would hold before being replaced in the furnace, as this was easier on the men than replacing the empty crucible and then charging it. On the first heat in any one pot, as the empty crucibles had remained in the furnace, a long funnel was used for charging.

This third pot tipped over and some of its contents were spilled, but it was righted without much trouble.

At 10.58 a. m. the fumes were very bad, the flue having no appreciable effect; it was impossible to see into the furnace, so that the tongs were not properly placed on one pot. In consequence this pot dropped from the tongs, spilling about half its contents. This hot metal, almost at its boiling point, spilled into a resistor trough, exploded because of the sudden volatilization of zinc, and a shower of molten metal burned the clothes of the bystanders. The granular resistor was blown out of the trough, and the fume from the spilled metal was so great that the test had to stop at once. As before, the metal in pots already in was melted by stored heat.

The difference in temperature between the end of the chamber and the side by the resistor began to equalize fairly soon after the power was turned off, as is shown below:

TABLE 3.—*Equalization of temperature between end of chamber and side of resistor.*

Time.	Temperature of chamber, no power on.		Temperature of wall near resistor.		Difference.
	° F.	° C.	° F.	° C.	
10.58 a. m.	2, 330	1, 275	2, 560	1, 465	130
11.30 a. m.	2, 300	1, 260	2, 500	1, 370	110
12.00 m.	2, 260	1, 240	2, 420	1, 330	90
12.30 p. m.	2, 225	1, 220	2, 320	1, 270	50
1.00 p. m.	2, 170	1, 190	2, 265	1, 240	50
a 1.24 p. m.	2, 130	1, 165	2, 210	1, 210	45

a Last pot out.

TABLE 4.—*Log of final test of Bailly 8-pot furnace, March 3, 1915; 58 per cent copper, 42 per cent zinc.*

Con- secu- tive heat No.	Pot No.	Time started charging.	Time all charge in but zinc.	Time zinc added.	Time pulled.	Remarks.
27	1	7.40 a. m.	8.39 a. m.	9.09 a. m.	9.21 a. m.	Pot not charged. Do. Do.
28	2	7.44 a. m.	8.40 a. m.	9.12 a. m.	9.35 a. m.	
29	3	7.48 a. m.	8.45 a. m.	9.15 a. m.	9.56 a. m.	
30	4	7.52 a. m.	8.46 a. m.	9.17 a. m.	10.08 a. m.	
31	5	8.10 a. m.	8.37 a. m.	a 11.42 a. m.	a 12.02 p. m.	
32	6	8.05 a. m.	8.49 a. m.	a 11.35 a. m.	a 11.51 a. m.	
33	7	8.02 a. m.	8.48 a. m.	a 10.29 a. m.	a 11.32 a. m.	
34	8	7.58 a. m.	8.47 a. m.	a 10.26 a. m.	a 10.32 a. m.	
35	1	9.30 a. m.	10.11 a. m.	a 12.29 p. m.	a 12.35 p. m.	
36	2	9.47 a. m.	10.13 a. m.	a 12.31 p. m.	a 12.45 p. m.	
37	3	10.05 a. m.	10.35 a. m.	a 12.54 p. m.	a 1.03 p. m.	
38	4	10.25 a. m.	10.45 a. m.	a 12.57 p. m.	a 1.14 p. m.	
39	8	10.47 a. m.	a 11.40 a. m.	a 1.20 p. m.	a 1.24 p. m.	

a No power on; melting done by stored heat.

Between 7.40 a. m. and 10.54 a. m., or in about three and one-fourth hours, nearly 525 kw. h. had been used, or the average power input was a little over 160 kw. The final temperature at 10.54 a. m. was 55° C. lower in the chamber and 50° C. higher near the resistor at the end than at the start; that is, the furnace temperature may be taken as the same at the start and finish. The work completed in this period was as follows:

Work done in heat.

Full heats.

5 pots out, equivalent to	5.0
4 pots all in but spelter, but not ready to spelter, approximately half done	2.0
1 pot ready to come out	1.0
2 pots ready to spelter, approximately 80 per cent done	1.6
1 pot just in, approximately 10 per cent done	.1
Total full heats	9.7

If these 9.7 full heats be taken as equivalent to 10 full heats (2,095 pounds) for truly continuous running, the power consumption was 500 kw. h. per ton.

During the whole test, from the first warming up of the furnace to the time the power was shut off, over 12,500 kw. h. were used, and about 4 tons of metal melted.

The crucibles seemed to be in fairly good condition at the end of the test, considering that they had been left empty continuously in the furnace during all heating and cooling periods and had been severely damaged by the first two sets of tongs.

METAL LOSS.

Disregarding the spilled pot, the 12 charges, 2,514 pounds (clean metal), melted in the last test gave 2,370 pounds of metal in ingots, a gross loss of 144 pounds, or 5.75 per cent. Ninety pounds of metal was recovered from spillings and skimmings, making the net loss 54 pounds, or 2.15 per cent. The furnace as built was a failure mainly for the following reasons:

1. Fume and heat made conditions extremely bad for the workmen. Nearly everyone working around the furnace had "brass shakes."

2. The rate of production was low.

3. The power efficiency was not good.

4. The furnace took so long to cool and to heat in case of needed repairs that an accident interfered greatly with production.

GIROD FURNACE.

Girod⁶⁶ suggested a crucible furnace with a number of crucibles contained in an inclosure, on the sides and bottom of which is a granular resistor. No data on its actual use or trial have come to light. As the design involves driving the heat through the walls of the inclosure as well as through the crucibles, it seems to be of low efficiency and to involve many difficulties in the maintenance.

SNYDER CRUCIBLE LIFT-OUT FURNACE.

The Commonwealth Edison Co., of Chicago, desiring to aid the development of electric furnaces, in order to lay the foundation for the sale of power for them, had a furnace designed by F. T. Snyder and tested at the plant of the L. Wolff Manufacturing Co., Chicago, in 1913 and 1914. The later tests were made by H. M. St. John.

This was a pit furnace, taking one No. 80 crucible. The resistor was a cylinder made of ordinary crucible material (graphite bonded with fire clay). Slots were cut in this cylinder and filled up with

⁶⁶ Girod, P., U. S. Patent 885,745, Apr. 28, 1908; application Dec. 20, 1905.

alundum cement, the object being to insure stability, and thus to increase the resistance of the resistor. The crucible was set inside the resistor, which, in turn, was surrounded by fire brick and heat-insulating brick walls. Electrical control was obtained by varying the voltage.

Some yellow brass was melted in this furnace, but despite many attempts to improve it the life of the resistors was short—not over 40 hours running time, even though for half of that time the resistor temperature was below $1,000^{\circ}\text{C}$. The resistor could not be run at a temperature, of the resistor itself, above $1,300^{\circ}\text{C}$; no efficient brass melting could be done in a furnace whose resistor temperature is not over $1,300^{\circ}\text{C}$. for reasons clearly pointed out by Northrup.⁶⁷ The short life of the resistor was due both to oxidation of the resistor, which was retarded somewhat by arranging a slow drip of oil into the furnace, and to the development of hot spots in the resistor and consequent melting of the fire-clay bond.

Whenever attempts were made to melt brass at a commercial rate of speed, the resistor broke down before reliable results could be obtained. The incomplete data indicated that even if the resistor troubles could be overcome, a furnace of such a type and size could hardly be expected to reach as low a figure as 500 kw. h. per ton of yellow brass, even under the most favorable conditions.

GREENWOOD AND HUTTON CRUCIBLE LIFT-OUT FURNACE.

Greenwood and Hutton⁶⁸ describe a furnace in which a crucible is set inside a cylindrical grid made up of carbon rods standing vertically, their ends so joined by carbon blocks that the rods are connected in series. (See figs. 2 and 3.) The rods are covered with a carborundum fire-sand—water glass coating to prevent oxidation. The furnace handled a 90-pound charge and took about 20 kilowatts at about 65 volts, 300 amperes. Data are given only on two heats of 90 pounds of “nickel-silver” (nickel brass or German silver) scrap. In these, starting from a cold furnace, the first heat came out in 3 hours, using 54 kw. h., the second in 1 hour and 35 minutes, using 29 kw. h. The authors calculate that on 10-hour operation the furnace would melt nickel brass with about 750 kw. h. per ton, and on 24-hour operation at about 600 kw. h. per ton. These figures would be somewhat reduced for alloys of lower melting points than nickel silver.

No data are given to indicate the probable life of the resistor, but the resistor is inherently fragile, subject to weakening by oxida-

⁶⁷ Northrup, E. F., *Production of high temperature and its measurement: Met. and Chem. Eng.*, vol. 17, 1917, p. 685.

⁶⁸ Greenwood, H. C., and Hutton, R. S., *Note on an electric-resistance furnace for melting in crucibles: Jour. Inst. Metals (British)*, vol. 17, 1917, p. 239.

tion, and because of the careful fitting of joints required in construction expensive in comparison with its probable life.

HOSKINS CRUCIBLE LIFT-OUT FURNACE.

A more fully developed crucible furnace is the Hoskins carbon-plate resistor furnace, which has been commercially used in small

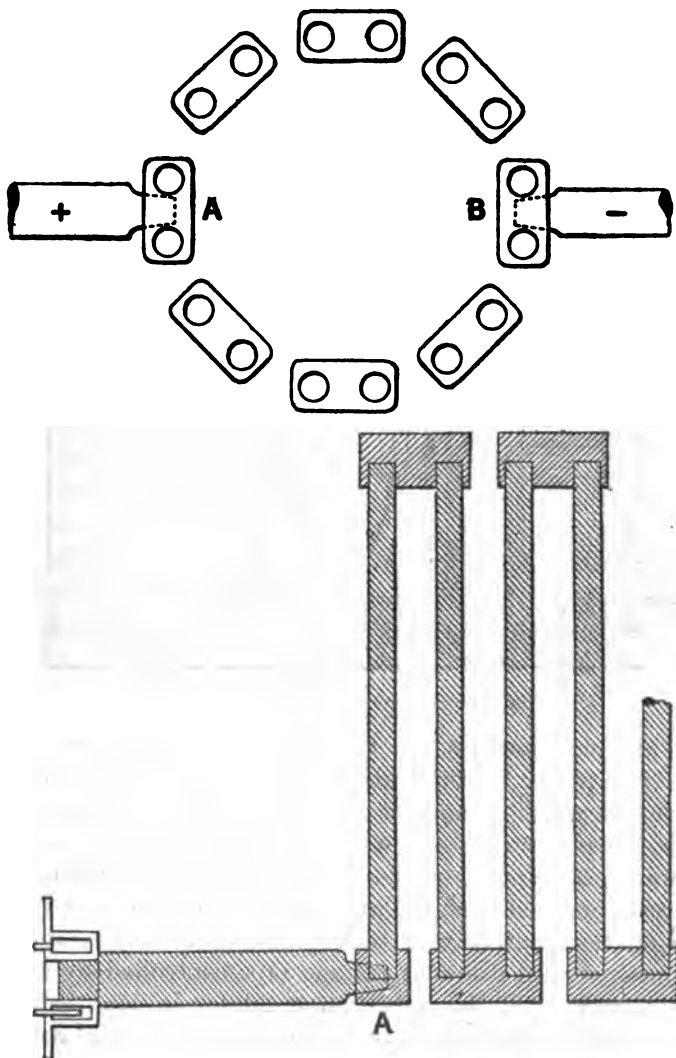


FIGURE 2.—Detail of Greenwood and Hutton resistor.

sizes by the Hoskins Manufacturing Co., of Detroit, in the production of nickel-chromium alloys. As the need for larger melting units has increased, the crucible furnace has been gradually supplanted for this purpose by the ordinary or horizontal ring type of

induction furnace, and the indications are that the latter type in turn will be displaced by direct-arc furnaces for melting these alloys.

The crucible furnace has a resistor composed of thin carbon plates, usually $\frac{1}{4}$ inch thick, $1\frac{1}{2}$ to $2\frac{1}{4}$ inches wide, and 8 to 15 inches high, in various sizes of furnaces. The plates are set upright and against each other, like the leaves of a book. Part of the resistance of this resistor is in the plates themselves, but most of it is in the contacts between the plates. By means of a screw bearing against a carbon or graphite block at one or both ends of a "book" the pressure between the blades can be varied at will, and thus the resistance of the resistor can be varied within wide limits.

Since the resistance of a carbon resistor falls markedly with increase of temperature, all carbon-resistor furnaces must have some

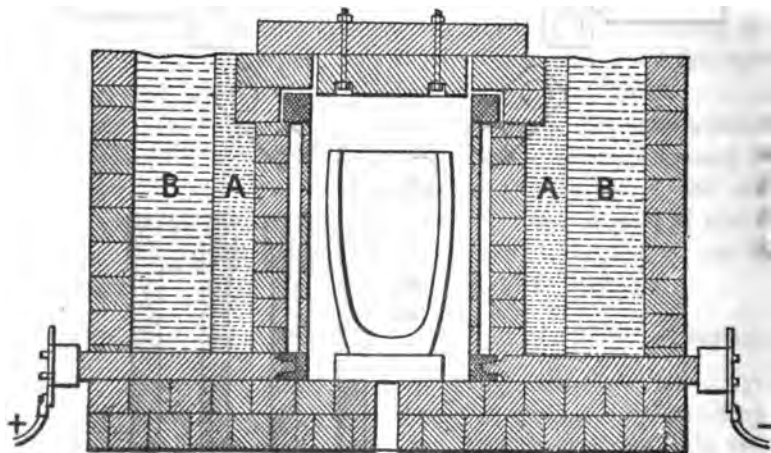


FIGURE 3.—Greenwood and Hutton furnace.

method of compensating for this change of resistance, and thus keeping the power input under control. In small experimental laboratory furnaces a constant voltage is sometimes used and part of the power is absorbed and wasted in a rheostat. For commercial use, however, some other means of varying the voltage must be used. An induction regulator on the primary side of the transformer gives some regulation, but even if that is used a variable-voltage transformer with several different voltage taps controlled by switches is usually required. If the induction regulator is not used, a larger number of different voltage taps is required to give real control over the furnace.

With the Hoskins furnace a considerable control of power input can be effected by altering the contact resistance between the plates, so that a transformer with fewer voltage steps and with longer steps can be used.

In the early days of the work of the Bureau of Mines, before experience had shown that electric crucible lift-out furnaces for brass were not likely to be of commercial use, the Hoskins type was tried out in several different forms and sizes, the first object being to determine the metal losses of different alloys in a crucible lift-out furnace. The patent⁶⁶ on this type of furnace is controlled by the Hoskins Manufacturing Co., but through the courtesy of Mr. A. L. Marsh of that firm the bureau was permitted to utilize the principle of the furnace in the construction of experimental furnaces for laboratory tests. In a later patent Thomson⁷⁰ also shows a crucible lift-out furnace surrounded by an adjustable carbon resistor, whose resistance lies mainly in the contacts between the carbon pieces.

BUREAU OF MINES TESTS OF HOSKINS-TYPE FURNACE.

A good many tests of the Hoskins furnaces were made which gave instructive data on metal losses and on furnace design. The original data of these tests, as well as much other early data, were lost in the fire that destroyed part of Morse Hall, Cornell University, in February, 1916. The entire contents of the Ithaca field office were lost in the fire, so that the data below are taken from summarized reports that had been made to the Washington office.

Most of the small Hoskins furnaces made by the Hoskins Manufacturing Co. have two rows of carbon plates on two sides of the furnace, connected by a heavy carbon or graphite plate at the back, heat thus being generated on two sides only. (See fig. 4.) Most of the furnaces built by the bureau had resistor plates completely surrounding the crucible. In five of the modifications tested, the furnace took but one crucible, and in one modification three No. 20 crucibles were used. Carborundum bricks usually supported the blades; with magnesite brick next the carbon plates, though, of course, not in contact with them. Back of the magnesite were fire

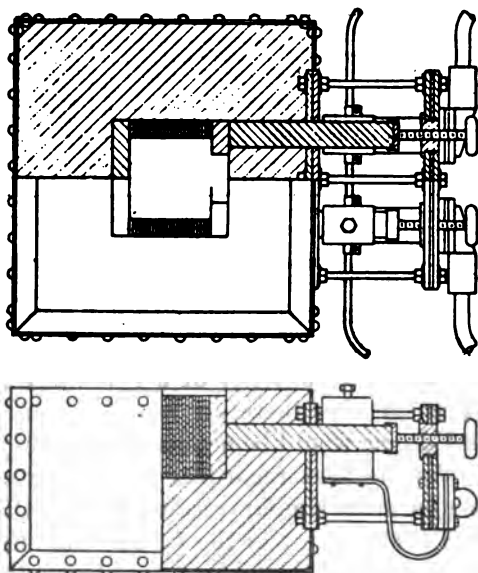


FIGURE 4.—Diagram of Hoskins crucible lift-out furnace.

⁶⁶ Marsh, A. L., U. S. Patent 882,788, June 5, 1906.

⁷⁰ Thomson, J., U. S. Patent 950,879, March 1, 1910 (see fig. 7).

brick, and on the outside either loose infusorial earth in a sheet-iron shell or infusorial-earth brick. Fire-brick covers fitting into a sand seal and provided with charging holes closed by smaller fire-brick covers were used. Table 5 shows some of the general dimensions of the furnaces.

TABLE 5.—*Dimensions of Hoskins small-sized crucible lift-out furnaces.*

No.	Dimensions of melting chamber (inside plates).	Crucible.				Height of carbon plates.	Location of bottom of crucible in relation to bottom of resistor.	Kw. rating.
		No.	Height.	Largest diameter.	Capacity, brass.			
	<i>Inches.</i>		<i>Inches.</i>	<i>Inches.</i>	<i>Pounds.</i>	<i>Inches.</i>		
1	11 by 11 by 14 $\frac{1}{2}$ high.....	20	10 $\frac{1}{2}$	8 $\frac{1}{2}$	60	8	Level.....	50
2	11 $\frac{1}{2}$ by 11 $\frac{1}{2}$ by 17 high.....	(a)	10 $\frac{1}{2}$	9	90	8	2 $\frac{1}{2}$ inches above.	50
3	10 $\frac{1}{2}$ diameter by 18 high.....	20	10 $\frac{1}{2}$	8 $\frac{1}{2}$	60	8	Level.....	50
4	13 by 13 by 16 high.....	40	12 $\frac{1}{2}$	10 $\frac{1}{2}$	120	10	do.....	75
5	12 $\frac{1}{2}$ by 12 $\frac{1}{2}$ by 23 high.....	(b)	16	9	145	15	1 inch below.	65
c 6	10 $\frac{1}{2}$ by 10 $\frac{1}{2}$ by 13 $\frac{1}{2}$ high.....	20	10 $\frac{1}{2}$	8 $\frac{1}{2}$	60	8	Level...	50

^a Special short cylinder.

^b Special tall cylinder.

^c Triple furnace.

^d Middle compartment; end compartments straight next to middle, oval toward ends.

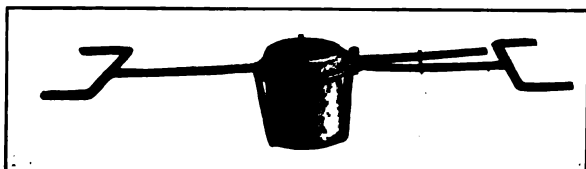
^e Each crucible.

No attempt was made to run any of these furnaces continuously, the aim being mainly to secure data on melting losses and a general idea of the efficiency of a given size and type in the use of power. The furnaces therefore were built as cheaply as possible and not as solidly as would be necessary were they intended for commercial use.

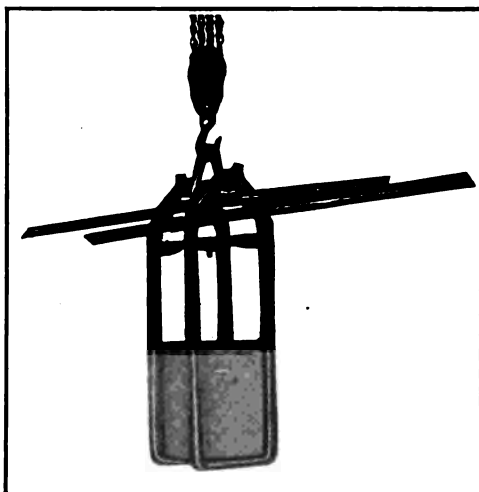
As the tests regarding loss of metal on the various sizes of furnaces gave similar results, these will be considered together after the performance of the furnaces in other respects has been discussed. The order in the table is not always that in which the furnaces were tested.

The results on furnaces 1, 2, and 3 may be considered together. Furnaces 1 and 2 were almost identical, having a square melting chamber, with resistor plates on all four sides, square 2 by 2 inch graphite end blocks as tall as the plates at three corners, each with two adjusting screws bearing on two carborundum bricks set back of the blocks for varying the pressure on the blocks. Graphite electrodes brought to the ends of the rows of resistor plates were installed at the remaining corner and insulated from each other by a magnesite brick.

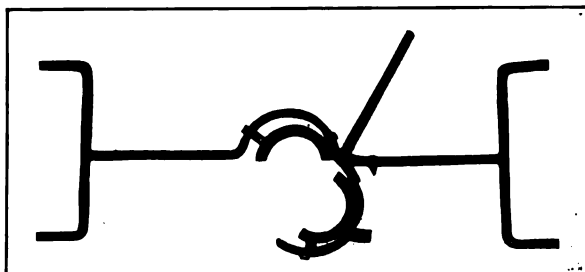
Furnace 1 used the standard crucible, handled by the usual tongs. The shape of such a crucible leaves some waste space in the melting chamber, and a cylindrical crucible was tried in furnace 2 to test the advantage of utilizing this waste space. Special means had to be provided for lifting the cylindrical crucible, as it had no bilge.



A. SHANK FOR CYLINDRICAL CRUCIBLE.



B. LIFTING DEVICE FOR CYLINDRICAL CRUCIBLE.



C. ANOTHER SHANK FOR CYLINDRICAL CRUCIBLE.



A. EXPERIMENTAL HOSKINS-TYPE CRUCIBLE LIFT-OUT FURNACE FOR NO. 40 CRUCIBLE.



B. HOSKINS CRUCIBLE LIFT-OUT FURNACE FOR NO. 70 CRUCIBLE.

This was accomplished by a pair of loop tongs. The crucible was set on a brick in the furnace, and when it was ready for pouring one loop was lowered into the furnace and slipped under one side of the crucible. The other loop was then similarly inserted and a pin, extending downward on a V-shaped projection extending from the upper part of the loop, was inserted into a hole on a similar V-shaped projection on the other loop. A VV-shaped bar held the two loops in the two outer curves and was hoisted by a hook in the center loop. The crucible was held securely as it was hoisted from the furnace. It was then set down upon a round fire block, the loop tongs removed, and a basket shank slipped about the crucible for pouring. The lifting and pouring devices are more thoroughly described by Lohr¹¹ and are shown in Plate II, *A, B, C*.

Furnace 3 was built to get away from the waste space in the corners of the furnaces with square melting chambers. In this the resistor was made as near to a circle as possible by making it in the form of a hexagon, five sides of which were made up of rows of carbon plates pressing against two sides of an equilateral-triangular prism of graphite. The adjusting screw bore against a carborundum brick back of the third side of the triangle. The electrodes, with magnesite brick insulation between them, were on the sixth side of the hexagon.

In order to lift the crucible, which was too close to the resistor to allow the use of any sort of tongs, it was set on a tall brick pedestal that could be raised by a handwheel and gearing till the bottom of the crucible was above the resistor blades, when the crucible could be grasped by tongs or could be lowered until the bottom of the crucible was on a level with the bottom of the resistor.

Representative runs on five furnaces, all melting an alloy of about 78½ per cent copper, 16 per cent zinc, 2½ per cent tin, and 3 per cent lead, charged in small ingots of 8 to 12 pounds, are tabulated in Table 6.

All furnaces were cold at the start save No. 4, which was at 375° at the start, and No. 5, which was at 350° C. The metal was poured at about 1,200° C., save with furnace 5, from which it was poured at about 1,165° C. A number of other runs were made with the furnaces, which agreed very well with those given below in Table 6.

¹¹ Lohr, J. M., U. S. Patents 1,173,444 and 1,173,445, Feb. 29, 1916 (dedicated to the public).

TABLE 6.—Results of tests of different types of Hoskins furnaces.

Furnace 1. Square melting chamber; regular No. 20 crucible.

Heat No.	Power on.	Power off.	Weight of charge.	Power used.
			Pounds.	Kw. h.
1.....	9.00 a. m.	9.55 a. m.	56	104.5
2.....	10.07 a. m.	10.55 a. m.	55	47
3.....	11.05 a. m.	11.47 a. m.	55	30
4.....	1.03 p. m.	1.37 p. m.	55	32.5
5.....	1.40 p. m.	2.27 p. m.	55	33
6.....	2.40 p. m.	3.13 p. m.	55	34.5
7.....	3.23 p. m.	3.53 p. m.	55	28.5
8.....	4.03 p. m.	4.30 p. m.	55	28.5
Total.....			440	434.5

Furnace 2. Square melting chamber; short cylindrical crucible.

			Pounds.	Kw. h.
1.....	9.00 a. m.	9.35 a. m.	85	46
2.....	9.45 a. m.	10.25 a. m.	80	37
3.....	10.37 a. m.	11.13 a. m.	80	31
4.....	12.20 p. m.	1.13 p. m.	80	33
5.....	1.23 p. m.	2.13 p. m.	80	29
6.....	2.24 p. m.	3.10 p. m.	80	28.5
7.....	3.20 p. m.	4.05 p. m.	80	28
Total.....			625	427.5

Furnace 3. Hexagonal melting chamber; regular No. 20 crucible.

			Pounds.	Kw. h.
1.....	9.00 a. m.	9.35 a. m.	55	72.5
2.....	9.49 a. m.	10.25 a. m.	55	36
3.....	10.26 a. m.	11.16 a. m.	55	32
4.....	12.27 p. m.	1.10 p. m.	55	30
5.....	1.50 p. m.	1.56 p. m.	55	28
6.....	2.06 p. m.	2.46 p. m.	55	26
7.....	2.56 p. m.	3.30 p. m.	55	28.5
8.....	3.40 p. m.	4.15 p. m.	55	25.5
Total.....			440	427.5

Furnace 4. Square melting chamber; regular No. 40 crucible (warm at start).

			Pounds.	Kw. h.
1.....	8.00 a. m.	9.41 a. m.	120	120
2.....	10.05 a. m.	10.42 a. m.	120	61
3.....	10.56 a. m.	11.47 a. m.	125	46
4.....	1.00 p. m.	1.41 p. m.	120	43
5.....	1.57 p. m.	2.43 p. m.	120	44
6.....	2.55 p. m.	3.35 p. m.	120	40
7.....	3.45 p. m.	4.28 p. m.	120	36
Total.....			845	430

Furnace 5. Furnace chamber at 350° C. at start; metal poured at 1,165° C., average.

			Pounds.	Kw. h.
1.....	7.00 a. m.	8.55 a. m.	147.5	132
2.....	9.17 a. m.	10.30 a. m.	146	60
3.....	10.50 a. m.	11.42 a. m.	140	45
4.....	11.59 a. m.	12.46 p. m.	140	49
5.....	1.42 p. m.	2.20 p. m.	138	33
6.....	2.36 p. m.	3.28 p. m.	138.5	30
Total.....			850	438

a All kw. h. figures taken on secondary side of transformer.

b 10 per cent zinc alloy.

c Average, 79 kw. h. per 100 pounds.

d Average, 43 kw. h. per 100 pounds.

e Average, 63 kw. h. per 100 pounds.

f Metal charged at 12.01; no power on furnace over noon hour.

g Average, 46 kw. h. per 100 pounds.

h Part of heat A charged at 1.03 and left in furnace; no power on over noon hour.

i Average, 41 kw. h. per 100 pounds.

In Furnace 5 a tall cylindrical crucible holding about 145 pounds of metal was used, the crucible being 16 inches tall, the resistor blades being 15 inches tall, with their tops level with the top of the crucible, the crucible bottom being thus 1 inch below the bottom of the resistor.

The crucible was split at the top in pulling heat 6 from the furnace, so the run could not be finished. Had two more heats been taken out, the power consumption would have been cut down to about $37\frac{1}{2}$ kw. h. per 100 pounds. The space available for pouring was so small and cramped that it was impossible to pour quickly and the time taken for pouring was excessive.

Much trouble was experienced in running furnace 5 on account of top heating. The top of the resistor blades were on a level with the top of the crucible and were 1 inch shorter than the crucible. The crucible was filled quite full on the first heats, and a good deal of heating was done by heat reflected by the roof. Melting took place first on top of the charge—not from the bottom as in the other furnaces where the resistor plates were at least 2 inches from the top of the crucible—and the bottom of the resistor was level with the bottom of the crucible ($2\frac{1}{2}$ inches below it in furnace 2). The metal on top was bubbling and spitting at the end of the heats, and although the pots were stirred the last metal poured was rather cold. The average pouring temperature was $1,165^{\circ}$ instead of $1,200^{\circ}$ C., as in the runs with the other furnaces.

In order to avoid trouble from overheating at the surface, it was necessary to reduce the power input below the figure desired in order to let conduction of heat through the metal keep pace with the rise in temperature and bring the metal out more evenly heated from top to bottom.

This test showed the desirability of heating crucibles from the bottom rather than from the top and indicated that surface overheating is likely to occur in hearth-type furnaces with deep baths heated from above unless some means of stirring the metal is adopted.

The boiling and spitting of the surface layer of metal was probably made more noticeable by the rather low barometric pressure (73.6 cm.) on the day of the test. For a crucible of this tall and narrow form to be useful it should be heated from the bottom, in order to set up thermal circulation by the rising of hotter metal, instead of being heated from the top and allowing the hotter metal to remain at the top. On regular operation these furnaces would give results about as follows, if they started on an eight-hour operation with the furnace at a temperature of 500° C. or more from the previous day's heating:

TABLE 7.—*Summary of the probable performance of different sizes of Hoskins-type furnaces.*

Furnace No.	Charge.	Kw. h. per ton.		Daily output.	
		8-hour operation.	24-hour operation.	8-hour operation.	24-hour operation.
	<i>Pounds.</i>			<i>Pounds.</i>	<i>Pounds.</i>
1.....	55	1,500	1,000	450	1,800
2.....	90	850	625	650	2,300
3.....	55	1,250	950	450	1,900
4.....	120	850	625	850	3,200
5.....	140	700	500	1,000	3,600

Besides the improvement in Nos. 4 and 5 over No. 1, which was to be expected, as the capacity of the furnace per charge was more than doubled, No. 3, in which the waste space between crucible and resistor was less than in No. 1, gives materially better results than No. 1; No. 2, in which the use of the short cylindrical crucible increases the capacity of the furnace over No. 1, is still more of an improvement over No. 1 and does as well in power consumption, though not in output, as No. 4. No. 5 did not show as much improvement as was expected because of the trouble from top heating.

Furnace 4, which is shown in Plate III, A, gave the results in Table 8 on yellow brass of 63 per cent copper, 33½ per cent zinc, 3½ per cent lead, charged as 10-pound ingots and poured at about 1,065° C.

TABLE 8.—*Test on Hoskins furnace 4 for melting yellow brass; No. 40 crucible square furnace.*

Heat No.	Power on.	Power off.	Pounds charged.	Kw. h.
1.....	6.40 a. m.	7.10 a. m.	125	114
2.....	7.36 a. m.	9.00 a. m.		
3.....	9.15 a. m.	10.11 a. m.	123	55
4.....	10.28 a. m.	11.17 a. m.	124	38
5.....	11.30 a. m.	12.10 p. m.	126	29
6.....	12.27 p. m.	1.10 p. m.	124	31
7.....	2.26 p. m.	3.08 p. m.	125	29.5
8.....	3.21 p. m.	4.20 p. m.	123	30
	4.32 p. m.	5.29 p. m.	124	32
Total.....			994	358.5

a Power off 26 minutes, fuse blown.

b Part of charge for heat 6 put in at 1.20 p. m.

c Average, 36, kw. h. per 100 pounds.

On 10-hour operation on yellow brass and on calculated 24-hour operation furnace 4 should then give the following results: Charge, 125 pounds; kw. h. per ton, 10-hour operation, 700; 24-hour operation, 500; output, 10-hour operation, 1,000 pounds; 24-hour operation, 3,600 pounds.

Furnace 4 (single No. 40 crucible) used 181 carbon plates 2 by 10 by ¼ inch. The cold furnace started at about 200 volts, 150 amperes

(30 kilowatts); at higher power inputs the voltage and amperage were as follows at various times:

TABLE 9.—*Electrical data on Hoskins furnace 4.*

Kw.	Volts.	Amperes.	Kw.	Volts.	Amperes.
79	101	780	60	115	520
78	104	750	18	90	c 200
50	150	a 450	66	94	700
50	100	b 500	60	100	600

a Starting up after cooling at noon.
b At full running temperature.

c Reducing power at end of run.

The current taken at any voltage can be carried through a rather wide range by changing the pressure on the plates by means of the adjusting screws.

TRIPLE FURNACE.

In a furnace taking a single crucible the heat from half of the resistor only is directed toward the crucible. It seemed likely therefore that a gain in efficiency could be made if the resistors were placed between the crucibles, so that the heat from both sides of the resistor struck the crucibles directly. Moreover, if three separate square furnaces were made into one holding three crucibles the latter could readily be constructed with about only two-thirds as much area in its surface as the total area of the surfaces of the three separate furnaces.

This design is more fully described by Lohr and Gillett,⁷² who show also how it is adapted for three-phase current.

FURNACE 6.

A triple furnace for No. 20 crucibles (regular form) was constructed, in which there were but two rows of resistor plates, connected in series by a back plate, like the usual Hoskins design. Instead of the straight side walls next the resistors of a furnace for a single-phase crucible, the furnace was extended lengthwise, space being made outside each resistor for another crucible. The ends of the furnace were in semicircular form, the end walls being brought

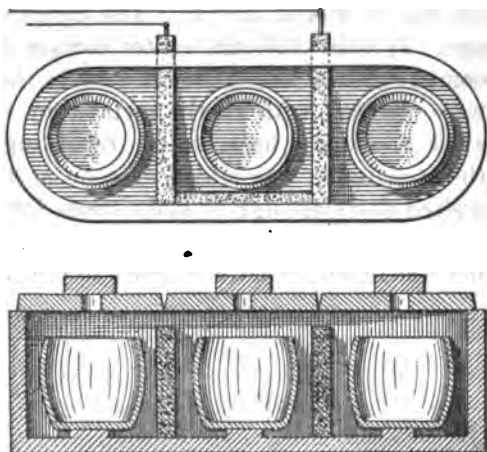


FIGURE 5.—Bureau of Mines crucible furnace, triple form.

⁷² Lohr, J. M., and Gillett, H. W., U. S. Patents 1,162,178 and 1,162,179 (dedicated to the public), Nov. 30, 1915.

as close to the outer crucibles as possible allowing room for tongs. The general principle of this furnace is shown in figure 5.

In the first construction the resistor plates were bare and not protected by troughs. As there were only two rows of resistor plates against the three rows in furnace 1, the resistance of the whole resistor was less, and it was desirable to run the furnace with as high a contact resistance between plates as possible; that is, with the plates as loose as possible. As the space allowed for the tongs was small in order to avoid waste, it was difficult to avoid hitting the plates when removing the crucibles. This tended to displace the plates. Hence the furnace was rebuilt, with the resistors inside carborundum troughs made up of slabs $\frac{1}{2}$ inch thick of pure unbonded carborundum, Refrax grade. Carborundum-slab covers were put over the troughs to protect the resistors from mechanical damage and from oxidation. The furnace cover was made in three portions, each having a charging hole, so that but one section had to be removed to take out any one crucible.

The data on a run melting the regular alloy used in comparing the different furnaces—78 $\frac{1}{2}$ per cent copper, 16 per cent zinc, 2 $\frac{1}{2}$ per cent tin, 3 per cent lead—follow:

The furnace was at 90° C. at the start; at 8.30 on the morning after the run it was at 350° C. The metal was poured at 1,200° C. average. In this table the center compartment is called No. 2, the end compartments Nos. 1 and 3. As in the previous tests, the crucibles were charged with all the metal they would hold (in small ingots), after pouring a heat, before returning them into the furnace, and the remainder was added as soon as the metal in the crucible had melted down enough to make room for it.

In Table 10 the time given, after the start, is the time at which a pot was taken from the furnace to pour.

TABLE 10.—*Log of run on furnace 6 (triple No. 20).*

Time.	Pounds poured from compartment No.			Kw. h. used in time intervals given.	Total kw. h. used so far. ^a
	1	2	3		
a. m.					
7.00.....	Start.	Start.	Start.		0
8.55.....		61		110	110
10.03.....		61 $\frac{1}{2}$		46	156
10.45.....			61 $\frac{1}{2}$	26	182
10.51.....		61 $\frac{1}{2}$		11	193
11.12.....	61			8	205
11.44.....		60		23	228
p. m.					
12.30 ^b				23	251
1.00 ^c					
1.23.....		61 $\frac{1}{2}$		7	258
1.45.....			69	28	286
2.15.....		60		27	313
2.36.....	67 $\frac{1}{2}$			9	324
3.05.....		61		24	348

^a Average, 41 kw. h. per 100 pounds.

^b Power off for noon.

^c Power on.

TABLE 10.—*Log of run on furnace 6 (triple No. 20)*—Continued.

Time.	Pounds poured from compartment No.			Kw. h. used in time intervals given.	Total kw. h. used so far.
	1	2	3		
<i>p. m.</i>					
3.10.....			60½	4	352
3.51.....		61		25	377
4.23.....			61½	14	391
4.39 ^d		60		19	410
4.43.....	70			0	410
Total ^e	198½	547½	252½		

^d Power off.^e 998 pounds poured.

The resistor between compartments 2 and 3 was evidently generating more heat than that between compartments 1 and 2, as only three heats were obtained in compartment No. 1 against five in No. 3. This was remedied in later tests by connecting a voltmeter to show the voltage drop across either resistor and by adjusting the blade pressure until each row of blades gave the same voltage drop. This test on furnace 6 was run one hour longer than that given above on the single No. 20 furnace (No. 1) in order to stop when all three pots had been poured within 10 minutes. The power was off furnace 1 for one and one-fourth hours instead of one-half hour. Had furnace 1 been run one and three-fourths hours more it would have produced three more 55-pound heats, or a total of 11, and would have used a total of 420 kilowatt hours.

We can then compare the single No. 20 (furnace 1) and the triple No. 20 (furnace 6) as follows:

Operation.	Output.		Kw. h. per ton.	
	Single.	Triple.	Single.	Triple.
9-hour operation.....	<i>Pounds.</i> 565	<i>Pounds.</i> 1,000	1,500	820
24-hour operation.....	1,800	4,000	1,000	700

The advantage of the triple furnace is even more marked when we consider that on 9-hour operation 420 kw. h. would produce 565 pounds of metal in the single furnace, whereas the triple 410 kw. h. produced 547½ pounds in the center compartment alone, as well as 450 pounds from the end compartments.

The directing of the heat from the outside of the resistors toward other crucibles instead of toward walls made much of that heat do

useful work and did not decrease the work done in the center compartment. The improvement doubtless would not be so striking in a furnace of larger size nor on continuous operation, but it would be decided.

In order to see if any saving could be made by keeping the furnace hot at night, in another test the furnace was run empty at a low power input for five hours, using a total of 161 kw. h. The melting chamber came up to $1,150^{\circ}$ in two and one-half hours, using 98 kw. h., and was held there two and one-half hours by the use of 63 kw. h. more, or an average power input of 25 kw. Melting was then begun, four heats of 60 pounds each being made in two hours (two from the center compartment, one each from the ends) using 98 kw. h. The metal was poured at $1,200^{\circ}$ C., the regular alloy being used.

The furnace was probably nearly but not fully heated to the "stationary state" or "steady state," in which the inside is at full running temperature, the walls are so heated that they no longer absorb and store heat on further running, and the outside of the furnace being at maximum temperature the losses by radiation from the walls are highest. In the "steady state" the power supplied to the furnace either is usefully applied in melting or is lost by radiation, conduction, and convection from the wall and cover surface into the floor and from the electrodes by water cooling. None is fed to the walls to be stored there, as the walls take only the energy needed to maintain them throughout in a temperature equilibrium with the hot interior of the furnace and the cold air outside.

It was intended to supply power to the furnace through the night as needed to hold the temperature of the furnace constant, but the porcelain protecting tube of the pyrometer used for taking chamber temperatures was broken and the couple itself injured just as it was inserted, so that the power supply had to be adjusted by visual rather than pyrometric control of temperature.

From 3.12 p. m., when the last heat was poured, to 8 o'clock the next morning the furnace was run empty at about 10 kw., 163 kw. h. being used in the 16 hours. The furnace then appeared to be nearly but not quite at running temperature. To test this a few heats were taken off, as given in Table 11.

TABLE 11.—*Log of run on triple furnace, starting hot after running at night.*

Time.	Pounds poured from compartment No.			Kw. h. used in time intervals given.	Total kw. h. after 8 a. m. ^c
	1	2	3		
a. m.					
8.00.....	Start.	Start.	Start.		0
9.11.....	do.	61		63	63
9.43.....			60	30	93
9.49.....	59½			5½	98½
9.55.....		60½		5	103½
10.40.....		62½		38½	142
11.04.....	55½			18	160
11.08.....			60½	7½	167½
11.20 ^b		59½		7½	175
Total c.....	115	243½	120½		

^a Average, 36½ kw. h. per 100 pounds.

^b Power off at 11.15; metal poured at 11.20.

^c 479 pounds poured.

Calculated to a 24-hour operation basis, the triple furnace gave about 3,500 pounds output at 730 kw. h., a fairly close check on the figures calculated from the other test given above. The calculated output is somewhat too low and the power consumption somewhat too high, because the furnace was not fully hot at the start.

On a 9-hour basis, keeping the furnace hot the other 13 hours at 10 kw., it appears, considering both tests, that the figures would become about as follows: Output in 9 hours, 1,400 pounds; kw. h. in 9 hours' running, 420; in 13 hours, empty, 130; total, 550, or 890 kw. h. per ton. The power cost per ton therefore would be about the same as on 9-hour operation without night heating, but the output would be increased 40 per cent.

The best method of operation can not be calculated accurately from such runs as these; it must be worked out by actual long-time tests. The triple No. 20 furnace did not do quite as good work as the single No. 40 furnace, as would be expected, because the triple furnace had greater wall area and more waste space than the single No. 40.

The Hoskins Manufacturing Co. later built a double furnace for two No. 20 crucibles in which nickel-chromium alloys were melted, in which there were two rows of resistor plates along the long sides of the rectangular furnace and two crucibles set side by side in the same chamber. Exact data are not available. It is said to have shown a materially decreased power consumption over the use of two separate No. 20 furnaces, but not as much decrease as was shown by the bureau's triple furnace over the single one.

OBJECTIONS TO CRUCIBLE FURNACES TESTED.

Aside from the general disadvantages of crucible furnaces—small units, cost of crucibles, and high cost of labor—the main objection is the short life of the resistor plates. At prewar prices the plates cost

about 3 to 5 cents apiece, according to size, and would be more expensive now. In the practice of the Hoskins Manufacturing Co. on melting nickel-chromium alloys the plates are replaced every Sunday or every other Sunday, when the furnaces are regularly used 9 or 10 hours a day, in order to avoid shutdowns during the week. The electrodes, end blocks, and connector plates also have a relatively short life. The failure of the carbon and graphite parts is due to the entrance of air when the furnace is opened to charge or pour and to leakage of air, while running, if the furnace is not tightly closed.

Sufficiently exact data were not obtained to give the true cost of carbon parts per ton of metal melted, but at prewar prices it appeared probable that even on the larger furnaces the cost would lie between \$1 and \$2 per ton. At any rate, it would be a considerable item of expense, and it was necessary to see if it could be reduced. Through the kindness of the National Carbon Co. some plates were obtained in which a carborundum rim was molded around the plate, forming a sort of "picture frame" of carborundum. Through the courtesy of the Carborundum Co. the edges of some plates were subjected to silicon vapor (the "siliconizing" process)⁷³ and a layer of "silundum" or "silfrax" formed thereon. These with plain companion plates were all put in a furnace which was run as follows, cooling to room temperature, with the furnace closed, between each heating:

TABLE 12.—*Times and temperatures of run on protected resistor blades.*

Run No.	Hours heating.	Furnace empty or melting.	Maximum temperature of furnace.	Temperature of metal poured.
			[°] C.	[°] C.
1.....	3	Empty.	1,350	
2.....	1½	do.	1,075	
3.....	1½	do.	1,000	
4.....	7½	Melting.		1,200
5.....	3	Empty.	800	
6.....	3	Melting.		1,300
7.....	1	Empty.	800	
8.....	2	Melting.		1,000
9.....	9	do.		1,075

Average weights after a total running time of 28½ hours.

	Average original weight.	Average final weight.	Average loss.	Loss.
	Grams.	Grams.	Grams.	Per cent.
Small carbon.....	92	36	56	60
Large carbon.....	139	61	78	56
Small "picture frame".....	111	62	49	44
Large siliconized.....	122	78	44	36

⁷³ Compare Tone, F. J., U. S. Patent 1,013,700, Jan. 2, 1912; Carborundum refractories: Met. Chem. Eng., vol. 11, 1913, p. 485.

The protecting coat tended to crack off from both sorts of protected plates rather early in the test. While such protection does increase the life, it probably does not increase it enough to make up for the increased cost. Protection of the plates by placing them inside a trough or box made of thin carborundum slabs, as was done on furnace 6, was a much more effective way of preventing oxidation of the resistor. Baffling the flow of heat by a solid wall is inadvisable from the point of view of thermal efficiency and might be expected to reduce the efficiency materially. In the triple furnace, however, there was no perceptible difference between the furnace with open blades and with blades in the carborundum box, either on rate of production or on power consumption.

Granular graphite was tried as a resistor material in the triple furnace, held in carborundum troughs, but it was not found possible to maintain the resistor in a tall narrow trough, as hot spots sooner or later developed. On making the troughs sufficiently wider and shallower to minimize that difficulty the voltage, even with the wider and shallower trough, had to be kept below a certain limiting value for each different resistor or the hot spots would again develop. The triple furnace was rated at 50 kw. and in order to get resistors in 12-inch lengths (2 resistors in series) that would take 50 kw. it was necessary to use troughs 7 inches wide by $4\frac{1}{2}$ inches deep, inside dimensions, filled evenly with 14-mesh graphite and covered with carborundum slabs. This resistor when fully hot took about 40 volts at 1,250 amperes. Coarser graphite would probably have caused less danger of hot spots, but none was available. The resistors were now each 8 inches wide outside instead of about 4 inches, as were the carborundum boxes inclosing carbon plates; hence it seems that the waste space and the 8 inches added length demanded by the substitution of granular resistors would be likely in so small a furnace to lower the efficiency of the furnace.

A few tests were also made on the construction of crucible furnaces of the multiple type, in which an arc is used as the heat source, but when an arc (between the two electrodes) is run near an ordinary graphite crucible bonded with clay the excessive local arc temperature soon melts the bond and makes the crucible disintegrate.

BOCUZE FURNACE.

Bocuze⁷⁴ has suggested a furnace in which a crucible rotates on its vertical axis and is heated by two arcs, one on each side, which move up and down, the motion of the crucible and arc distributing the heat uniformly over the surface of the crucible. This plan appears to require an unusual amount of mechanism for the heating

⁷⁴ Bocuze, J. M., U. S. Patent 1,053,563, Feb. 18, 1913.

of one crucible, and such a furnace would therefore be expensive to build.

Attempts were made by the Bureau of Mines to heat a furnace containing two crucibles between which was one arc separated from the crucibles by carborundum slabs to distribute the heat more uniformly; the carborundum, however, was raised to its decomposition point near the arc before the crucibles were hot enough to melt brass.

The bureau's work on crucible lift-out furnaces indicated (1) that, although metal losses for alloys high in zinc were much less in electric crucible furnaces than in fuel-fired furnaces, and (2) although the efficiency of crucible lift-out furnaces could be increased by the use of multiple furnaces arranged for internal heating, with resistors between the crucibles, in place of a single crucible with resistors along the walls, nevertheless (3) the crucible furnace can not be made to compare in efficiency with furnaces of the hearth type. The cost of crucibles, also, and the fact that at best the crucible type is a small-unit furnace, made it advisable to turn to other forms of electric furnace.

FACTORY TEST OF NO. 70 HOSKINS FURNACE.

In the meantime a test on melting brass was made by the Hoskins Manufacturing Co. at its own factory in a larger-size furnace than those described above, a representative of the Bureau of Mines being present. This furnace is shown in Plate III, *B* (p. 51).

ARRANGEMENT OF FURNACE.

The lift-out furnace, called the "F. C. 108" furnace, was built up in an iron shell 43 by 43 by 29 inches. The furnace chamber proper inside this shell was 19 by 19 by 16 inches high. About this was a lining of magnesite brick $4\frac{1}{2}$ inches thick, backed up by a $2\frac{1}{4}$ -inch layer of fire brick on the sides and by $8\frac{1}{2}$ inches of fire brick on the bottom. About the sides, the space between fire brick and shell, 5 inches on each side, was packed with infusorial earth. There was a square hole $2\frac{1}{2}$ by $2\frac{1}{2}$ inches in the bottom of the furnace to allow the metal to run out if the crucible should break in the furnace. This was closed with a fire-clay plug at the bottom.

The resistor consisted of two rows of carbon blades $2\frac{1}{2}$ by 15 by $\frac{1}{4}$ inches, standing at opposite sides of the furnace. There was just room in the furnace chamber for a No. 70 crucible. The tongs were put on diagonally, the corners allowing space for them. A No. 70 crucible is rated at a capacity of about 210 pounds of red brass, or about 195 pounds of yellow brass. The hoist used to lift the crucible

was a little difficult to handle and the metal was likely to spill into the furnace during the lifting of the crucible if filled too near the top. The electrodes were graphite cylinders 8 inches in diameter. These were laid in a carborundum fire-sand packing and pressed against contact blocks of graphite $2\frac{1}{4}$ by $2\frac{1}{4}$ by 15 inches, which, in turn, pressed against the blades. At the other end of the electrodes water-cooled electrode holders made contact with flexible laminated copper connectors, to which the leads were joined. The electrode holders were provided with adjusting screws by which the contact resistance between the blades could be varied by varying the pressure. The two rows of blades, about 60 blades on a side, 120 in all, were joined in series through three carbon plates $1\frac{1}{2}$ inches thick, 5 inches high, and 19 inches long. As there is no contact resistance on this side, the carbon plates, though forming part of the resistor, have a lower resistance than the blades and generate little heat. On the electrode side of the furnace the furnace chamber was lined with magnesite brick, standing between the two graphite contact blocks or "push blocks." Thus the melting chamber had magnesite on one side, high-resistance resistors on two sides, and low-resistance resistor on one side. The heat was generated mainly in the sides with the two high-resistance resistors and was not evenly distributed about the crucible.

The furnace chamber was raised 4 inches above the furnace shell by large fire-brick blocks about 2 by 1 by 4 feet, so laid as to project $2\frac{1}{4}$ inches over the resistor blades, at 1 inch distance above them. This distance is rather small, and the fire brick began to drip a little at the end of the tests, although no serious trouble resulted. The projecting ledge served to protect the tops of the blades from oxidation and the operator from some of the direct radiation when he pulled the crucible from the furnace.

The fire-brick blocks were not tightly laid, and the iron shell might well have been 4 inches taller. The fire-clay plug in the drainage hole in the bottom was not tight, and there was some draft through the furnace from top to bottom, as the furnace was not properly closed. This draft hastened oxidation of the blades and increased the zinc loss from volatilization. On top of the massive blocks was a cover of 4-inch fire brick, with a charging hole in the center, that was in turn covered with fire brick.

DETAILS OF TEST.

The run on May 5, 1914, began at 10.15 a. m., the furnace being cold. The metal melted on this day was yellow brass ingot containing 66 per cent copper, 33 per cent zinc, and 1 per cent lead. The first two charged were of ingots weighing about 30 pounds each. The

metal was poured into smaller ingots weighing about 10 pounds each. The transformer efficiency was not known, but it was probably not as high as on ordinary transformers, as a special testing set of transformers had been designed for ready control by a number of changes in voltage rather than for efficiency. Calculating transformer efficiency at 100 per cent, the furnace was started on 234 volts, 240 amperes, 55 kv. a. on the primary, and 42.5 volts, 1,300 amperes, on the secondary. As the resistor warmed up the power taken increased, till in half an hour the primary showed 220 volts, 390 amperes, 85 kv. a., corresponding to 40 volts, 2,150 amperes, on the secondary. This was too high a current for the leads, which began to heat up, as the leads consisted of four cables, each of about 300,000 circular mils. Each of these two leads was about 6 feet long. The transformer ratio was changed at 10.57 a. m., so that for 220 volts, 390 amperes, 85 kv. a. on the primary, the secondary was taking about 45.5 volts, 1,750 amperes. About 90 pounds of metal had been charged at the start, and it took till noon, or $1\frac{1}{4}$ hours, before this had melted down so that another 30-pound ingot could be charged. Up to noon 180 kw. h. had been used. (The watt-hour meter being of course connected on the primary all kw. h. figures include transformer and lead losses.) The heat went on more rapidly from that point, till at 12.23 p. m. the charge, totaling $181\frac{1}{2}$ pounds of metal, was all in. The furnace was run at an average of 85 kw. until 12.30 p. m., when the power was reduced by regulation of the blade pressure to about 80 kw., at which figure it ran till 12.39; it was then reduced further to 40 kw., and then progressively to 15 kw. at 12.45, when the metal was at $1,060^{\circ}$ C. and the heat was poured. The heat took $2\frac{1}{2}$ hours and used 183 kw. h., or 101 kw. h. per 100 pounds on the first heat from a cold furnace. The men who poured the metal into ingots were not used to crucibles of this size and spilled a good deal, increasing the gross loss. There was poured $176\frac{1}{4}$ pounds of ingot, and $4\frac{1}{2}$ pounds of clean metal were recovered from spillings and skimmings. The net loss was thus 0.4 per cent. A little salt was used as a flux on this heat and on subsequent heats.

The power was kept on between heats, but the adjusting screws were loosened, so that not much power was drawn by the furnace. The power thus used while the furnace was empty is charged to the heat next following. The other heats will not be described in detail, but are tabulated in Table 13.

TABLE 13.—*Test of Hoskins furnace, No. 70 crucible, on yellow-brass ingot.*

Heat No.	Time current on.	Current off.	Amount charged.	Ingot poured.	Metal recovered from spill and skim.	Total.	Net metal loss.	Pouring temperature.	Kw. h. used.
			<i>Pounds.</i>	<i>Pounds.</i>	<i>Pounds.</i>		<i>Pounds.</i>	<i>° C.</i>	
1.....	10.15 a. m.	12.45 p. m.	181.50	178.25	4.5	180.75	0.75	1,060	a 183
2.....	12.55 p. m.	1.56 p. m.	188.25	183.00	4.5	157.50	.75	1,065	78
3.....	2.04 p. m.	2.57 p. m.	194.00	(b)				1,065	75
4.....	3.15 p. m.	4.03 p. m.	192.00	186.50	4.0	190.50	1.50	1,060	68
5.....	4.10 p. m.	4.42 p. m.	107.50	104.00	3.0	107.00	.50	1,065	46
Total.....			863.25				c 3.50		d 450

a Furnace cold at start.

b Much metal spilled into furnace in lifting pot.

c Net metal loss on 699.25 pounds, 3.50 pounds, or 0.52 per cent.

d That is, 1,010 kw. h. per ton.

At 5.06 p. m. the outside wall of the furnace was at 80° C. At 7.10 the next morning it was at 70° C., and the melting chamber was at 600° C.

On the next day, May 6, with the furnace hot from the previous day's run, a red brass containing 33 per cent copper, 4 per cent tin, 5 per cent lead, and 7 per cent zinc was melted. Using the same transformer ratio as was used at the last (and during most of the run of May 5), the furnace was started at 7.20 a. m., taking 53 kw. As the resistor became heated more power was drawn, till at 8.50 a. m. it had reached 92.5 kw.; from this point it was run at 80 to 85 kw.

Seventy-five pounds of metal (in 25-pound ingots) was charged at the start. No more could be charged till 8.40 a. m., and the whole charge of 202½ pounds (all in about 25-pound ingots) was in at 9.09 a. m.. At 9.20 a. m. the metal was at about 1,050° C., and at 9.36 a. m. it was at 1,200° C., when it was poured. This heat and the later heats are tabulated in Table 14.

TABLE 14.—*Test of Hoskins furnace, No. 70 crucible, on red-brass ingot.*

Heat No.	Time current on.	Current off.	Amount of ingot charged.	Ingot poured.	Metal recovered from skim and spill.	Total.	Net metal loss.	Pouring temperature.	Kw. h. used.
			<i>Pounds.</i>	<i>Pounds.</i>	<i>Pounds.</i>		<i>Pounds.</i>	<i>° C.</i>	
1.....	7.20 a. m.	9.36 a. m.	202.50	198.25	3.25	201.50	1.00	1,200	a 164
2.....	9.45 a. m.	10.40 a. m.	203.00	(b)				1,200	79
3.....	10.53 a. m.	11.45 a. m.	192.25	189.00	2.00	191.00	1.25	1,200	70
4.....	11.55 a. m.	c 12.55 p. m.	196.00	194.50	1.00	195.50	.50	1,200	e 47
5.....	1.04 p. m.	1.50 p. m.	193.75	192.00	1.25	193.25	.50	1,180	c 65
6.....	1.57 p. m.	2.41 p. m.	198.50	188.75	4.25	193.00	.50	1,200	d 48
7.....	2.47 p. m.	3.33 p. m.	192.00	190.00	1.50	191.50	.50	1,180	49
8.....	3.40 p. m.	4.27 p. m.	190.00	187.50	2.00	189.50	.50	1,180	48
Total.....				1,563.00			d 4.75		e 570

a Furnace warm at start from previous day's run.

b Pot too full; metal spilled into furnace in lifting.

c Low power for 20 minutes and none at all for 12 minutes (noon heat) while metal was in crucible. Note reduction in kw. h. used because of use of stored heat, which had to be replaced on next heat.

d Net metal loss on 1,360 pounds, 4.75 pounds or 0.35 per cent.

e Average, 730 kw. h. per ton.

The results on red brass, starting with a warm furnace, are much better than on the run of five heats on yellow brass, starting with a cold furnace. With the furnace hot in the morning from a previous day's run, the power consumption for yellow brass would apparently be about 600 kw. h. per ton. The crucible was in far better shape after 13 heats in the electric furnace than it would have been after 13 heats in a fuel-fired furnace. The efficiency of the No. 70 furnace was only a very little, if any, better than that of the No. 40 furnace, although the larger furnace would be expected to show better results. The relatively better results of the No. 40 furnace are probably due to the use of resistor plates on four sides of the crucible instead of on two only. There was a visible difference in temperature in the No. 70 size between the sides on which the resistors were placed and the other two sides.

FOUNDRY TEST OF NO. 70 HOSKINS FURNACE.

In order to collect information on electric brass melting, the Commonwealth Edison Co., of Chicago, purchased a Hoskins furnace designed to take a No. 70 crucible, installed it, and tested it at the plant of the L. Wolff Manufacturing Co. The test was conducted by Mr. H. M. St. John, of the Commonwealth Edison Co., in 1915, by whose courtesy the following data are made available.

The furnace was in actual operation 40 days, in which it produced 171 heats of about 200 pounds each, or about 17 tons of metal. This is an average of only a little over four heats a day, but does not represent the productive capacity of the furnace, as operation was on an experimental basis.

RELIABILITY.

During the test the furnace had to be shut down eight times for furnace troubles, four times when the breaking of a crucible let the metal into the furnace chamber, twice for failure of cover slabs, once for failure of the magnesite lining, and once for failure of the resistor.

The crucible breakage was apparently due to the use of a No. 80 crucible in a furnace only designed for a No. 70, thus bringing the crucible very close to the resistor. Vertical cracks developed at the top of the crucibles, apparently due to too rapid heating of the parts of the crucible nearest the resistor. The crucibles averaged a life of only 19 heats, against a record of 35 heats claimed for the same crucibles in the oil furnaces of the plant. There was much less oxidation and scaling in the electric than in the oil furnaces; if the cracking could be prevented the crucible life in the electric

furnace should be superior. It seemed probable that by using resistor plates on all four sides of the furnace instead of on two only and by using the No. 70 crucible, for which the furnace was intended, a more even heating of the crucible and a satisfactory crucible life would result.

A larger, more easily opened tap hole in the furnace bottom could readily be installed, and shutdowns thus avoided if a crucible did break in the furnace. The shutdowns due to failure of the cover bricks could be prevented by using a better grade of brick or by keeping spare covers on hand.

The magnesite-brick lining was not a success, as under the intermittent operation of the test it spalled badly after two weeks' service. It was patched up and kept going throughout the test, but it was a cause of continual anxiety. Inasmuch as the Hoskins Manufacturing Co. had very good results from furnaces with carborundum linings in melting nickel-chromium alloys, and hence run under more severe conditions than a brass furnace, it was believed that by replacing the 4½-inch magnesite lining by one made of 2½-inch magnesite and 2½-inch carborundum, with the carborundum inside, the life of the lining could be made satisfactory.

Shutdowns from failure of the resistor or other carbon parts can be prevented by replacing such parts on the Sunday before they would fail. Experience is needed to enable the operator to predict just when failure would occur; with such experience, shutdowns should be infrequent.

While it was impossible to make, during the test, the changes indicated by the test as desirable, there seemed no reason why the furnace should not operate steadily and reliably if the suggested improvements should be made.

EFFICIENCY.

The furnace could not be run at maximum efficiency, as it was placed so far from the molding floor that the metal had to be carried much farther than would be necessary in practice. This meant that the metal had to be heated more than normal and that the loss of time in pouring was much more than should be, probably handicapping the furnace about one heat in a 9-hour day. Operating on yellow brass of 65 per cent copper, 33 per cent zinc, and 2 per cent lead, with the metal poured at 1,100° C., the average results shown in Table 15 were obtained.

TABLE 15.—Results of foundry test of Hoskins furnace, 200-pound heats.

Monday, starting with furnace practically cold.

Heat No.	Time of heat. ^a	Kw. h. used.	Output.
	<i>H. m.</i>		
1.....	2 42	209	<i>Pounds.</i> <i>b</i> 1,200
2.....	1 26	96	
3.....	1 20	91	
4.....	1 10	76	
5.....	1 10	72	
6.....	1 05	66	
Total.....	8 53	610	

Regular operation, no night heating, but furnace hot from previous day's run.

	<i>H. m.</i>		
1.....	2 10	162	<i>Pounds.</i> <i>c</i> 1,400
2.....	1 22	91	
3.....	1 18	79	
4.....	1 10	77	
5.....	1 10	68	
6.....	1 05	62	
7.....	1 00	58	
Total.....	9 15	597	

Saturday (half day).

	<i>H. m.</i>		
1.....	2 10	162	<i>d</i> 600
2.....	1 22	91	
3.....	1 18	79	
Total.....	4 50	332	

^a Actual running time per heat, plus 10 minutes to pour and charge.^b 1,000 kw. h. per ton.^c 875 kw. h. per ton.^d 1,100 kw. h. per ton.

Results of a week's operation.

	Output.	Kw. h. used.
	<i>Pounds.</i>	
Monday.....	1,200	610
Tuesday to Friday.....	5,600	2,388
Saturday.....	600	332
Total.....	7,400	3,330

^a 900 kw. h. per ton.

By using 125 kw. h. during the night (about 8 kw.) to keep the furnace hot, the following results were obtained;

Results of heats.

Heat No.	Time of heat. ^a	Kw. h. used.	Output.
0.....	H. m. (^b)		
1.....	1 44	125	
2.....	1 19	136	
3.....	1 18	91	
4.....	1 06	77	
5.....	1 03	75	
6.....	1 00	71	
7.....	1 00	68	
8.....	1 00	c 55	Pounds.
			d 1,600
Total.....	9 30	749	

^a Actual running time per heat, plus 10 minutes to pour and charge.

^b Night.

^c Power input reduced on last heat, utilizing some stored heat.

^d 950 kw. h. per ton.

Night heating at this rate increases the output by one heat, but raises the energy needed per ton. Eight kw. are insufficient to hold the furnace hot enough over night to affect materially the production rate.

On a 24-hour operation the furnace could produce 200 pounds per hour, or 4,800 pounds per day, at 55 kw. h. per heat, or 550 kw. h. per ton.

About 10 per cent of the power supplied (about 80 kw. on the first couple of heats of a 9-hour day, about 60 kw. thereafter) was lost in the electrode cooling water.

ELECTRICAL CHARACTERISTICS.

Like any purely resistance furnace, the power factor was unity. Under usual operation the voltage and amperage for different rates of power were as follows:

TABLE 16.—*Comparison of current values.*

Kw.	Volts.	Amperes.	Kw.	Volts.	Amperes.
100	50.0	2,000	75	41.7	1,800
90	47.4	1,900	70	41.2	1,700
85	47.2	1,800	65	40.7	1,600
80	45.8	1,750			

By varying the pressure on the plates at any given voltage the power input can be varied 25 per cent and still give good operating conditions. Larger variations could be made, but result in less satisfactory operation, so that voltage control is also necessary.

Where resistor plates are used on four sides of the furnace instead of two, there would be a greater allowable percentage of power regulation by altering the resistor pressure, the voltage would be

increased, the current lowered, and hence the power loss in electrode-cooling water decreased and the efficiency raised somewhat.

LIFE OF CARBON PARTS.

The average working day's life of the various carbon parts was as follows: Carbon resistor plates and graphite back plates, 12 to 14 days; graphite push blocks and electrodes, 24 to 30 days.

METAL LOSS.

Only a few determinations of metal loss were made. On the yellow brass (33 per cent zinc) used, the charge containing 12 per cent oily borings and 18 per cent gates, with some adhering sand, the average net loss, by comparing weight of metal charged and weight of ingot poured (1,100° C.), was 1.1 per cent, no deduction being made for the nonmetal in the charge. By weighing the charge and weighing the crucible full of molten metal before and after pouring in order to get the weight of metal in the pot, a net loss of 0.3 per cent was found on the yellow brass. The latter method, on a few heats of red brass (20 per cent oily boring and 13 per cent gates) gave 0.35 per cent net loss. The net loss on yellow brass in the crucible lift-out oil furnaces ran between 3 and 4 per cent.

The castings made with electrically melted metal were subjected to regular inspection and to hydraulic tests and were reported to be of the same quality as metal from the oil-fired crucible furnaces.

A comparison of melting costs was made for yellow brass, as follows (1915 conditions):

TABLE 17.—*Comparative melting costs with oil and electric crucible lift-out furnaces.*

Oil Furnace.	
44 gallons oil per ton, at 4 cents per gallon.....	\$1. 76
Pumping oil and blowing air.....	. 50
Maintenance.....	. 25
Metal loss 3 per cent, at 10 cents per pound.....	6. 00
Labor.....	1. 10
Crucibles at \$5.25 each, life 35 heats of 200 pounds.....	1. 50
Total	11. 11
Electric Furnace.	
9-hour operation, no night heating:	
Power, 900 kw. h. per ton at 1 cent per kw. h.....	\$9. 00
Maintenance (resistors, linings, etc.).....	1. 00
Labor (same as in oil furnace).....	1. 10
Crucibles (life assumed same as in oil).....	1. 50
Metal loss 1 per cent, at 10 cents per pound.....	2. 00
Total	14. 60

Electric Furnace.—Continued.**24-hour operation:**

Power, 550 kw. h. per ton at 1 cent per kw. h.-----	5. 50
Maintenance -----	1. 00
Labor -----	1. 10
Crucibles -----	1. 50
Metal loss -----	2. 00
Total -----	11. 10

The better working conditions resulting from the use of the electric furnace, due to its great cleanliness, quiet, and coolness, compared to the oil furnaces are not assigned pecuniary valuation in the above comparison, although the advantages were apparent. These factors were assumed to balance the interest and depreciation charges on the electric furnace, which were not included in the calculation. The price of a furnace, taking a No. 70 crucible, with complete auxiliary equipment ready to attach to a 110-volt, 60-cycle current (but not including transformer for stepping down from 2,200, 4,400, or 6,600 volts to 110 volts), on August 1, 1916, was about \$1,500. It is seen that for the electric crucible lift-out furnace to compete with an oil furnace at these figures the electric furnace must be used 24 hours a day and must be developed to equal or surpass the oil furnace as regards crucible life. Even then, at the assumed prices of oil and of electric power, the only opportunity for the electric crucible furnace is its reduction of losses in metal. The higher the value of the metal lost the greater is the chance for the electric furnace.

It is obvious that under the 1915 conditions at the plant the electric crucible furnace was not promising and the furnace was not used after the test, nor was the test continued to try out the improvements suggested.

CADWELL FURNACE.

A crucible lift-out furnace designed by Cadwell,¹⁵ primarily in portable form for melting copper for bonding purposes on an electric railroad, and hence made to utilize current from the trolley wire or third rail, contains one or more small crucibles resting on (but not surrounded by) a bed of granular carbon, to which the granular resistor current is led by two vertical, adjustable electrodes. From these electrodes arcs are drawn to the resistor, so that the crucible is heated by radiation from arcs on two sides and by conduction and radiation from the resistor below.

Further details of the design will be found in the patent. It is stated that two crucibles, each containing 20 pounds of copper, may be brought to 1,375° C. in 20 minutes, but it is not stated whether

¹⁵ Cadwell, . A., U. S. Patent 1,290,902, Jan. 14, 1919; application filed May 1, 1915.

this refers to starting from the cold or to starting with the furnace already up to temperature. No data on power consumption, furnace upkeep, etc., are given.

RENNERFELT CRUCIBLE LIFT-OUT FURNACE.

A recent patent to Rennerfelt⁷⁶ for a multiple-crucible lift-out furnace for melting metals, glass, etc., is of interest, although no data on its operation are available. In this furnace is used the principle of internal heating previously utilized in the triple-crucible furnace tested by the Bureau of Mines. The heating element of the furnace is in the center, and crucibles are placed all around it in a circular furnace or on both sides of it in a rectangular furnace.

The heat is generated by arcs struck from three vertical, adjustable electrodes, in a three-phase furnace, to a bed of granular carbon or graphite held in a dish-shaped carbon or carborundum receptacle in the circular furnace, or in a trough of one of those materials in the rectangular furnace. The presence of the granular resistor material stabilizes the arcs, and some heat is generated by the passage of current through the resistor.

As to the source of heat, then, the furnace would be classed as a combination smothered-arc, granular resistor furnace. While some heat is radiated directly to the crucibles, it is mainly radiated from the arcs and from the resistor to the roof, from which it is reflected downward by the arched or domed roof. The central part of the roof directly over the heating element will receive extremely severe treatment, as is recognized by the inventor, who stipulates that carborundum, zirconia, or similar extremely refractory materials should be used in those parts of the furnace receiving direct radiation. Instead of using a sectional roof, with openings through which to withdraw the crucibles by direct vertical lifting, openings in the sides of the furnace are provided for taking out each crucible as well as for charging it. This detail would make a special pair of tongs with an extension arm necessary for thrusting the tongs into the furnace and also a counterbalance on the other side of the hoist hook, so that the crucibles might be taken out.

From the point of view of thermal efficiency alone this design is probably one of the best that has been suggested for an electric crucible lift-out furnace. Aside from the danger of short life in the roof, the main questions, which could be answered only by experiment, are whether the crucibles heated primarily from one side by a source of high temperature of heat will show a reasonable life and whether the heating from the top would not give rise to the same trouble from top heating on yellow brass that was encountered in

⁷⁶ Rennerfelt, I., U. S. Patent 1,313,834, Aug. 19, 1919; application filed May 12, 1919.

the Bureau of Mines experiments on the Hoskins type furnace 5 in using a tall, cylindrical crucible more strongly heated at the top than at the bottom.

If such difficulty is encountered in the design shown in the Rennerfelt patent, it might be reduced by raising the crucibles higher in the furnace, obtaining thus more direct heating on the sides of the crucibles.

If any alloys are found which it is metallurgically impossible to handle in an electric hearth-type furnace and if an electric crucible lift-out furnace is to be tried, the Rennerfelt idea—worked out with care in lining the furnace with the most refractory materials, in avoidance of excessive top heating, and in convenience in removing the crucibles—seems one of the most promising yet presented. The scheme has been worked out on a hearth-type furnace and will be referred to again under that heading.

GENERAL ELECTRIC CO. CRUCIBLE FURNACE.

The General Electric Co. has recently developed a small single-phase, 50-pound, 40-kw. crucible lift-out furnace in which heat is developed by the principle of contact resistance, which will be described in connection with the General Electric hearth-type brass melting furnaces. This is designed for use in metallurgical laboratories by manufacturing jewelers or by manufacturing concerns so situated that they can not use fuel-fired furnaces, as in a loft building of a large city. In other words, it is designed for users who melt relatively small amounts of nonferrous metals under such conditions that a high-power consumption is immaterial. The furnace is not claimed to be efficient, as its rate of melting copper (heated to 1,300° C.) on 24-hour operation, with the furnace fully heated up, is given as 700 to 800 kw. h. per ton.^{76a}

PFRETZSCHNER FURNACE.

A furnace of which nothing is known beyond the patent specification is described by Pfretzschner.⁷⁷ A crucible rests on one electrode, and another electrode, coated with electrically insulating material for melting materials that are conductors, is located inside the crucible and by a screw device exerts variable pressure on the bottom of the crucible. The bottom of the crucible, and to a certain extent the upper electrode, are supposed to act as resistors. Thus the crucible is heated from the bottom. After the charge is melted the upper electrode is raised and the crucible lifted out. It does not seem possible to operate such a furnace, as the amperage required

^{76a} Anonymous, Muffled-arc electric furnace for nonferrous metals: *Iron Age*, vol. 107, 1921, pp. 985-1047.

⁷⁷ Pfretzschner & Co., German Patent 244,171, class 21h, group 7, Mar. 1, 1912.

would be enormous, the voltage infinitesimal, and the chances very small of generating sufficient heat in the bottom to do any melting without ruining the crucible.

METAL LOSSES IN FUEL-FIRED PRACTICE.

It is quite impossible to make any accurate generalizations as to metal losses in fuel-fired practice. Figure 6 shows the data received

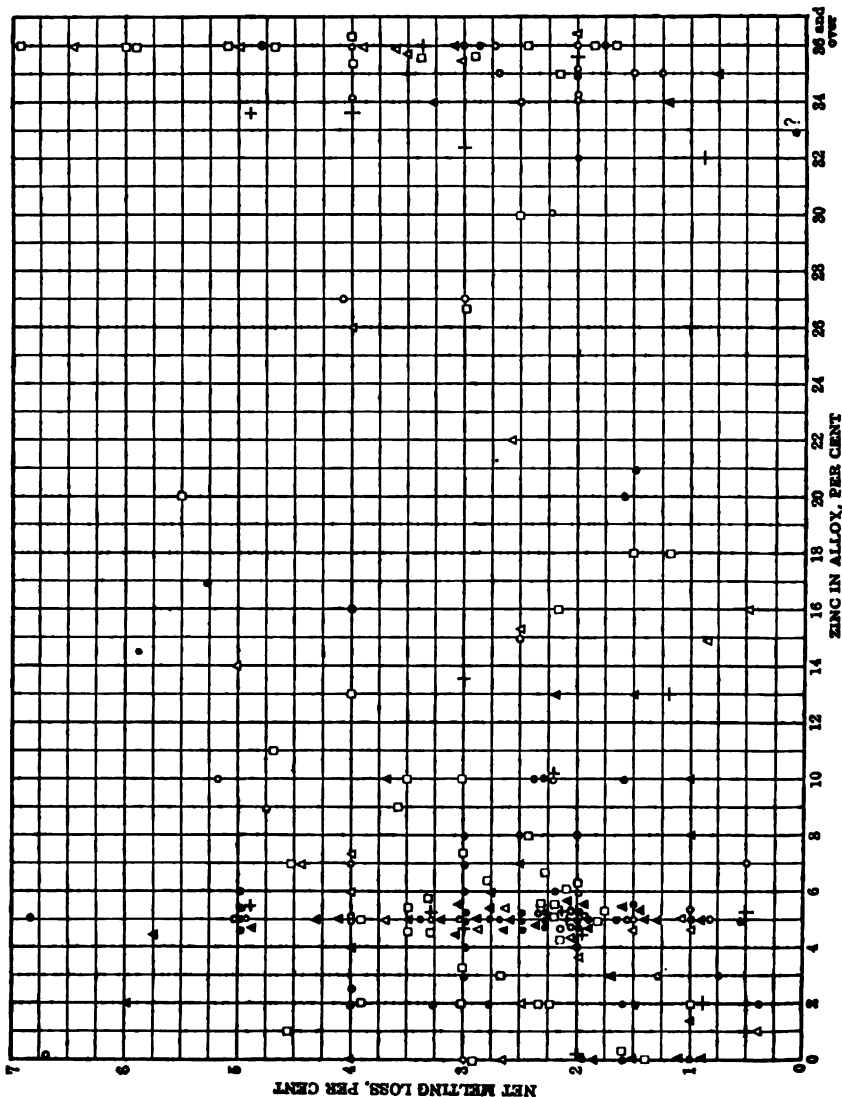


FIGURE 6.—Metal losses in fuel-fired furnace.

on this point in the preparation of Bulletin 73. The amount of fine material in the charge, the amount of nonmetallic material present, the rate at which products of combustion pass over the metal, the

area of metal surface exposed, the atmosphere in the furnace (reducing, neutral, or oxidizing), the time the metal is in the furnace, whether or not charcoal and fluxes are used, and the care taken in the operation of the furnace, as well as the composition of the alloy and the type of furnace, are all variables.

Nevertheless, to draw tentative conclusions, it might be said that from the evidence collected on fuel-fired practice the net metal losses on different alloys would average fairly close to those in Table 18, provided (1) that the type of fuel-fired furnace best adapted to give low metal losses were used for each alloy; (2) that each furnace were well operated; and (3) that suitable allowance were made for the nonmetallic part of the charge and for recovery of metal from ashes, spillings, and skimmings.^{7a}

TABLE 18.—*Metal losses in fuel-fired furnaces.*

Composition of alloy, per cent.				Net metallic loss on new metal or ingot.	Net metallic loss on all borings.
Cu.	Zn.	Sn.	Pb.		
88	2	10		<i>Per cent.</i> 1.0	<i>Per cent.</i> 1.5
85	5	5	5	2.0	2.5
80	10	5	5	2.25	3.0
75	16	3	6	2.75	4.0
65	32		3	3.5	5.0
66	34			2.5	
60	40			4.0	10.0

* Foundry practice. ♢ Rolling-mill practice.

Many plants may show poorer figures; a few may show better, but most managers are content if their average is as good as the data above.

The task set for the electric furnace, then, is to improve these figures. Data to show that the task has been accomplished will follow throughout this report.

METAL LOSSES IN BUREAU OF MINES EXPERIMENTS WITH CRUCIBLE LIFT-OUT FURNACES.

The foundry test on the Hoskins furnace previously cited emphasizes the importance of the reduction of metal losses in melting, as it was only the reduction of these losses that decreased the cost of melting to a point where there was any possibility of competing with fuel-fired furnaces. These losses on both the foundry and factory tests of the No. 70 Hoskins furnace were far lower than in fuel-fired practice. During the Bureau of Mines tests of the smaller Hoskins type of furnaces figures of metal loss were regularly obtained. Only

^{7a} See also Collins, E. F., Melting some nonferrous metals and their alloys in the electric furnace: Table 10, Chem. and Met. Eng., vol. 21, 1919, p. 679; Metal-loss figures in the brass foundry: Met. Ind. (London), vol. 15, 1919, p. 481.

a few of these figures are summarized here, as they merely corroborate the finding of low losses on the larger furnaces, and they did not differ in various sizes or forms of the crucible lift-out furnaces. In the tests of metal loss made by the Bureau of Mines the metal was poured into ingots. When oily borings were used, the amount of oil was determined and its weight deducted from the total weight of the charge.

TABLE 19.—*Metal losses in electric crucible lift-out furnaces.*

Heat, composition of charge, per cent.					Material.	Average pouring temperature.	Weight of metal charged	Weight of ingot poured.	Weight recovered from spillings.	Total weight recovered.	Loss.	Net loss.
	Cu.	Zn.	Sn.	Pb.								
43 to 44....	100				Ingot....	* C. 1,225	Pounds. 73.25	Pounds. 72.80	Pounds. 0.40	Pounds. 73.20	Pounds. 0.05	Per ct. 0.07
45 to 46....	93.5		6.5		Borings..	1,100	96.00	95.85	.15	96.00	.00	.00
47 to 49....	84.4	5.5	4.6	5.4	Oxidized millings passing 30-mesh.	1,120	147.00	144.50	a .50+	145.00	a 2.00	a 1.40
50 to 66....	78.5	16.0	2.5	3.0	Ingot....	1,230	989.75	978.25	5.00	983.25	6.50	.66
98 to 110....	78.5	16.0	2.5	3.0	do.....	1,200	720.00	713.50	1.50	715.00	5.00	.70
111 to 120....	78.5	16.0	2.5	3.0	do.....	1,200	530.00	525.00	1.80	528.80	3.20	.61
67 to 69....	78.5	16.0	2.5	3.0	do.....	1,155	425.75	419.25	4.75+	424.00	a 1.75	a .41
70 to 72....	61.4	34.9		3.7	Borings..	1,005	146.25	140.85	2.40	143.25	3.00	2.04
73 to 76....	62.5	33.8		3.7	Ingot....	1,075	198.15	195.20	0.00	197.20	.95	.48
77 to 83....	63	33.5		3.5	do.....	1,060	870.30	861.25	6.10	867.35	2.95	.34
88 to 90....	58	40.5	.6	(b)	Turnings.	1,015	146.20	136.00	5.00	141.00	5.20	c 3.55

a Not all of spilled metal recovered.

b Also 0.40 per cent Al, 1.50 per cent Fe (manganese bronze); composition approximate—that aimed at in making up the alloy from which the turnings came.

c The ingot from heats 88 to 90 on manganese bronze turnings averaged, on analysis, 57.70 per cent Cu, 39.80 per cent Zn, 0.90 per cent Sn, 0.21 per cent Pb, 1.05 per cent Fe, 0.13 per cent Al, 0.04 per cent Mn. The analysis indicates that the zinc loss in melting will not account for the calculated net loss and make it probable that the turnings carried dirt, which is here counted as metallic loss.

All these data compare very favorably with the figures assumed for fuel-fired practice and show that the higher the alloy is in zinc the greater the reduction in metal losses. This increase is to be expected, since the metal loss on alloys high in zinc is mainly due to volatilization of zinc rather than to oxidation. In fuel-fired furnaces there is always a flow of products of combustion over the metal which continually carries off zinc vapor, whereas in the electric furnace there is no flow of products of combustion and the furnace can be kept tightly closed.

In fact, even if the electric crucible lift-out furnace is not tightly closed at the top, as long as there is no opening in the bottom or sides of the furnace through which air can enter to cause a draft there is no appreciable escape of zinc vapor, as it is heavier than air and lies stagnant in the furnace until the cover is removed and air currents displace it. The metal can be so hot that it releases copious zinc fumes when the crucible is removed, and yet only the slightest escape of zinc fumes, or none at all, can be seen at small openings in the cover of the furnace.

TILTING ELECTRIC CRUCIBLE FURNACES.

Besides the lift-out furnaces described above various tilting crucible furnaces, electrically heated, have been suggested or tried.

STANSFIELD FURNACE.

Stansfield⁷⁹ shows sketches of crucible tilting furnaces for laboratory use in which a crucible, a relative nonconductor of electricity, is embedded in a granular resistor. In the earlier edition of his book he suggests that such a furnace could be made larger than the laboratory furnaces and could be used for brass or steel melting. The suggestion is omitted from the later edition.

EXPERIMENTAL BAILY AND OTHER FURNACES, WITH GRANULAR RESISTOR ABOUT CRUCIBLE.

It is not surprising that many experiments have been made on the use of crucibles embedded in a granular furnace which tilts to pour. When electric power is available for electric furnaces, the simplest method of making in the laboratory a small experimental heat of any ordinary material is to embed a crucible in a granular resistor, heat it, remove the crucible, and pour out the metal. Many such resistance furnaces for laboratories have been described,⁸⁰ and a German furnace of this type (the "Kryptol") has been sold in laboratory sizes.

As the resistor is disarranged by removing the crucible, it is obvious that the furnace should be tilted to discharge the contents of the crucible. The Bureau of Mines, in 1912, tried out some small tilting furnaces of this type. One furnace taking a No. 20 crucible started with a cold furnace and melted 50 pounds of copper in one hour, using 23 kw. h. A second 50 pounds was then melted in 35 minutes, using 14 kw. h. The copper was poured at 1,250° C. This fair-degree efficiency in so small a furnace, compared to the behavior of the first two heats on a No. 20 crucible lift-out furnace, which would take twice as long and use four times the power, shows the

⁷⁹ Stansfield, A., *The electric furnace* 1907, p. 22; 1914, pp. 26, 157.

⁸⁰ Compare Watts, O. P., *An electric furnace for heating crucibles: Electrochem. and Met. Ind.*, vol. 4, 1906, p. 273; Tucker, S. A., *A granular carbon-resistance furnace: Trans. Am. Electrochem. Soc.*, 1907, vol. 11, p. 307; Goodwin, J. H., *Granular carbon-resistance furnace: Met. and Chem. Eng.*, vol. 9, 1911, p. 188; Calhane, D. F., *A convenient furnace: Met. and Chem. Eng.*, vol. 8, 1910, p. 581; Calhane, D. F., and Bard, E. E., *An efficient electric furnace: Met. and Chem. Eng.*, vol. 10, 1912, p. 461; Lohr, J. M., *Tensile strength of copper-zinc alloys: Jour. Phys. Chem.*, vol. 17, 1913, p. 5; Jeffries, Z., *Notes on the gran-annular electric furnace: Met. and Chem. Eng.*, vol. 12, 1914, p. 154; Mitchell, W., U. S. Patent 494,586, Apr. 4, 1893; Potter, H. M., U. S. Patent 875,284, Dec. 31, 1907; Fulton, C. H., and Coursen, W. A., U. S. Patent 1,048,144, Dec. 24, 1912.

advantage of generating heat close to the material to be melted. In another smaller furnace the following results were obtained:

TABLE 20.—*Results on small granular-resistor crucible tilting furnace.*

Heat No.	Time.	Charge.	Material.	Pouring temperature.	Kw. h. used.
	<i>H. m.</i>	<i>Pounds.</i>		<i>° C.</i>	
1.....	1 10	25	Red brass.	1,225	42
2.....	40	25	do.....	1,220	30
3.....	25	25	do.....	1,225	22
4.....	20	28	Copper....	1,180	13½
Total.....	2 35	103		• 1,200	107½

• Average pouring temperature.

After two hours of running, hot spots developed in the resistor, or the crucible failed, or some other mechanical difficulty arose. Unless the crucible is lined or covered with some nonconductor of electricity current is carried by the crucible and its contents. With a loosely packed charge not much current could be carried by the metal in the crucible, so that the current in the furnace, even with a conducting crucible, might not be excessive, but when the metal was melted, unless the nonconducting layer was perfect, the current was short-circuited through the center of the furnace by the charge itself. The main heating was done between crucible and electrodes rather than in the body of the resistor, an arrangement which soon caused hot spots to develop, with resultant burning out of the crucible. When a crucible breaks in such a furnace, it has to be cooled, torn down, and refilled with resistor.

Very small tilting brass furnaces of this type, holding 5 or 10 pounds of metal and using a crucible lined with alundum cement, have been used by the Bureau of Mines, and found useful in such laboratory work as the calibration of pyrometers for use in molten brass and copper up to 1,475° C.

As the size increases the difficulty of maintenance increases disproportionately, and in anything like commercial sizes, run at the necessary temperatures for brass melting, this type becomes impractical.

Various experiments have been made on furnaces of this type. A Baily tilting furnace⁸¹ to hold some 400 pounds of brass was tried out on a laboratory scale and was advertised⁸² for melting brass and aluminum.

It was reported that the National Carbon Co. carried on some experiments on a brass furnace of this type, in the hope of extending the market for resistor carbon. It was also reported in 1913 that the

⁸¹ Data from conversation of Apr. 28, 1913, with R. P. Bicknell, Electric Furnace Co. of America.

⁸² Advertisement, front cover, July, 1913, issue *Electrochem. and Met. Eng.*

General Electric Co. were testing or had designed a furnace of this general type in which the crucible was heated from the bottom, resting on a bed of granular resistor. It is not known whether this design was intended as a tilting or a lift-out furnace. In 1913 also C. E. Bonine attempted to develop a tilting crucible steel furnace for the Hess Steel Castings Co., in which a crucible lined with insulating material was embedded in a granular resistor which used three-phase current. The other furnaces of this type that were tried were probably all single phase. The Bonine furnace was to be applied to brass as well as to steel if successful.

Of all these granular-resistor tilting furnaces only the Baily ever reached the point where it was considered a real possibility for brass melting, and the Baily, in this form, never received commercial test. The work of the Electric Furnace Co. on this form of the Baily furnace gave experience which finally bore fruit in the present commercial form of the Baily furnace, which is like the original only in using a granular resistor and in being a tilting brass furnace.

Nonuniform heating of the resistor, disarrangement of the resistor when the furnace is tilted, necessity for low voltage, high amperage current, poor crucible life, and inability to maintain an electrically insulating coating on the crucible all make it probable that without new refractory materials for crucibles and for insulating coatings of different properties than those now available the crucible furnace with the crucible in direct contact with a granular resistor will prove impractical in commercial sizes and will only be useful in very small sizes for laboratory use.

FOUNDRY TEST OF BAILY TILTING CRUCIBLE FURNACE WITH GRANULAR RESISTOR BELOW CRUCIBLE.

In September, 1914, a Baily tilting furnace built by the Electric Furnace Co. of America was tested at the plant of the Detroit Copper and Brass Rolling Mills. Through the courtesy of the Detroit Copper and Brass Rolling Mills a representative of the Bureau of Mines was permitted to witness the test.

This furnace consisted of a vertical cylinder lined on the bottom and sides with fire brick and having infusorial earth packed between the brick lining and the furnace shell. In the bottom of the cylinder was a horseshoe-shaped resistor trough, rammed up from carborundum fire sand with a water-glass binder, 8 inches wide, 6 inches high, inside, with 2-inch walls. The inner side of the trough was 1 inch away from a 12-inch diameter fire-brick pedestal some 15 inches tall, on top of which rested a shallow dish-shaped crucible with a pouring lip. The crucible was wedged against the furnace walls with brick to hold it in place when the furnace, which was hung on trunnions,

was tilted. The fire-brick roof was dome shaped, coming down within a few inches of the crucible at the outside of the furnace, but leaving space for piling up of the charge in the center. The charging hole was in the center of the roof and closed by a fire-brick cover. The pouring spout of the crucible led into another cut in the furnace wall at front. The spout was plugged with a fire brick. Below this furnace spout was another hole which was also plugged, through which granular resistor (4-mesh chip graphite) could be charged into the resistor trough as it burned away. A third hole below formed a tap hole for drawing off metal should a crucible break.

Two 4 by 4 inch carbon electrodes, carrying flexible leads to allow tilting, entered the back of the furnace and led the single-phase current into the resistor troughs. The 100-kw. transformer had various voltage taps from 80 to 40 volts by 5-volt steps, through which, by a switch, the voltage and hence the power input could be regulated.

The crucible was made of regular fire-clay-graphite mixture, had a capacity of some 700 pounds of metal, and cost, at 1914 prices, \$21.50 each. The shallow, flaring form of the crucible was designed to catch as much heat as possible from the resistor below it by direct radiation and also to allow pouring the metal completely without exceeding the angle of repose of the granular resistor in the open trough below.

Power was thrown onto the cold furnace at 7.15 a. m. At 11.26, when 210 kw. h. had been used, a pyrometer above the empty crucible read $1,160^{\circ}$ C. A charge of 370 pounds of leaded yellow brass and some new copper, lead, and zinc, but mainly scrap brass, was put in at intervals, the last (save the zinc) being charged at 12.20 p. m. The zinc was added at 12.40 p. m., and the metal poured, at about $1,100^{\circ}$ C., at 12.44 p. m., the heat proper having used 112 kw. h. The furnace could not be tilted far enough to drain all the metal from the furnace, and about 100 pounds was retained, the crucible having sunk below its original position on account of the softening and flattening of the fire-brick pedestal. No appreciable zinc fume had been given off during the first heat, but during the pouring vast clouds of it arose through the wedges from the open space between the furnace walls and the crucible. Metal had found its way into the resistor trough. It was supposed that this metal had leaked down between the crucible spout and the furnace spout, the fire-clay luting of this joint having been broken away by the settling of the crucible as the pedestal flattened.

A second 370-pound charge was started at 12.54 p. m., and an attempt at pouring it made at 1.45. The clouds of zinc oxide rolling out of the furnace during this heat had indicated that there was a good

deal of metal down in the resistor, and the current had to be shut off at 1.25 p. m., only 41 kw. h. having been supplied to the furnace on the second heat, the melting of the last 20 minutes before attempting to pour being done by stored heat.

Only about 100 pounds could be poured from the crucible. A tap hole in the bottom of the furnace designed to drain off metal in case of crucible breakage was opened, and some 100 pounds drained from the furnace bottom.

On raising the roof and examining the furnace it was found that the fire-brick pedestal had softened and flattened and the crucible had cracked between its center, which had of course sunk down with the pedestal, and the projecting lip held in the furnace spout. Metal had then dripped down from the crack into the resistor below. The rear wedge between the furnace and wall had been torn away by the dropping of the crucible. The roof brick were dripping somewhat. It is not surprising that the fire-brick pedestal softened. The inner circumference of the resistor, which was in the shape of a horseshoe, was only about $2\frac{1}{2}$ feet, the outer circumference about 4 feet. Naturally the current takes the shorter course and concentrates on the inside of the resistor, next the pedestal. Since the electrical resistance of hot graphite is lower than that of colder graphite, the inner path became progressively easier for the current to follow, and the inner part of the resistor necessarily became hotter and hotter. Fire brick would not stand up at a distance of 1 inch from so hot a resistor.

In a previous test another fire-brick pedestal had been softened in one heat and the crucible slipped down. Had a pedestal of carborundum been used the furnace would probably have stood up long enough to show the efficiency of the design as regards power consumption and metal losses.

The test showed the trouble that would occur whenever a crucible so held above the resistor might break. The crucible itself would have to last 100 heats at the crucible prices then prevailing in order to equal the crucible cost then being obtained in coke fires; the design therefore was abandoned and the Electric Furnace Co. of America turned to the 8-pot crucible lift-out furnace, which had been previously described, and to the design of the early forms of the Baily tilting hearth-type furnace.

WEINTRAUB FURNACE.

As the crucible tilting furnaces of the granular-resistor type have been shown to be impractical, it remains to discuss other forms. Laboratory furnaces of the wire-wound resistor type using platinum or nickel-chromium resistors and commercial furnaces using such

resistors, as the General Electric furnace for heat treatment of forgings for big guns,⁸³ if supplied a constant-voltage current, draw less and less current as the resistor heats up, and utilize less power. Most metals become poorer conductors of electricity as they become heated. To maintain the power supply, the voltage has to be progressively raised. Carbon and graphite, on the other hand, become better conductors, as they heat up; at a constant-voltage supply they draw progressively more current and power. Either type of resistor, then, requires some system of voltage control. Metallic silicon, however, increases in resistance with increase in temperature,⁸⁴ while graphite decreases in resistance. By using a combination of silicon and graphite as a resistor it should be possible to construct a furnace which would require less regulation than most resistance furnaces. Silicon also has the desirable property of acquiring a thin oxide coating in air at high temperature and thereafter not oxidizing rapidly.

Weintraub⁸⁵ designed a crucible tilting furnace to use a cage or grid of silicon, silicon-coated tungsten rods, or silicon-coated graphite rods as resistors, similar to the carbon-rod resistor of Greenwood and Hutton, previously described. A tilting crucible brass furnace using this resistor and holding 125 pounds of metal was constructed at the Lynn works of the General Electric Co. and was reported to show an acceptable figure as to power consumption. The resistor was said to be capable of running steadily at 1,300° C.

Silicon-coated, tungsten-cored rods were not practicable, as they became brittle, evidently from a combination of the tungsten and silicon. Plain silicon rods were tried and could be operated by using a transformer with two taps, one to give the starting voltage, the other the operating voltage.

Graphite rods coated with silicon were made, however, and the current passing through the graphite heated the silicon, starting from the cold; by properly proportioning the core and the coating the resistance was so uniform at different temperatures that only 5 per cent external resistance or 5 per cent voltage regulation was required. The problem of making electrical connection to the silica-coated surface of the silicon was solved (1) by making the elements of the resistor in U or hairpin form, thus halving the number of contacts; (2) by inserting the end of the in a nickel cap filled with a molten alloy of 90 per cent silicon and 10 per cent aluminum, and letting the alloy solidify; (3) by making the con-

⁸³ See Johnson, I. F., *How the power house aids the forge*: Iron Trade Rev., vol. 1919, p. 1221.

⁸⁴ See Anonymous, *Silicon for resistor material*: Elec. World, vol. 55, 1910, p. 409.

⁸⁵ See Weintraub, E., U. S. Patents 1,118,387, Nov. 24, 1914; and 1,134,788, Apr. 6, 1915.

nection through the nickel. The resistor does not change in the course of time, and in small sizes its elements, though rather fragile, could be made fairly strong and tough as compared to the usual brittleness of silicon. Further development would have to be made, according to Dr. Weintraub,^{85a} before a furnace using such a resistor could become a commercial success.

Silicon, unfortunately, melts at about 1,420° C.; even 1,400° C. is a low temperature for the source of heat when a material is to be heated by radiation from the resistor to 1,250° or 1,300° C., as in gun metal or red brass to be poured into very light castings. As Northrup⁸⁶ points out, the efficiency of a resistor running at 1,300° C. is zero for melting a material to be brought to 1,300° C.; other things being equal, the greater the difference in temperature between the source of heat and the material to be heated the greater is the efficiency to be expected, and, conversely, the lower the difference the lower the efficiency.

In crucible resistor furnaces for brass in general the interior of carbon resistors probably runs at much nearer 2,000° C. than 1,300° C. Besides the disadvantage of a low melting point, silicon has another—its brittleness. Casting a large grid in one piece as a resistor would probably be impracticable, and the problem of making stable electrical connections between the parts of a built-up grid would also be difficult. The grid is a desirable form for such a resistor, as it offers great radiating surface, and hence can be run with less danger of melting the inside than could a more massive form.

The silicon-coated graphite resistor should serve for use in a furnace to melt aluminum, possibly yellow brass, and other lower melting alloys, but its limitations as to temperature and its fragility make it of little promise for general use in brass furnaces. No other development of this idea is known.

CRUCIBLE TILTING FURNACES WITH SOLID OR MOLDED RESISTORS.

FOUNDRY TEST OF HELBERGER FURNACE.

One of the early trials of an electric furnace in a brass foundry was that of the Helberger at the works of the National Cash Register Co., at Dayton, Ohio, in January and February, 1914. Through the kindness of this firm a representative of the bureau attended the

^{85a} Weintraub, E., Personal communication, Jan. 12, 1920.

⁸⁶ Northrup, E. F., Production of high temperature and its measurement: Met. Chem. Eng., vol. 17, 1917, p. 686.

earlier part of the test. The Helberger furnace,⁸⁷ which Japing and Krause⁸⁸ say has been in use in German brass foundries, is a furnace with a tall, cylindrical, carbon resistor, held by pressure between a lower carbon plate and an upper carbon ring, serving as electrodes, and surrounded by refractory walls.

A tall, narrow crucible made of material of fairly low electrical conductivity, but resembling the ordinary crucible material, slips snugly inside the carbon resistor. A pouring spout made of perforated sheet iron coated with crucible material fits inside the upper electrode and joins onto the crucible, the joint being luted up. A

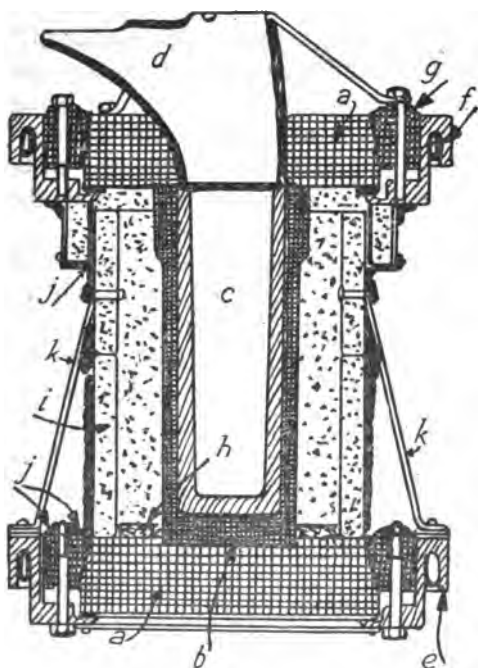


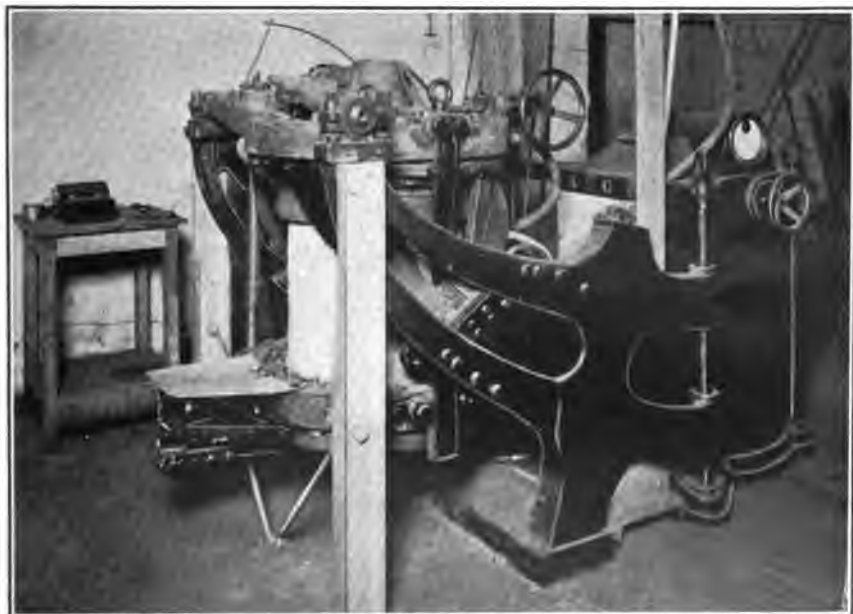
FIGURE 7.—Diagram of Helberger furnace: *a*, electrodes; *b*, resistor; *c*, crucible; *d*, pouring spout; *e*, cast-iron ring; *f*, water chamber for cooling; *g*, wedge, graphite or cast iron; *h*, granular magnesite; *i*, fire-clay cylinder; *j*, asbestos rope; *k*, brace.

heavy current is passed through the carbon resistor at a low voltage—between 4,000 and 8,000 amperes at from $6\frac{1}{2}$ to $11\frac{1}{2}$ volts. To avoid losses of lead in carrying so heavy a current, the 440-volt, 100-kw. variable-voltage transformer is made an integral part of the furnace, and no flexible secondary leads are provided, but very large laminated copper contacts or brushes under pressure allow breaking the circuit when the furnace is tilted. The furnace tilts about the lip.

The pressure between the electrodes is regulated by a handwheel, which actuates three screws, each one of which can also be independently regulated to obtain uniform pressure around the circumference.

⁸⁷ See Helberger, H., U. S. Patents 558,357, Apr. 14, 1896; 769,250, Sept. 6, 1904; 932,986, Aug. 31, 1909; and 1,023,309, Feb. 20, 1912; Anonymous, The Helberger furnace: *Metal Ind.*, vol. 9, 1911, p. 212; Anonymous, Electric transformer crucible furnace: *Met. Chem. Eng.*, vol. 10, 1912, p. 501; Helberger, H., German Patents 228,918 of Jan. 14, 1908, class 21h, group 11; 238,486 of Dec. 9, 1910; 246,083 of Sept. 2, 1910; Anonymous, Brass World, vol. 7, 1911, p. 79; Anonymous, *Metal Ind.*, vol. 9, 1911, p. 212; Harden, J., The Paragon electric furnace and recent developments in metallurgy: *Trans. Faraday Soc.*, vol. 7, 1912, p. 188; von Schwarz, M., *Int. Zeit. Metallog.*, vol. 2, 1912, p. 258; Perkins, F. C., German crucible electric furnace: *Chem. Eng.*, vol. 18, 1913, p. 97.

⁸⁸ Japing, E., and Krause, H., *Kupfer und Messing*: 1912, p. 93.



HELBERGER CRUCIBLE TILTING FURNACE.



POURING METAL FROM HELBERGER FURNACE.

The furnace appears to have been used in Germany in very small sizes for such purposes as jewelers' use. The makers of this furnace, the German Allegemeine Electricitats Gesellschaft, advertised it in a 100-kg. (220-pound) capacity. The National Cash Register Co. ordered one in August, 1912, and when it was received a year and a half later no tilting device was supplied with it. A counterbalanced tilting device was designed and installed by the National Cash Register Co. The furnace and transformer together weighed 9,000 pounds, though its capacity was but about 200 pounds, and cost \$6,000 without the tilting device.

It is very likely that the furnace, though listed in this size, had not hitherto been made in as large a size and was built to fill the order of the National Cash Register Co. The furnace is illustrated in figure 7 and Plates IV and V. The test at the National Cash Register Co. has been fully described by Dorsey.²²

The first German crucible failed after six heats; the crucible was perforated and the metal ran down between crucible and resistor. The next crucible lasted 21 heats, during which the following figures were obtained on red brass; the furnace, having been operated until quitting time the previous day, was warm on the morning of the test.

TABLE 21.—*Test of Helberger furnace on Feb. 4, 1914.*

Heat No.	Time.	Amount of metal.	Temperature.	Kw. h.	Kw. h. per 100 pounds, average.
	<i>H. m.</i>	<i>Pounds.</i>	<i>°C.</i>		
15.....	2 23	150	1,200	120	
16.....	49	150	1,270	55	
17.....	40	150	1,200	51	
18.....	30	150	1,200	39	
19.....	32	150	1,200	50	
20.....	30	180	1,270	42	
21.....	42	165	1,200	40	
22.....	34	150	1,270	55	
23.....	31	150	1,200	50	
24.....	36	150	1,200	48	35.6
Total.....	7 47	1,545		550	

After four more heats the crucible failed, and the resistor and upper electrode also had to be replaced.

A new crucible, new resistor, and new top plate, all from the makers, were installed and 18 successive heats were made, the furnace having been brought up to running temperature by heating it empty on the afternoon of February 22, 1914.

²² Dorsey, H. G., Tests on electric furnaces for brass foundries: Trans. Am. Inst. Metals, vol. 8, 1914, p. 246.

TABLE 22.—Further tests of Helberger furnace on February 23 and 24, 1914.

Heat No.	Time.	Amount of metal.	Temperature.	Kw. h.	Kw. h. per 100 pounds, day average.
	<i>H. m.</i>	<i>Pounds.</i>	<i>° C.</i>		
22.....	2 39	150	1,175	139	
33.....	40	150	1,200	67	
34.....	32	150	1,175	56	
35.....	41	150	1,175	50	
36.....	30	150	1,200	46	
37.....	1 33	150	1,175	44	
38.....	44	175	1,175	80	
39.....	22	150	1,240	51	
40.....	20	150	1,200	38	41.5
Total.....	8 1	1,375		571	
41.....	2 34	150	1,175	175	
42.....	40	150	1,200	67	
43.....	32	155	1,175	62	
44.....	30	150	1,240	60	
45.....	31	150	1,175	33	
46.....	1 10	150	1,175	28	
47.....	37	170	1,240	72	
48.....	32	170	1,175	51	
49.....	30	156	1,200	45	42.2
Total.....	7 36	1,401		593	

After these 18 heats the third crucible wore out. The average of the three all-day runs was 1,440 pounds melted in $7\frac{1}{2}$ hours, using 571 kw. h., or 800 kw. h. per ton. On 24-hour operation the furnace would melt at about 600 kw. h. per ton. The average life on three crucibles was 15 heats. As no more German crucibles were at hand, attempts were made to ram up a lining to the resistor of carborundum fire clay; a maximum of six heats was thus obtained. The pouring spout was flimsy, became badly damaged during the test, and required frequent patching.

There was no convenient way to close the top of the furnace tightly to prevent volatilization and oxidation, so a layer of charcoal was put on after the metal had melted. The crucible was so tall and narrow—7 inches in diameter by 25 inches high (not including the pouring spout or hood)—that it would hold but little bulky scrap, and the operator had to stand on the little step in front of the crucible and feed in the charge continually.

Only one determination on metal lost in melting was made, this being on one of the earliest heats. Two hundred pounds (75 per cent gates, 25 per cent new metal) of an alloy of 89 per cent copper, 3 per cent zinc, 5 per cent tin, and 3 per cent lead was charged and 199 pounds of ingot was poured, a metal loss of 0.50 per cent.

The low crucible life and the great amount of attention demanded from the operator in charging, which would require one man per furnace, caused the furnace to be discarded by the National Cash Register Co. When a crucible failed during a heat and metal ran

down between crucible and resistor, it picked up metallic silicon and aluminum reduced (in the presence of another metal to collect the aluminum) by the carbon of the resistor from the fire-clay bond of the crucible. If any of this metal contaminated with silicon and aluminum poured out with bulk of the charge or diffused back into the main charge, the metal was ruined for the purposes of the National Cash Register Co.

A smaller Helberger furnace was tried out at the New York assay office in 1913 for melting precious metals, but gave a great deal of trouble and was discarded.

An 80-kw. Helberger was tried out for melting steel about the same time by the Hess Steel Castings Co., of Bridgeton, N. J., and is reported to have given much trouble and to have been discarded also. All in all, the Helberger furnace was complex and costly and seems to be quite impracticable for use under American foundry conditions.

ELECTRO-METALS EXPERIMENTAL FURNACE.

Electro-Metals, Ltd., of London, England, according to Boving,⁹⁰ experimented with a brass furnace in which the crucible, made to have high electrical-resistance, is used as the resistor. The furnace gave trouble, especially in the contacts between crucible and the upper and lower electrodes, so that no real success was achieved.

MORGAN FURNACE.

The Morgan Crucible Co., of Battersea, London, has advertised "Morgan's patent electrically heated crucible," which is used both as resistor and metal container. This is designed in two styles,⁹¹ one very much like the Helberger, with a crucible held between water-cooled upper and lower electrodes, which serves as resistor and metal container as well, being coated inside with an electrically nonconducting coating. It is said that fine adjustment of the resistance is obtained by varying the pressure between the resistor and the terminals. It would appear to be a very dangerous procedure to lighten this pressure enough to alter the resistance, thus obtaining a contact with an appreciable drop in voltage absorbing power, and causing local heating at the contacts. The vertical tilting crucible type is designed in two sizes, one to hold 20 pounds and one to hold 150 to 200 pounds of brass.

⁹⁰ Boving, J. O., Letter of July 3, 1918, to the director of the Bureau of Mines.

⁹¹ Anonymous, Electric furnace developments for nonferrous metals: *Metal Ind.* (London), vol. 14, 1919, p. 444; *Metal Ind.* (N. Y.), vol. 19, 1919, p. 424; Speirs, C. W., *British Patents*, 24,626 of 1912, 162,246 of Oct. 7, 1919; U. S. Patents 1,335,079, Mar. 30, 1920; and 1,366,135, Jan. 18, 1921; Discussion: *Jour. Inst. Metals* (British), vol. 17, 1917, p. 277; compare von Zeerleder, A., U. S. Patent 1,367,442. Compare also Soucni, C., U. S. Patent 1,375,615, Mar. 3, 1921.

A drain spout in the bottom of the refractory inclosure about the resistor crucible acts as a safety outlet in failure of the crucible. In view of the short life of such a crucible, indicated by the Helberger test, this outlet seems a necessary provision.

In larger sizes the furnace has a shallow trough (fig. 8) for single-phase current, or a dish for three-phase (fig. 9), provided, respectively, with lugs at two or three points, by which pressure contact is made with the water-cooled terminals. A spout is molded in the trough or dish for pouring. The trough or dish which forms the resistor of the tilting furnace is lined with an electrically non-conducting layer, within which the metal rests. The Morgan Crucible Co. advertised to supply this type in any size up to a capacity of 1 ton of copper or brass. It is claimed that on the second heat of a day 800 pounds of 65:35 brass is melted in two hours with 120 kw. h., or at the rate of 300 kw. h. per ton, and that in regular operation (it is

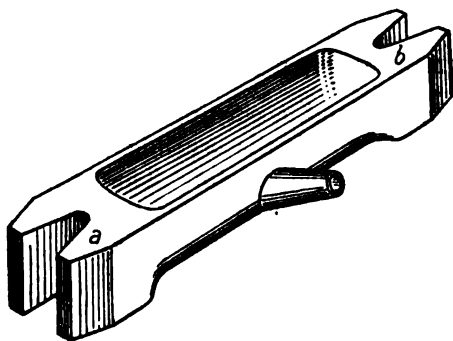


FIGURE 8.—Detail of Morgan furnace design, single phase.

not specified whether on a 9 or 10 or a 24 hour basis) the power consumption averages 300 kw. h. per ton. It is said that $1,300^{\circ}\text{C}$. is the limit to which metal can be safely heated in the furnace.

The crucible is said to be completely protected from mechanical damage and thus the life is long, but no figures on the actual life are given.

Such a resistor-crucible furnace would be thermally efficient while the resistor lasted, as the heat is generated so close to the metal. The resistor would take a low voltage and a high amperage and would involve large heat losses in the water-cooled terminals. Its reliability would depend on two factors—the uniformity of the molded fire-clay (or similar bond) graphite resistor and the impregnability of the electrically nonconducting lining.

If the resistor is not uniform, so that local heating gets started at any point, a hot spot will surely develop and failure ensue by melting or vaporizing the bond, since a graphite resistor will continue to heat up progressively in any spot that becomes locally overheated, due to the fall in resistance as the material heats up. This fall in resistance in any spot offers an easier path to the current, which tends to concentrate in the hot spot, heating it up still more, and the vicious circle goes on till a temperature is reached at which the material fails.

Impregnability of the electrically nonconducting lining is close to impossible to attain with material now available. If the current in

the single-phase trough shown in figure 8 fails to pass through the walls of the trough about the cavity for holding metal and by reason of a crack in the insulating lining at both ends follows the path of lowest resistance through the molten metal in the trough, the voltage drop through the metal would be small compared to the total, and practically all of the resistance would be at the ends *a* and *b*. When this occurs, unless the rise in amperage is at once noted, the charge poured immediately (if it is hot enough to pour), the furnace cooled, and the insulating lining patched or replaced, practically all the power will be absorbed in the ends *a* and *b* and the temperature must rise till the ends of the resistor fail. If the increase in current on failure of the insulating lining is prevented by fuses or circuit breakers, so that there is no further heating, it is likely that the charge may be too cold to pour and must be left to freeze in place, with every chance of its removal ruining the resistor.

There are few materials, even though good electrical insulators at low temperatures, which do not become more or less conducting at 1,000° to 1,300° C., the temperatures of a brass furnace. One of the best high temperature insulators for electricity is alumina, and alundum cement is the best material for use as the insulating coating.

This coating would have to be thick enough that its resistance is much higher than that of the resistor trough, or the current would divide, part passing through the resistor and part, from *a* through the coating into the molten metal, out through the coating at the other end, into *b*.

If even two small cracks should allow the metal to reach the resistor, short-circuiting would begin. When it is considered that the

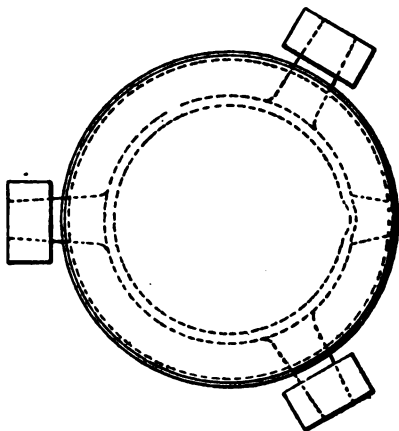
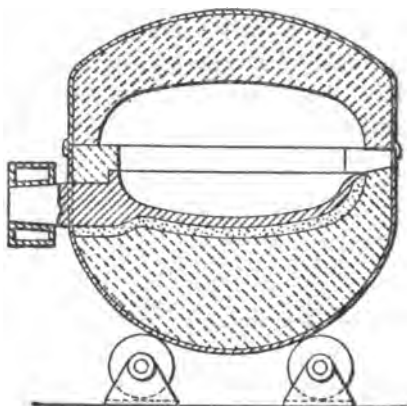


FIGURE 9.—Detail of Morgan furnace, three phase.

coefficients of expansion of the coating and of the resistor may not be and probably are not the same, it is obvious that thermal expansion would itself tend to crack the coating. Added to this danger is that of cracking the coating when a heavy ingot is dropped accidentally into the trough.

If this type has been developed to the extent that the resistor and coating in an 800-pound or a 1-ton furnace show a life long enough to make the type of any commercial value, unusual judgment must have been exercised in the selection of materials, and supremely skillful workmanship must have been used in molding the resistor

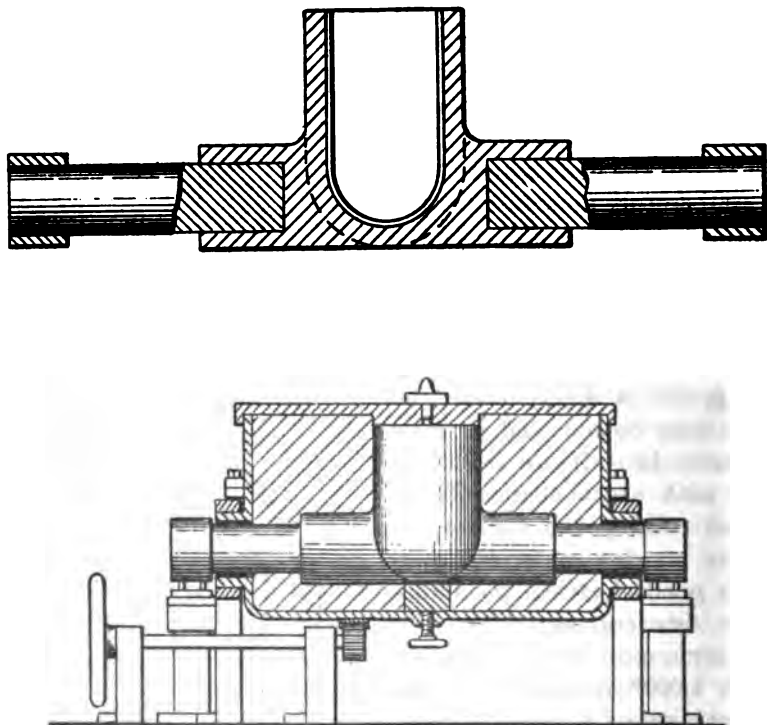


FIGURE 10.—Conley crucible tilting furnace.

and in applying the coating. No final conclusions on this type can be drawn until it has been operated under foundry conditions. Although the furnace was advertised as ready for use a couple of years ago on foundry test has yet been recorded, and the advertisements have not appeared in late issues of the periodicals formerly carrying them.

CONLEY FURNACE.

The Conley²² furnace is another resistor furnace (see fig. 10) in which the metal is contained in a crucible integral with the resistor,

²² Conley, M. R., U. S. Patents 789,250, Sept. 6, 1904 (application filed Oct. 1, 1902), 989,568, Apr. 18, 1911, and 1,023,996, Apr. 23, 1912.

and a lining of electrically nonconducting material is provided. The furnace is arranged to tilt for pouring. Experiments are said to have been made as early as 1897.⁹² In 1910 this furnace was tested at the Electrical Testing Laboratories, New York, and was demonstrated to representatives from rolling mills and foundries. The foundrymen did not consider it suitable for practical foundry use.

Some figures on the 1910 tests were submitted by Florence and Hampton, who control the Conley patents. According to these figures the furnace, which had a capacity of 50 to 70 pounds, ran at 14 to 22 volts, taking 300 to 400 amperes when started cold and 1,200 to 1,300 amperes when fully hot. The data are summarized in Table 23.

TABLE 23.—1910 tests on experimental Conley furnace.

Heat No.	Probable state of furnace at start.	Amount charged.	Time.	Kw. h. used.	Time per 100 pounds.	Kw. h. per 100 pounds.	Kw. h. per ton.
		Pounds.	Min.		Min.		
1.....	Cold.....	50	55	17.41	110	34.8	696
2.....	Hot.....	50	23	8.65	46	17.3	346
3.....	Cold.....	68½	75	20.82	110	31.6	632
4.....	Hot.....	68½	39	15.62	67	22.8	456
5.....	Cold.....	70	70	20.44	100	30.0	600
6.....	Hot.....	55	55	17.67	100	32.2	644
7.....	do.....	72	56	17.41	78	31.2	624
8.....	do.....	20	25	4.81	125	24.0	480
9.....	Cold.....	45	50	19.13	111	38.3	766
10.....	do.....	53	53	(b)	100		
11.....	Hot.....	51	30	(b)	59		
12.....	Cold.....	50	90	21.5	180	43.0	860
13.....	Hot.....	52	60	17.3	116	28.9	598
14.....	do.....	100	70	24.3	70	24.3	486
15.....	do.....	35	40	10.34	114	25.9	518
Total.....		840		215.40			

^a This charge consisted of scrap copper and some scrap brass. All the other charges consisted of copper only.

^b Not given.

In a report from another source the 70-pound furnace is said to have melted two heats of 2:1 brass, the first heat taking 37 minutes (temperature of furnace at start not stated) and the second 27 minutes. The metal loss was said to be 0.15 per cent and the power consumption at the rate of 363 kw. h. per ton, no statement being made as to whether this refers to the second heat only or to both. No figures are given on life of the resistor crucible on the 70-pound size.

The resistor crucible of a furnace of 20 pounds capacity is said to have lasted 45 heats.

Miller^{93a} quotes the inventor of the furnace as claiming that this type can be built up to 1,200-pounds capacity, capable of melting copper at 240 kw. h. per ton, and giving a metal loss (alloy not

⁹² Electric-furnace development for metals: Metal Ind., vol. 17, 1919, p. 927.

^{93a} Miller, D. D., The electric furnace as a medium for heating nonferrous metals; Jour. Am. Inst. Metals, vol. 11, 1917, p. 275.

stated) of less than 0.20 per cent. There was a report in 1917 that the furnace was to be developed for use in an American plant producing shrapnel disks on a French order, but no installation of it is known. It is quite certain that the furnace was never actually built in a 1,200-pound size.

The comments made on the Morgan furnace apply also to the Conley.

GUTHRIE AND KARCH FURNACE.

Guthrie and Karch⁹⁴ patented a similar tilting furnace. In one form, instead of a solid resistor forming the crucible, each electrode was so shaped as to half inclose a separate crucible, granular carbon being placed between the electrodes and the crucible. Arcs were to be run between the electrodes and the granular carbon, and the heat was thus to be generated by a combination of both arc and resistance.

No actual construction of such a furnace is known, nor was it suggested especially for brass. The crucible life in such a furnace would probably be extremely short.

BARIO FURNACE.

Still another furnace of this general type, made by the Bario Metal Corp., has been advertised. The Bario furnace consists merely of a carbon electrode in which a trough is gouged out to contain metal. The electrode is held in water-cooled terminals, and an extremely heavy current is passed through the electrode, which serves as a solid resistor. No attempt is made at putting an insulating coating inside the trough. The resistor is in the open air.⁹⁵ The furnace is said to have a capacity of 1 to 20 pounds. The electrode, resistor, or crucible, whichever it be termed, is said to last 25 to 50 heats.

In another form⁹⁶ the resistor is placed inside a furnace chamber and radiates heat onto an ordinary crucible placed below it, thus forming a laboratory type of crucible lift-out furnace.

The first form was priced, in January, 1918, at \$1,200, the second form at \$1,400 (list prices), both complete with variable-voltage transformers.

This firm advertises to build electric furnaces to contain from 1 pound to 3 gross tons of metal. No description of the larger furnaces is given in the advertisements, but from correspondence with the makers it appears that arc furnaces are planned for the larger sizes.

Such a furnace might find limited application for some types of laboratory use, and forms somewhat similar in the principle of the

⁹⁴ Guthrie, B., and Karch, J. P., U. S. Patents 979,663, Dec. 27, 1910; and 1,086,164, Feb. 3, 1914.

⁹⁵ De Milles, P., U. S. Patent 1,317,863, Oct. 7, 1919.

⁹⁶ De Milles, P., U. S. Patent 1,317,864, Oct. 7, 1919.

resistor are patented by Pfanstiehl⁹⁷ and Bristol and Johnson⁹⁸ as a laboratory device for the commercial melting of very small quantities of precious metals or raw materials.

These devices are, of course, applicable only to small-scale or laboratory work and have no bearing on the problem of commercial brass melting.

Fitzgerald and Moyer⁹⁹ have described a laboratory furnace which can be used to heat a crucible by radiation from an arc over it or to heat one embedded in granular carbon.

REED FURNACE.

Reed¹ has devised a crucible tilting furnace primarily designed for steel melting, in which is used a carbon resistor grid similar to the Greenwood and Hutton resistor previously mentioned, but instead of radiating heat to the crucible through an air space, as in the Greenwood and Hutton furnace, and leaving the resistor open to oxidation by air, the space between resistor and crucible is filled with molten glass; the composition of the glass is chosen to give the correct fluidity for the temperature to which the contents of the crucible are to be heated. The glass covers the resistor, prevents oxidation, and also conveys heat from resistor to crucible by conduction.

The crucible rests on a slotted pedestal and is held in place at the top of a refractory cap, which also serves to retain the glass when the furnace is tilted to pour.

The furnace is in the early stages of experimental development and no data have yet been obtained on its power consumption, crucible life, and resistor life under operating conditions. It is claimed that the furnace can be shut down, the glass allowed to freeze, and the furnace be started up again without damage to the resistor; that the resistor, crucible, and refractory container for the glass are not attacked by the glass, and that the electrical resistance of suitable glass is so high that the current flows only through the resistor.

The furnace is not primarily designed for brass melting, but if the claims of the inventor are substantiated by actual test it might be adapted for such use. No active development work is being done on this furnace at present, but such development is planned for the near future both on steel and brass.

⁹⁷ Pfanstiehl, C. A., U. S. Patent 1,283,285, Oct. 29, 1918.

⁹⁸ Bristol, W. H., and Johnson, M. J., U. S. Patent 1,323,576, Dec. 2, 1919.

⁹⁹ Fitzgerald, F. A. J., and Moyer, G. C., Electric furnace for experimental work: Trans. Am. Electrochem. Soc., vol. 36, 1919 (not yet published); Chem. and Met. Eng., vol. 21, 1919, p. 769.

¹ Reed, R. R., U. S. Patent 1,357,901, Nov. 2, 1920.

NORTHRUP FURNACE

There is one furnace which can be made in a crucible lift-out form as well as in tilting form which is in successful commercial use. This is of an entirely different type from furnaces previously considered, as it uses high-frequency current and is a form of induction furnace, the heat being generated in the metal itself or in the crucible containing the metal and not outside the crucible. This exception, therefore, will be described under induction furnaces.

HEARTH FURNACES USING NO CRUCIBLES.**GENERAL CONSIDERATIONS.**

Out of all the many types, forms, and sizes of electric crucible lift-out and crucible tilting furnaces suggested and of the fairly large number tried in the laboratory or in the foundry none has had or offers any real degree of commercial success, with the exception of the one radically different type, the Ajax-Northrup, which will be described later.

The faults of the crucible furnaces in general are small capacity, consequent high labor cost, low production, expense for crucibles, lack of reliability due to fragility of crucibles and insulating linings, and low thermal efficiency, which means a high cost for electric power. All the usual crucible furnaces of the lift-out type use too much electrical energy per ton. Furnaces of the tilting type—with a resistor forming or in contact with the crucible—that might give better results as to amount of energy used per ton, are as a class unreliable because of the inevitably short life of the resistor crucible or insulating lining.

The advantages of crucible furnaces lie in the production of crucible-quality metal and in reducing the metal losses. As these advantages can be obtained in the hearth-type electric furnace without the disadvantages of the crucible type, the obvious next step is to turn to the types that do not use crucibles and that hold the metal in a hearth, discharging it by tilting or tapping.

Vickers² argues against the use of a hearth furnace made of "silicious materials such as bricks, carborundum, etc., which form chemical compounds with the constituents of some alloys, to their great detriment." He does not elaborate the exact cases to which he has reference. Such cases doubtless exist, and there is good reason to consider, in choosing a hearth lining, whether or not it may injure the metal; but the fact that all the electric brass furnaces that are in successful commercial use are of the hearth type is sufficient proof that for many alloys, at least, hearth materials can be found which are

² Vickers, C., *The electric furnace as a copper-melting medium*: Foundry, vol. 45, 1917, p. 280.

not detrimental to those alloys. It is important, however, to make sure that the design of the furnace is not such as to allow the reduction of metallic silicon and its introduction into the bath of brass.

FUNDAMENTAL FEATURES IN THE DESIGN OF USEFUL HEARTH-TYPE FURNACES.

In considering the design of hearth-type electric furnaces for commercial use there are several factors to be considered. Rodenhauser^{*} has attempted to set forth the requirements of an ideal furnace for melting and refining steel and to show that these are best filled by an induction furnace. Commercial practice has not borne out his argument, as induction furnaces for steel are rarities in the United States, but it is useful nevertheless to follow Rodenhauser's example in pointing out some fundamental requirements of an electric furnace for brass, remembering always that the ideal furnace for one set of melting conditions is not necessarily perfect for another set. These fundamental requirements may be discussed under the topics, metallurgical fitness, reliability, rate of production, thermal efficiency, and electrical characteristics.

METALLURGICAL FITNESS FOR THE WORK.

The prime requisite of a furnace is metallurgical fitness for the work to be done. Unless the quality and uniformity of product desired can be obtained the furnace is of no use. If ladle pouring will not give a satisfactory product, for example, the furnace must be capable of pouring the metal direct into molds. If a given type of furnace will not operate on a given alloy without causing too great metal losses, as a direct-arc furnace on yellow brass, it is barred from consideration unless some abnormal condition makes this loss acceptable on account of a greater saving.

For melting a variety of alloys no furnace can be used from which one charge can not be removed completely enough to prevent injurious contamination of a charge of a widely dissimilar alloy to follow. Versatility, or the ability to melt all the alloys handled by a plant, is a desirable feature of a furnace, and where the output of a plant will not allow the allocation of separate furnaces to separate types of alloys, this feature may become a necessity. Another sort of versatility that is desirable in a furnace is the ability to work economically on either 8, 16, or 24 hour operation per day.

If close temperature limits must be held lest the product suffer, a furnace incapable of close, prompt, temperature control would be either useless or at a disadvantage.

^{*}Rodenhauser, W., and Schosnawa, J., translated by vom Baur, C. H., *Electric furnaces in the iron and steel industry*, 1917 ed., p. 66.

RELIABILITY.

The furnace must be reliable and must stand up to its work day after day. St. John,⁴ in discussing this subject, has well expressed the situation when he says: "Sustained mediocrity is preferable to intermittent perfection."

Ruggedness is an essential. Reliability is to be attained by rugged construction, by the use of high-grade and suitably chosen refractories, and by the avoidance of complicated construction of heated parts. One can determine with a reasonable degree of accuracy the behavior as to power consumption and as to metal losses of a given furnace type by laboratory tests on a furnace of comparatively flimsy construction; but none but a robustly built furnace, can stand up under constant use in rapid production in foundry conditions.

It is not vital to successful operation that all the appurtenances of an electric furnace be of simple construction or foolproof if they are not placed where they become hot, but are placed where they can be attended to. The more simple and foolproof they can be made the better, of course, but electric furnaces can operate profitably with complicated and delicate appurtenances, as is proven by the operation, for many years, before more simple and less delicate controls were devised, of steel furnaces with complicated electrode-regulating mechanisms that had to be protected by glass cases and that required the almost constant attention of a high-class electrician.

But when delicate or "fussy" construction is attempted in the heated interior of the furnace results are likely to be disastrous. Complicated, fragile resistors, projections in the refractory lining, or anything in the heated zone that can not be readily replaced while the furnace is hot are highly detrimental to reliability. Simplicity and durability of construction inside the furnace are of paramount importance. For example, the tilting furnaces in which a molded fire-clay graphite resistor serves also as crucible or hearth by the interposition of an insulating coating have low reliability, because the coating is almost necessarily fragile, and its failure usually means that of the whole resistor; when that failure occurs the furnace has to be cooled and practically rebuilt.

Similarly, a furnace with a resistor built of a lattice or grid of carbon rods or bars is not likely to be reliable, as the failure of any part of the resistor means a complete shutdown. Arc furnaces have much to commend them, as a broken or damaged electrode can be replaced with but little delay while the furnace is hot.

⁴ St. John, H. M., Commercial testing of metallurgical electric furnaces: Chem. and Met. Eng., vol. 21, 1919, p. 378.

The selection of refractories is of utmost importance in attaining reliability. The ideal combination of extreme refractoriness, mechanical strength, great resistance to heat flow, or low thermal conductivity, and cheapness is not attainable in any one refractory. The way to obtain the best results seems to lie in the use of three layers. The inner layer, forming the working chamber of the furnace, should be built of materials chosen primarily for refractoriness, strength, freedom from spalling under changes of temperature, and for ability to withstand such conditions of the furnace as a strongly reducing atmosphere or as contact with a given alloy, when used in the hearth, without being damaged by such contact and without damaging the alloy. Low thermal conductivity and low cost of this layer are secondary to the considerations cited above. The intermediate layer should be of good refractoriness and strength, with the lowest thermal conductivity consistent with these qualities. Cheapness becomes a factor in choosing this layer. The outer layer should be chosen primarily as a heat insulator against the rather low temperature of the outside of the intermediate layer.

Sometimes two layers only prove satisfactory, in which case the outer layer must combine the properties of the intermediate and the outer layers of a three-layer lining.

There are no grave difficulties in obtaining highly satisfactory materials for all but the inner layer, the choice of which is limited by the materials available and represents a balance between initial cost and life. Depending on conditions, it may be cheaper to use expensive materials and to avoid the labor and interest charges and loss of production due to shutting down more frequently for relining or it may be better to use cheaper materials and to reline more often.

RATE OF PRODUCTION.

Next in order of importance comes the rate of production. A high rate of production cuts down overhead, labor, and interest charges and is essential to justify the high initial cost of an electric furnace, as this initial cost is necessarily far above that of a fuel-fired furnace. A high rate may be obtained by a design of high thermal efficiency, but no less by other factors of design. A furnace should be capable of rapid charging and discharging; one that can be charged from overhead, by dumping of the charge into it, or that is capable of mechanical charging in other ways is preferable to one in which slow hand charging, as through a small side door, must be done. If slagging or stirring is necessary on a given charge, the furnace must allow easy access to all parts of the bath for these purposes. Even so small a matter as a difference of half a minute in the speed with which a charging door, pouring spout, or tap hole may be

opened or closed may affect materially the rate of production, since these operations are repeated many times.

THERMAL EFFICIENCY.

A high thermal efficiency in total kw. h. used per ton of product is of importance, because electric energy is a more expensive source of heat than the combustion of fuel. For a given amount of power used a high thermal efficiency also means a higher rate of production.

Thermal efficiency is to be attained in the following ways: (1) By generating the heat in or as near to the charge as possible; (2) by supplying power at the highest practicable rate of power input; (3) by the avoidance of waste space inside the furnace, thus making the furnace compact, with less wall volume to heat and less surface area to radiate heat, and (4) by choosing refractories which will keep the heat inside the furnace instead of allowing it to flow out and be wasted.

The energy supplied does one of two things: It heats the charge or it goes to waste into or through the walls, roofs, doors, and electrodes. If the charge is so placed that it gets the first chance at the heat, taking it directly instead of indirectly after it has first gone to a roof or wall, from which only part of the energy is reflected back to the metal, more will be absorbed by the metal. The energy lost through the walls of a given furnace remains approximately constant for a given operating temperature; if the rate of energy input is increased, not only is the speed of production increased, but the ratio of energy usefully applied to that lost is greater. If a furnace takes 100 kw. and loses 35 kw. through shell radiation and electrode losses, 65 kw. do useful work, and the efficiency is 65 per cent. If the same furnace takes 125 kw. and 37½ are lost, 87½ will do useful work; the output is increased 35 per cent, and the thermal efficiency rises to 70 per cent. Hence the highest rate of power input that the refractories can stand gives the best results as to thermal efficiency and production.

A furnace may not be used continually, but may cool during the night, the walls losing heat, which must be put back during the first heats the next morning before the walls again reach thermal equilibrium. Under these circumstances the thermal capacity, or the quantity of heat the walls can store, is of importance; a furnace for intermittent work, therefore, may show a higher over-all thermal efficiency if it has thin walls of low thermal capacity, even though the rate of heat loss through the walls is greater than with thicker walls. The way the furnace is to be used may influence its design.

In general, to build a given type of furnace in a larger size and to increase its power input at the same time, results in marked in-

crease in thermal efficiency, as the capacity increases faster than the wall volume and wall area, and therefore increases the ratio of usefully applied heat to the amount lost through the walls.

The relative importance of thermal efficiency depends on the cost of electric power. If power is cheap, thermal efficiency can be sacrificed for stability, convenience, or for securing other advantages that might be more readily obtained in a less efficient type; as the cost of power increases it becomes of greater importance in the cost sheet, so that the choice of a type must take account of this item and of the difference in the thermal efficiency of different types of furnaces.

ELECTRICAL CHARACTERISTICS.

For use in a given plant the furnace must be able to utilize the available supply of alternating current, either from the power house of the plant or from a central station, without undue interference with other users of power or with efficient generation of the power.

Few electric furnaces can be designed to take transmission-line voltages, and none can be so designed with safety to the users; hence transformers are always used to step high-tension current down to furnace voltages.

Some furnaces, notably direct-arc furnaces, tend to disturb the regulation of the lines on which they are located. This interference is most noticeable in the case of a furnace of high power requirements on a low-capacity line, and is especially objectionable where a single-phase furnace has to operate on one phase of a three-phase line. The type of furnace must be chosen with care, if lines of low capacity must be used; on a line of large capacity it is often possible to use without trouble a 300-kw. single-phase arc furnace.

A furnace of low power factor involves losses in the transmission of power, and power is priced higher to such a furnace than to one of higher power factor. Few furnaces operate at a power factor below 75, however, and this figure is usually satisfactory to the power plant.

The power department of a plant or the central station that supplies power to a plant proposing to install an electric furnace must always be consulted to ascertain whether the electrical characteristics of the furnace to be installed will fit the power supply of that plant or locality. Usually little difficulty is experienced in large manufacturing districts, but in isolated plants or in small communities it might not be feasible to use any electric furnace whatever because of the lack of suitable power supply.

The electrical characteristics of the furnace may affect its operation or its thermal efficiency. Attempts to get too high a power factor on an arc furnace may result in an unsteady arc. Other things being equal, the higher the voltage and the lower the current

used on a furnace the better its thermal efficiency, because the heat losses from the electrodes increase rapidly as the current increases. The electrode losses from a 100-kw. furnace taking 100 volts, 1,000 amperes, may be 5 kw., while from another 100-kw. furnace taking 25 volts, 4,000 amperes, they may be 15 kw. or higher. Arc furnaces thus usually have an advantage in thermal efficiency over resistance furnaces, which are normally designed to use a much lower voltage than arc furnaces. On the other hand, arc furnaces continually use up electrodes, while the electrodes of resistance furnaces may last for many months.

COST OF OPERATION.

As has been stated, the initial cost of any electric furnace is relatively high, so that charges for interest and depreciation can not be neglected, especially if the rate of production of the furnace is low. Cost of maintenance for refractories and electrodes is also usually higher than on fuel-fired furnaces. The final test of an electric furnace is its ability to produce, under the conditions in any given plant, molten metal of the desired quality in the ladle at a lower over-all cost than any fuel-fired furnace or any other electric furnace. The operation of a given furnace affects the cost as much as the design; therefore it is necessary to consider furnace operation separately from the design. This phase of the subject will be considered later, as regards the commercial furnaces, after a discussion of the design and of the capabilities of all hearth-type furnaces, whether now in commercial operation or not.

HEARTH-TYPE FURNACES—THEIR VARIOUS FORMS.

EXPERIMENTAL ROCKWELL FURNACE.

The W. S. Rockwell Co., furnace engineers, constructed a long time ago a tapping furnace in which metal was to be contained on a shallow refractory hearth, probably supported by refractory piers, below which was a resistor. Work on this type did not progress very far; if the floor of the hearth were made thick enough to hold up the weight of metal, it baffled too much the flow of heat, and the efficiency became very low, only about 6½ per cent, calculated from the test.

All other attempts to design a hearth-type furnace have specified that the heating of the metal should be done from the top or from the sides.

BRISTOL FURNACE.

Bristol^a suggests a tilting furnace for brass with a carbon-rod resistor, a stream of steam being injected to prevent the entrance of air

^a Bristol, W. H., U. S. Patent 1,315,208, Sept. 9, 1919.

and to cut down oxidation of the resistor and of the charge. An outlet is provided for the steam, and there is nothing to prevent the volatilization of zinc through such an outlet. In fact, the stream of steam would tend to sweep out the zinc vapor as fast as it should be evolved.

Instead of reducing the oxidation of the resistor, the steam would attack the resistor, just as red-hot coke is attacked by steam in the production of water gas. Steam passed over red-hot iron filings will produce iron oxide and hydrogen, and it seems very likely that an analogous decomposition of the steam would be brought about by the zinc in brass, forming zinc oxide⁶ and hydrogen, as zinc in the presence of impurities is capable of thus decomposing even liquid water.

. Divine⁷ states that steam blown through a molten zinc alloy oxidizes the zinc.

SNYDER RESISTOR TILTING FURNACE.

The Industrial Electric Furnace Co. show a photograph⁸ and a diagram of a furnace in the form of a horizontal cylinder with conical ends, the melting chamber inside being spherical. The diagram shows a carbon-rod or tube resistor on the horizontal axis of the furnace. No commercial installations have been made for melting brass and according to Booth⁹ no tests were made on brass, though it was suggested that it might be available for melting nonferrous alloys.

CLINGMAN RESISTOR FURNACE.

The Clingman resistor furnace, it was stated in 1918, was being manufactured for trial in a brass and copper mill by the Columbia Electric Furnace Co. It has a carbon-rod resistor, which is said to last 70 to 120 heats, located centrally in the furnace. The furnace walls are extended nearly to the resistor, these projections being made of carbon. Above the rod the furnace has a sort of hopper shape, and below it a hearth from which metal can be poured. The furnace is designed to take 10 to 20 volts and is planned for being carried bodily to the molds for pouring. Different sizes are designed for capacities of 100 to 500 pounds per heat. The 500-pound furnace is to take 60 kw. The cold charge is to be placed over the resistor, supported by the resistor and by the carbon projections in

⁶ Compare Schenck, R., and Dean, R. S., *The physical chemistry of the metals*, 1919, p. 155.

⁷ Divine, R. D., Separation of lead, zinc, and antimony oxides: *Bull. Am. Inst. Min. Eng.*, No. 92, Aug. 1914, p. 1877.

⁸ *Electric Furnace Bulletin* No. 40, October, 1917, p. 27.

⁹ Booth, C. H., Booth electric rotating brass furnace: *Metal Ind.*, vol. 17, 1919, p. 317, par. 1.

the walls, thus being in contact with the resistor until it melts; then it drops through the space between the resistor and the wall into the hearth, where it is heated to pouring temperature by radiation from the resistor.

It is not clear how the current can be prevented from short-circuiting through the solid charge piled on the resistor rod, nor how the rod is to support the weight of the charge without breaking. No actual tests on this design have been reported, although certain claims for its performance have been made, including a claim of a consumption of power at the rate of 140 to 160 kw. h. per ton, though 140 kw. h. per ton is less than the power theoretically required in a furnace of 100 per cent thermal efficiency. It is possible to obtain such figures experimentally if heat stored in a furnace is utilized and no account taken of it. For example, one could charge a few pounds of brass into a steel furnace after pouring a heat of steel and, without using any electric power at all, melt the brass and raise it to pouring temperature. The furnace would then melt brass at an apparent power consumption of no kw. h. per ton.

STEINBERG AND GRAMOLIN FURNACE.

Two Russian inventors¹⁰ have designed another carbon resistance rod furnace, having three rods over a hearth, for three-phase current. No application of the design seems to have been made.

JOHNSON FURNACE.

W. McA. Johnson¹¹ suggests a furnace in which a resistor built up of a latticework of carbon bars is located over the hearth. Granular resistor may also be piled over the lattice. Unusual features of the furnace are a propeller-wheel stirrer set in a recess at one side of the hearth to cause circulation of the metal and an impeller pump, which is designed to circulate the gaseous contents of the vapor space to cause a gas flow from the resistor downward till it strikes the metal in the hearth; this mechanism would convey heat by the bodily motion of the hot gas as well as by radiation. It is not clear how the stirrer and the pump, which are to be made of graphite or fire clay, could be kept in steady operation without difficulty.

The furnace is designed mainly as a mixing and superheating furnace rather than as a melting furnace, and the charging opening is small. Molten copper and molten zinc in the proper proportions are to be melted in other suitable furnaces and poured into this furnace

¹⁰ Steinberg, S., and Gramolin, I., British Patent 111,879 of Dec. 13, 1917.

¹¹ Johnson, W. McA., U. S. Patent 1,243,416, Oct. 16, 1917; application filed Mar. 8, 1914.

for mixing.¹² Borings or other fine material can also be charged. This furnace has not been actually constructed or operated on brass.

FITZGERALD-THOMSON FURNACE.

The power input that can be taken by a single carbon rod used as resistor is limited, and such a rod is fragile. If its diameter should be increased to strengthen it, its resistance would be so reduced that huge currents at low voltage would be needed. If a complex resistor were built in the form of latticework, for example, there would be difficulty in building it up so that it would give uniform heating.

All such resistors to be used in a brass furnace run at temperatures so high that the refractories which support the ends of the resistors receive severe treatment. Fitzgerald¹³ built a furnace, after a design by Thomson, in which the resistor consisted of a layer of small carbon rods laid side by side, the current passing at right angles to the axes of the rods. The contact resistance between the rods was thus utilized as well as the resistance of the rods themselves. The ends of the rods were supported on magnesite ledges at the sides of the furnace, the electrodes entering from the ends, the resistor being placed over the hearth. The furnace was designed for use in carrying out the Imbert zinc-smelting process, but was also tried for melting iron and copper.

The resistor burns away when the air strikes it; a modification of the furnace provides therefore for placing a septum between the resistor and the hearth to prevent an access of air to the resistor. The septum is made of carborundum or siliconized graphite to resist oxidation and to have heat conductivity high enough to baffle the flow of heat as little as possible.

The weak point of the furnace was the support of the ends of the resistor rods, which were so hot that the supporting ledges of magnesite failed. To avoid this weakness, a resistor was designed of molded carbon plates 5 inches high, $8\frac{1}{2}$ inches long on top, $5\frac{1}{2}$ inches long on the bottom, and about an inch thick in the thickest part. These plates, instead of being flat and of uniform thickness, were ribbed or fluted; by placing alternate ones higher than the others, so that a projection fitted into a depression, the row of plates was self-supporting if the ends of the row and not the sides were held up. As the ends are held up by the large carbon or graphite electrodes, the sides of the resistor need no support and the resistor does not come in contact with any refractories.

¹² Compare also Addicks, L., U. S. Patent 1,041,940, Oct. 22, 1912; Clamer, G. H., U. S. Patent 1,198,618, Sept. 19, 1916.

¹³ Fitzgerald, F. A. J., A new electric resistance furnace: Trans. Am. Electrochem. Soc., vol. 19, 1911, p. 273.

The thickness of the plates is slightly less at the bottom than at the top, so that when the resistor is assembled the long row of plates forms an arch.

This makes the lower part of the arch offer a shorter path to the current, and, added to the fact that the plates are narrower at the bottom of the resistor than at the top, means that the current density is higher on the bottom of the resistor than at the top and the resistor will heat up more at the bottom. The corrugated plates offer much contact resistance, so that of the total resistance of a row of 71 plates 98 per cent was due to the contact resistance and only 2 per cent to that of the body of the plates.

THOMSON FURNACE.

The principle of a self-supporting carbon resistor described above has been patented by Thomson in a large variety of forms.¹⁴ In the later forms the use of molded carbon plates has been given way to the use of a continuous graphite resistor which could be made from a long, square, graphite bar by shaving down the bar into a wedge shape to make the lower part of the resistor narrower than the top, as was the case in the ribbed-plate resistor, then by sawing the resistor nearly but not quite through lengthwise, giving it the shape of a U, and then making a series of smaller U's alternately upright and inverted by cutting each leg of the big U nearly but not quite through by alternate cuts from top and bottom. This gives a resistor in which the current enters one leg of the big U, takes a sinuous path around the cuts in that leg into the other leg, and again around the cuts out the second leg. The slots in this zig-zag grid are filled with some refractory material of low electrical conductivity when hot, such as alumina, and the surface of the resistor is likewise coated with it. This increases the strength and stability of the resistor and also reduces its tendency to oxidation. The cylindrical, molded graphite-fire-clay resistor used in the Snyder crucible lift-out furnace previously described was slotted in an analogous fashion to increase the resistance, and the slots were filled with alundum cement to increase the stability. The pure-graphite resistor used in the Thomson furnace is of course capable of being run at a much higher temperature than the graphite-fire-clay resistor used by Snyder.

¹⁴ Thomson, J., U. S. Patents 950,877, 950,880, 950,882, 951,086, Mar. 1, 1910; 994,217, June 8, 1911; 1,080,862, 1,080,863, 1,080,864, 1,080,865, 1,080,866, Dec. 9, 1913; 1,298,421, Mar. 25, 1919; 1,808,877, July 8, 1919; 1,818,028, 1,818,029, 1,818,030, 1,818,081, Oct. 7, 1919; 1,821,682 and 1,821,683, Nov. 11, 1919.

The John Thomson Press Co. advertised^{14a} an electric brass furnace in January, 1919, and in February replied to an inquiry from a brass foundry that the resistor (of the style just described) is free from air burning, and therefore practically permanent. The furnace temperatures are controlled by a transformer having taps to supply 20 to 50 volts in $2\frac{1}{2}$ -volt steps and 50 to 100 volts in 5-volt steps at the furnace terminals. The kv.-a. capacity of the transformer is not given. Blue prints dated July 10, 1918, of a furnace to hold 1,000 pounds of brass were sent to the foundry, but it was stated that various modifications were being made in that design as a result of trial. In a test by the makers the furnace is said to have melted brass at the rate of 250 kw. h. per ton. The figure doubtless refers only to 24-hour operation.

The 1,000-pound furnace, complete with transformer, was priced at \$11,800 in February, 1919. The upkeep cost was said to be "very small."

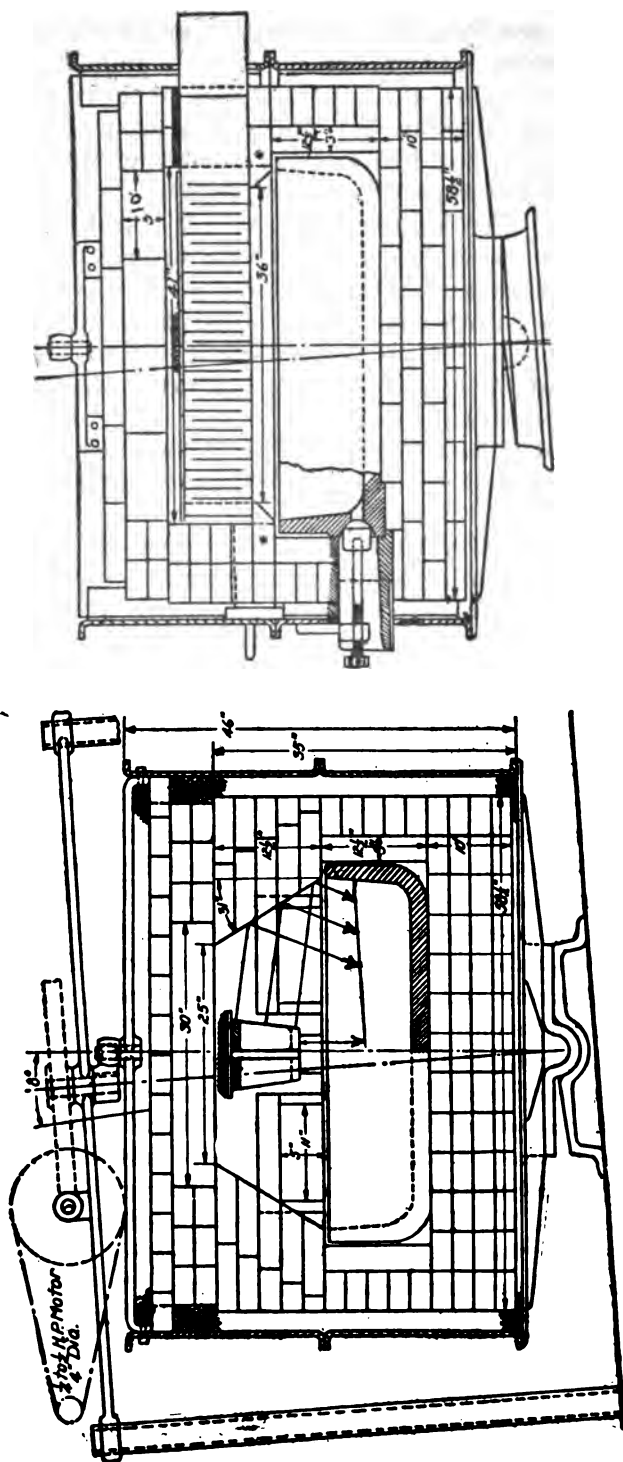
The blue prints, from which figure 11 is drawn,¹⁵ show a tapping furnace with a rectangular hearth, over which is a zigzag resistor, each end of which rests on a carborundum support. There is no septum between resistor and hearth, as the resistor is claimed to be free from oxidation. Both terminals of the single-phase circuit are connected to the ends of the resistor at one end of the furnace, the tap hole and spout being at the other end of the furnace. Charging doors are provided at the ends of the furnace. The roof is arched over the resistor in order to reflect heat upon the bath.

The whole furnace is mounted in a framework of channel iron on a base. In the center of the base is a hemispherical projection fitting into a cup in a smaller base fastened to the floor. Outside the cup the small base has a flatly conical surface. At the center of the top of the framework is a bearing with an eccentric arm. The eccentric rides a rigid bearing and is rotated by a motor through a geared and belted transmission. The motion of the eccentric arm produces a gyrating movement of the furnace, evidently designed to stir the metal and to minimize surface overheating of the bath, on the principle of stirring a bath of brass by motion of the furnace previously shown by the Bureau of Mines to be desirable in the rocking arc-furnace. A furnace motion, designed to stir the metal, of about the degree obtained in the Thomson design, was used on some of the early forms of the Stassano¹⁶ steel furnace. Flexible leads must, of course, be provided to allow the gyrating motion.

^{14a} Chem. and Met. Eng., vol. 20, 1919, advertising pages, January issue.

¹⁵ See also Thomson, J., U. S. Patent 1,308,877, July 8, 1919.

¹⁶ Rodenhauser, W., and Schoenawa, J., translated by vom Bauer, C. H., Electric furnaces in the iron and steel industry. 1917, p. 108.



Transverse center section.

Longitudinal center section.

FIGURE 11.—Diagram of Thomson slag-resistor gyrating furnace.

LATER FORM OF THE THOMSON FURNACE.

The Thomson gyrating brass furnace appears to have been abandoned in favor of a cylindrical, rotating furnace with a central, tubular zigzag resistor. The latest design is stated by Thomson¹⁷ to be based on his Patents Nos. 950,878, 950,880, 1,318,030, and 1,318,031.

The original method of protecting the resistor from oxidation by a coating of carborundum is said to have been a "disappointment," as it did not prove satisfactory under severe test. Another method is said to have been devised.

The Thomson furnace could be provided with two or three resistors in order to use polyphase power. The resistor could not be withdrawn while charging, as can the electrodes of the rocking and rotating indirect-arc types, which will be described farther on in this report. It would be necessary to exercise great care in charging in order to avoid breaking the resistor. Mechanical charging from overhead would therefore be impracticable.

If this type of furnace is made to rotate completely, the problem of maintaining a good electrical contact at the sliding contact would be even more serious than in the completely rotating arc type, as the resistor takes a lower voltage and a higher amperage than the arc.

Until foundry tests have been made to show the behavior of the Thomson furnace its value can not be predicted, for that will depend on the life of the resistor in actual practice, on the actual number of kilowatt hours needed per ton of metal, and on the rate of production, life of the lining, and on its general convenience.

HARVEY FURNACE.

Harvey,^{17a} an English designer, has described a design in which the furnace is cylindrical, with a rounded bottom and an open top, which is closed by a door carrying a solid resistor or electrodes, between which an indirect arc is struck. The furnace is pivoted at the bottom and is inclined on its side, resting on a support, as shown in figure 12. The furnace is then rocked or rotated to wash as much as possible of the walls with the metal.

The use of a solid resistor supported in this way seems hardly practicable, and the electrodes for an indirect-arc form would be in an inconvenient and cramped position.

HOSKINS OVERHEAD-RESISTOR TILTING FURNACE.

A Hoskins furnace with resistor over the bath was tested on brass, two tests being made in 1913 and 1914 at the plant of the Hoskins Manufacturing Co., Detroit, at both of which a representative of the

¹⁷ Thomson, J., Personal communication, Apr. 30, 1920.

^{17a} Harvey, L. C., U. S. Patent 1,837,839, Apr. 20, 1920.

Bureau of Mines was in attendance. Further tests were made in 1915 by the bureau in the Cornell laboratories at the bureau's field office in Ithaca.

1913 TEST.

The furnace had been designed for melting nickel-chromium alloys and consisted of a rectangular iron box built up from sheet iron and

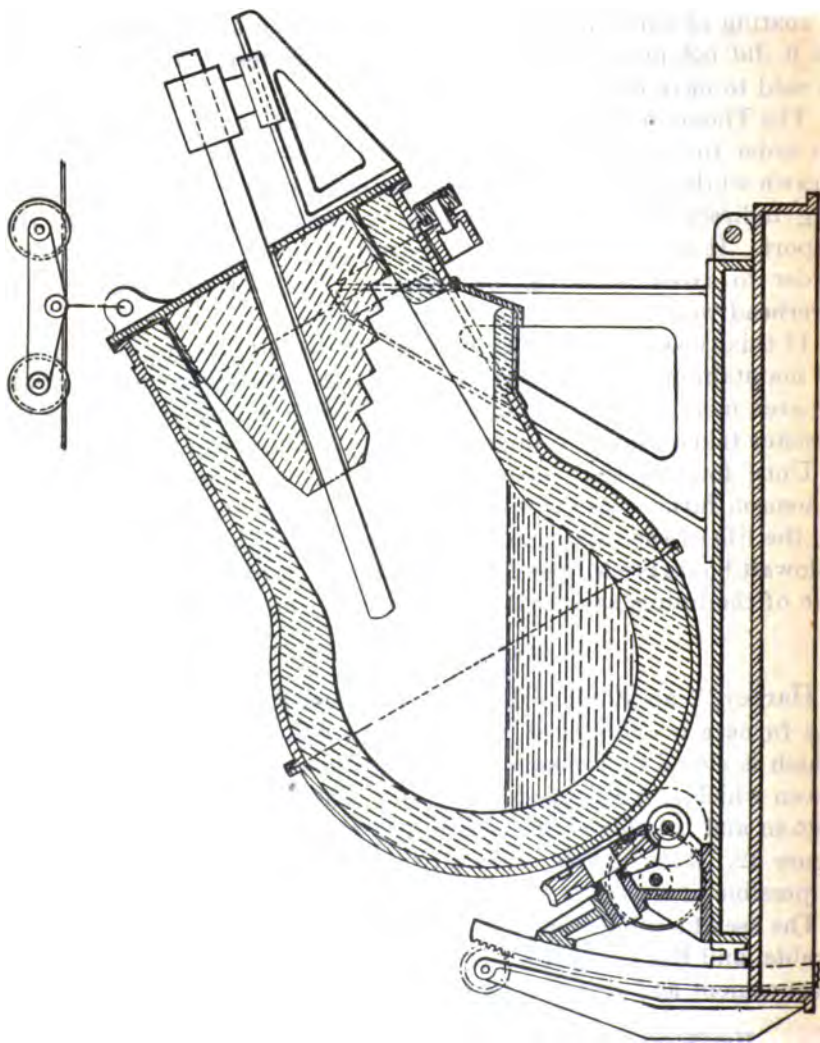


FIGURE 12.—Harvey rotating furnace.

angle iron 37 by 38 by 23 inches high and had a cover 30 by 31 by 7 inches thick. Figure 13 is a diagrammatic sketch showing the principle. The shell was lined with infusorial earth for about $1\frac{1}{2}$ inches, then with $1\frac{1}{2}$ inches of crushed magnesite, and then with 4 inches of magnesite brick. A melting hearth 13 inches square and about 6

inches deep was covered smoothly with magnesite. This hearth was provided with a pouring spout also lined with magnesite, which led into a lip provided on the furnace shell. This spout had also to serve as the charging door, although the opening was only about 5 inches in diameter.

Over the melting hearth the sides were so built up of magnesite bricks as to give an opening about 15 inches square. Two opposite sides had the bricks so placed as to form ledges for the support of the resistor. This resistor consisted of some 60 flat carbon plates

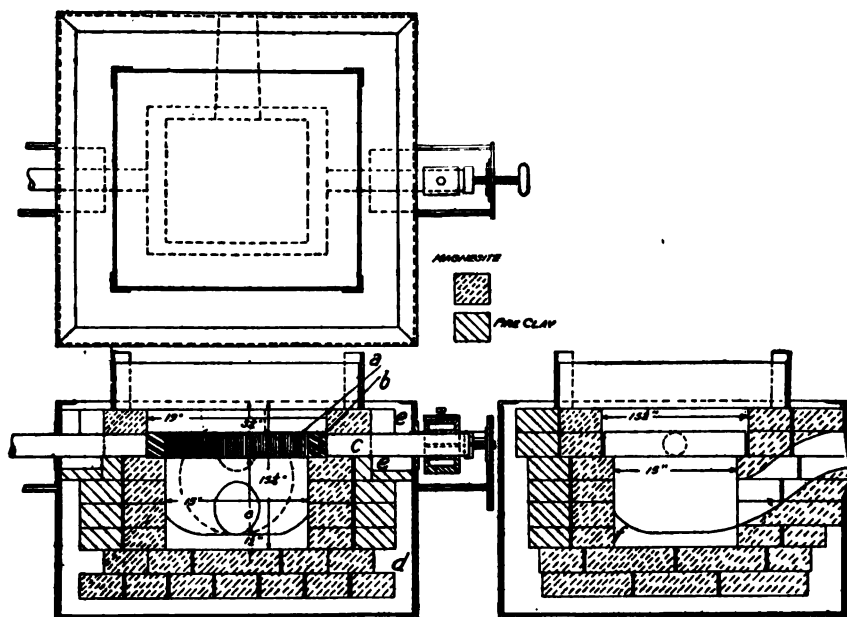


FIGURE 18.—Diagram of Hoskins overhead-resistor tilting furnace: *a*, Carbon resistor plate; *b*, graphite front plate; *c*, graphite electrode; *d*, fire-sand pocket; *e*, infusorial earth; *f*, magnesite.

$2\frac{1}{4}$ by $\frac{1}{4}$ by 15 inches, placed side by side, thus giving a carbon resistor 15 by 15 by $2\frac{1}{4}$ inches high over the hearth.

At the sides graphite "push blocks" were provided, by means of which, as they were crowded in or loosened by the screw-regulating device, the contact resistance of the plates in the resistor could be altered at will.

The lower surface of the carbon resistor was 6 inches from the bottom of the melting hearth. The roof or cover, which could be lifted off bodily by a crane, was 1 inch above the upper surface of the carbon resistor and consisted, next to the resistor, of a layer of carbon slabs $1\frac{1}{2}$ inches thick. Above the carbon slabs was $8\frac{1}{2}$ inches of fine magnesite, and above this infusorial earth $1\frac{1}{2}$ inches thick at the edges of the cover and crowned up to $2\frac{1}{4}$ inches or 3 inches at the center.

Two different alloys were melted in the furnace, the first having approximately the following composition: Cu, 88.4 per cent; Sn, 7.3 per cent; Pb, 2.0 per cent; Zn, 2.3 per cent. This is one grade of leaded gun metal; it should melt at about 985° C. and will be called "alloy A." The other alloy, which will be called "alloy B," had approximately the following composition: Cu, 76 per cent; Sn, 1 per cent; Pb, 1 per cent; Zn, 22 per cent. This alloy is about midway between ordinary red and yellow brass and would be commercially termed a "half-yellow half-red" metal, which should melt in the neighborhood of 920° C.

The current was brought in at 220 to 240 volts and was run through transformers giving the ratio of 27½ volts on the secondary to 220 volts on the primary. A recording watt-hour meter was provided, which was connected back of the transformer, so that the consumption of power recorded below includes all transformer and lead losses as well as the power actually used in the furnace itself. The results of the test are given in Table 24.

TABLE 24.—1913 test on Hoskins overhead-resistor furnace.

Heat No.	Power on.	Power off.	Alloy used.	Weight of charge.	Material charged.	Pouring temperatures.	Kw. h. used.	Average kw. input. ^b
				<i>Pounds.</i>		<i>° C.</i>		
1 ^a	8.15 a. m.	11.00 a. m.	A	100½	Gates.....	1,210	115	42
2.....	11.16 a. m.	11.55 a. m.	B	95½	do.....	1,065	33	47
3.....	12.30 p. m.	1.40 p. m.	A	96	Ingot.....	1,220	47	40
4.....	1.53 p. m.	2.23 p. m.	B	93½	do.....	1,120	23	46
5.....	2.33 p. m.	3.14 p. m.	A	95	do.....	1,210	32	49
6.....	3.27 p. m.	3.54 p. m.	B	93	do.....	1,125	21	48
7.....	4.05 p. m.	4.36 p. m.	A	94½	do.....	1,220	24	48
Total.....				668			c 295	

^a Furnace first heated empty. Charging begun at 8.46 a. m.

^b The furnace was cold at start and when first started up took 28½ volts, 1,400 amperes, about 40 kw.

^c Average 885 kw. h. per ton.

Of the charge 658½ pounds of metal were recovered, the loss being 1.4 per cent. It appeared from this test that a higher power input would be desirable, that the hearth should be of greater capacity, and the charging door should be enlarged. These changes were made and another test was run at the plant of the Hoskins Manufacturing Co. in 1914, at which one of the writers was again present.

1914 TEST.

The furnace was rebuilt by deepening the hearth, so that instead of its bottom being but 6 inches from the bottom of the resistor blades, it was 10½ inches from them in the center and 8 inches at the sides. The molten charge was about 4 inches deep at the center. The pouring spout and charging hole was increased in size to an oval hole 7½ by 5 inches, and the opening into the furnace was tapered

back so that all parts of the hearth could be seen and could be reached easily with a poker. By slightly tilting the furnace back it could be readily charged. Instead of a maximum of 55 to 60 kw., which was all the power available in the first test, up to 85 or 90 kw. could be drawn.

The blades were supported on two carborundum (Refrax) slabs 2 inches thick by 4 inches wide by 16 inches long, so laid that about 1 inch of the slab extended out beyond the inner wall of the furnace. On top of the slabs was put $\frac{1}{4}$ inch of magnesite cement to reduce leakage of current through the slabs. The damp cement was covered with thin strips of cardboard to prevent the blades from sinking into the magnesite before it was dry. It was found to be more convenient to build the furnace so as to allow a 2-inch space between the top of the resistor blades and the bottom of the carbon slabs in the cover instead of the 1-inch space used in the previous test. Tests on red and yellow brass are shown in Tables 25 and 26.

Before the test of Table 25 the furnace was heated up slowly for a short time till the melting chamber showed a dull red, in order to dry out the new hearth. The furnace was not brought up to full running temperature and was only at 95° C. the next morning.

TABLE 25.—1914 test of Hoskins overhead-resistor furnace on red brass.

[An alloy of 83 per cent Cu, 7 per cent Zn, 4 per cent Sn, and 6 per cent Pb was melted, ingot metal being^a charged.]

Heat No.	Power on.	Power off.	Weight of charge.	Pouring temperature.	Kw. h. used.	Average kw. h. input.
			Pounds.	° C.		
1.....	7.13 a. m.	9.33 a. m.	231 $\frac{1}{2}$	1,200	163	70
2.....	9.50 a. m.	10.51 a. m.	282	1,180	81	82
3.....	11.15 a. m.	12.00 m.	300	1,200	63	63
4.....	12.47 p. m.	1.00 p. m.				
5.....	1.15 p. m.	2.16 p. m.	300	1,160	65	65
6.....	2.28 p. m.	3.24 p. m.	300	1,200	49	53
	3.45 p. m.	4.50 p. m.	300	1,150	52	45
Total.....			1,713 $\frac{1}{2}$		473	

^a Average 555 kw. h. per ton.

Metal losses were not taken, since a crack developed at the upper part of the hearth, through which considerable metal was lost into the furnace lining.

The test on yellow brass, given in Table 26, was made the next day after the test of Table 25 and started with the furnace chamber at 540° C., no power having been used at night. Ingot metal of 66 per cent copper, 33 per cent zinc, and 1 per cent lead was melted.

TABLE 26.—1914 test of Hoskins overhead-resistor furnace on yellow brass.

Heat No.	Power on.	Power off.	Weight of charge.	Pouring temperature.	Kw. h. used.	Average kw. input.
			<i>Pounds.</i>	<i>°C.</i>		
1.....	7.08 a. m..	8.40 a. m..	255.50	1,060	100	66
2.....	8.52 a. m..	9.52 a. m..	250.00	1,070	64	64
3.....	10.03 a. m..	10.50 a. m..	260.25	1,060	44	57
4.....	11.03 a. m..	11.51 a. m..	252.75	1,070	49	61
5 ^a	12.30 p. m..	1.08 p. m..	250.75	1,060	35	55
6.....	1.19 p. m..	2.09 p. m..	250.00	1,050	50	60
7 ^b	2.20 p. m..	3.08 p. m..	250.00	1,060	44	73
8 ^c	3.18 p. m..	4.02 p. m..	255.00	1,050	35	48
Total.....			2,014.25		^d 421	

^a Part of metal charged at 12.10 p. m.^b Power input increased on this heat.^c Power input lowered on this heat.^d Average, 420 kw. h. per ton.

The metal poured weighed just a ton, the loss being 0.7 per cent and varying on individual heats from 0.6 to 0.8 per cent. The charges were reduced to 250 pounds on this run in order to keep the metal level below the crack in the hearth and to make it possible to obtain figures on the loss in metal.

On both these tests the time occupied in pouring was excessive, as suitable ladles were not at hand. Without this delay 2,000 pounds of red brass or 2,500 pounds of yellow brass could have been melted in a 9-hour day. This would have made it possible to melt red brass on 9-hour operation with about 500 kw. h. per ton and yellow brass with about 400 kw. h. per ton. Had the hearth been tight, so that 300-pound charges of yellow brass could have been made, the production should have been about 2,700 pounds at something under 400 kw. h. per ton. On 24-hour operation the 300-pound furnace should operate at about 335 kw. h. per ton of red brass heated to 1,200° C, or 275 kw. h. per ton of yellow brass.

The resistor (which was made up of the same plates as were used in the 1913 test), the carborundum slabs, and all parts of the furnace were in good shape save for the crack in the hearth.

The increase in furnace capacity and in power input had made the furnace much more efficient than it was in the 1913 test. Through the courtesy of the Hoskins Manufacturing Co. the furnace shell was loaned to the Bureau of Mines for further experiments.

While the carborundum ledges on which the resistor plates rested were in good condition after these runs, the ledges gave trouble from spalling when the furnace was used later by the Hoskins Manufacturing Co. for melting nickel-chromium alloys. The furnace was not used enough on nickel-chromium alloys to show how serious this difficulty was; carbon flaked off from the resistor and dropped into the bath, doing no damage on brass, but great damage on nickel-chromium alloys.

BUREAU OF MINES TESTS OF HOSKINS OVERHEAD-RESISTOR FURNACE.

Two carbon rafters were placed over the hearth, pure-white alundum slabs, the most refractory grade, $\frac{1}{2}$ inch thick, were put over them for electrical insulation, and carborundum slabs 1 inch thick were placed on top of the alundum. If the resistor plates, which ran at above $1,800^{\circ}$ C., were put directly on the alundum, the alundum was attacked and some aluminum carbide and nitride was formed.

In order to increase the voltage required, 12-inch resistor plates were used instead of the former 15-inch plates. A run was made on red brass of $81\frac{1}{2}$ per cent copper, $8\frac{1}{2}$ per cent zinc, 4 per cent tin, 6 per cent lead, made up of about 40 per cent scrap copper, lead, and tin, 40 per cent yellow chips, and 20 per cent gates. The furnace was cold at the start. The metal was poured at $1,120^{\circ}$ C. The results of the test are given in Table 27.

TABLE 27.—*Test of Hoskins overhead-resistor furnace made by the Bureau of Mines.*

Heat No.	Power on.	Power off.	Weight of charge.	Kw. h. used.	Average kw. h. input. ^c
			<i>Pounds.</i>		
1.....	8.00 a. m..	10.52 a. m..	300	172	60
2.....	11.00 a. m..	12.19 p. m..	300	68	52
3 ^b	1.00 p. m..	2.11 p. m..	300	61	58
4.....	2.18 p. m..	3.38 p. m..	300	66	51
5.....	3.43 p. m..	4.58 p. m..	300	59	47
Total.....			1,500	^c 426	

FURNACE AT 575° C. NEXT MORNING; NO POWER ON OVERNIGHT.

1.....	8.00 a. m..	9.57 a. m..	300	113	57
2.....	10.15 a. m..	11.35 a. m..	300	87	66
3.....	11.48 a. m..	12.26 p. m..	206	38	60
Total.....			806	^d 238	

^a With the furnace hot from the previous day's run, the first two heats used but 200 kw. h. and took 3 hours and 35 minutes against 240 kw. h. and 4 hours and 19 minutes when starting with a cold furnace. Therefore, on regular 16-hour operation the power consumption would drop 40 kw. h. on the first two heats, and 40 kw. h. could be used on a 200-pound heat at the end, so that the output would be 1,700 pounds with 426 kw. h., or 600 kw. h. per ton.

^b Charge in, no power on at 12.24 p. m.

^c Average, 620 kw. h. per ton.

^d Average, 580 kw. h. per ton.

These energy figures check those calculated from the previous run on red brass. A higher power input was used on the previous run, which would give a higher efficiency, other things being equal, but the metal was not heated as hot as in the previous run. Other tests gave similar results to the one above cited. Notwithstanding that the borings were oily and the gates and scrap somewhat dirty, the gross metal loss on these runs was 1.9 per cent.

The figures of consumption of power and of the metal loss are low enough on this type of furnace to make it a possible competitor of fuel-fired furnaces, as crucibles are eliminated. But the furnace has

several drawbacks. The resistor will require replacement from time to time and to replace it without allowing the furnace to cool would be difficult. In the small 300-pound size only a small charging door can be provided, so that much labor is required to charge the furnace. As the capacity is increased the mechanical difficulties in supporting the resistor increase, so that a self-supporting resistor of the Thomson type—if that proves practical—would be preferable for large furnaces, even at the sacrifice of the many advantages of a resistor whose resistance can be adjusted.

The overhead-resistor furnace with a resistor nearly as large as the surface of the hearth, worked satisfactorily on 800-pound charges of yellow brass, showed no signs of surface overheating, and required no agitation of the metal to prevent zinc losses. The question of uniformity of product was not studied. The agitation in the Thomson furnace may be needed because of the small radiating area of the resistor, which gives a more localized source of heat. It is a question how much effective stirring a slight gyrating motion will give. If it mixes the metal, it might be a desirable feature on a large furnace with a rather deep bath in which gravity may cause the alloy to be richer in heavier constituents at the bottom of the crucible.

HOSKINS TILTING FURNACE WITH RESISTORS AT THE SIDES.

After the Hoskins Manufacturing Co. found that the overhead resistor would slowly disintegrate and allow carbon powder to drop into the bath, thus ruining their nickel-chromium alloys, another tilting furnace was built in which the hearth was placed between and below two rows of resistor blades set upright on two sides of the furnace, a carborundum slab about 1 inch thick and 4 inches high being placed outside and 1 inch away from each row of resistor plates, resting on the carborundum bricks on which the resistors also rested. The purpose was to form a pocket to catch any carbon powder that fell off the resistors and prevent it from dropping into the bath. This furnace was never tried for brass, and little work was done in it for nickel-chromium alloys, as an induction furnace was installed and attention was diverted to that type. Some difficulties were encountered in maintaining the refractories back of the resistor and in the roof.

FURNACES WITH CARBORUNDUM RESISTORS.

Silicon carbide (carborundum or crystolon) is the only refractory material, save carbon or graphite, available in commercial quantities that does not fuse or soften at 2,200° C., is not affected by a reducing atmosphere at that temperature, and is oxidized very slowly. It can be made into bricks or shapes, and its resistivity at around 2,000° C.

is of the same general order as that of a granular carbon resistor. It has been often suggested as a resistor material for electric furnaces.

Unlike carbon, when cold the resistance of silicon carbide is very high and the slope of its temperature-resistivity curve is great. It is therefore necessary either to impress a very high voltage on the resistor when starting from the cold or to preheat the carborundum to a red heat, when it will begin to carry current at a reasonable voltage.

Both the General Electric Co. and the Westinghouse Co. have experimented with carborundum resistors, and Thornton¹⁸ has described such a resistor for use in electric furnaces for heating steel, for forging, heat treatment, and annealing. He says that it might be possible to apply such a furnace to the melting of brass and copper.

When a carborundum trough is used, as in the Baily furnaces, to contain a resistor of granular carbon, it carries current and becomes, in effect, a part of the resistor. In designing an electric furnace in which carborundum is to be used as a refractory lining it is necessary that suitable insulation, such as alundum, should be used where hot carborundum would allow a flow of current at points where such flow is not desired.

EXPERIMENTAL CARBORUNDUM RESISTOR FURNACE.

The work of the Bureau of Mines on the triple lift-out Hoskins-type furnace showed the advantages of generating heat within the mass to be heated instead of outside it. As a result it seemed probable that if a furnace could be made in which the metal was contained in two hearths, with a resistor between them and in contact with the metal in both hearths, it would be thermally efficient. Obviously such a resistor made as a septum between two hearths should be metal tight, and hence should be built permanently into the masonry of the furnace; it could not be made of any short-lived material. It should have a high specific resistance, as its cross section must be large compared to its length. Fine carborundum powder, mixed with 10 per cent of alundum cement to increase the resistance of the mixture, bonded with a little dextrin, was tried as a resistor material.

The furnace was made of an old cast-iron sink lined with fire brick, to give a chamber about 9 by 22 by 9 inches. This was divided into two hearths 9 by 9 by 9 inches by a central septum 4 inches thick, which was built as follows: Two split carborundum brick (Refrax) 9 by 4½ by 4½ inches were set up 1½ inches apart and cemented in place by alundum cement. The space between them was rammed full of the dampened carborundum, alundum, dextrin

¹⁸ Thornton, F., jr., A resistance furnace: Trans. Am. Electrochem. Soc., vol. 32, 1917, p. 141; see also Thornton, F., jr., and Colby, O. A., U. S. Patents 1,306,289, 1,306,250, and 1,306,251, June 10, 1918.

mixture, which was then baked. Two other split carborundum brick (Refrax) were placed on top of the others, cemented into place, and the space between them was rammed full of alundum cement, which was then baked.

The furnace then contained two hearths, each a 9-inch cube, separated by a septum 4 inches thick, the upper half of which had a nonconducting alundum layer inside it and so could not act as a resistor, but the lower half was solid carborundum on the sides and carborundum plus a little alundum tamped in the center. Graphite electrodes were set vertically into each hearth, so that with the septum hot enough to carry current and the hearths half full of molten metal the current would enter one electrode, pass through the metal in one hearth, through the resistor, out into the metal in the other hearth, and thence out the other electrode.

A Y spout connected each hearth to a central pouring spout and the furnace was arranged to tilt for pouring. The resistor was heated by placing granular carbon in each hearth and bridging over the septum with a graphite U, one leg of which reached down into the granular carbon on each side. Current was turned on and the furnace warmed up by the granular carbon. After the septum (the resistor) was hot enough to carry current the graphite U was pulled out, the granular resistor dumped out, and brass borings were charged into each hearth to make connection between electrodes and resistor. The furnace took 2,000 amperes at about 6 volts; that is, about 12 kw. The power input could not be raised, as 2,000 amperes was the limit of the power supply. Most of the furnace lining was still rather cool; hence the power input was only adequate to partly fuse the brass borings and to stick them together near the resistor.

The furnace was cooled, the brass taken out, and a test made with aluminum, as it was obvious that 2,000 amperes would not give power enough to melt brass.

The furnace was preheated with the graphite U in place and with granular carbon in the hearths, $12\frac{1}{2}$ kw. h. being thus used. The U was then removed and the current was passed through the granular resistor and septum to heat the furnace more; $10\frac{1}{2}$ kw. h. were thus used, or a total of $23\frac{1}{2}$ kw. h. preheat. The granular resistor carbon was then dumped out and borings and ingot of 92 per cent aluminum and 8 per cent copper were put in as rapidly as the furnace would melt them. In $1\frac{1}{2}$ hours $51\frac{1}{2}$ pounds of the aluminum alloy had been melted, raised to 720° C., and poured; $27\frac{1}{2}$ kw. h. were used in melting, or a grand total of $50\frac{3}{4}$ kw. h. from the cold start.

The voltage fell to about 8, with 2,000 amperes flowing, in about 10 minutes after the cold metal had first been charged and ran very steadily at those figures. The voltage was taken across the electrodes,

but the kw. h. of figures include also all lead losses. Probably half of the voltage drop was due to the resistance of the molten aluminum.

Considering the small size and the low power input of the furnace and the absence of a roof, the hearths being left open to the air to permit of observation, the power consumption alone was not unpromising; the high current required, however, made the furnace useless, and it was torn down. The carborundum in the center of the resistor had formed into larger crystals and had begun to decompose into silicon and graphite, showing that in the center of a 4-inch resistor, both sides of which were in contact with metal at not over 720° C., a temperature of over 2,200° C. had been reached.

For any such design to be of value some resistor material or mixture of materials of high resistance, of high melting point, and of high thermal conductivity is required.

Tungsten powder might make a good material for such a resistor, as it melts at about 3,000° C. and has a high thermal conductivity; but its electrical resistance is too low and it would have to be mixed with some refractory nonconductor. Chemically pure zirconia might serve for this purpose, as most chemically pure metallic oxides are nonconductors at high temperatures, and pure zirconia is said to melt only at about 2,600° C. Zirconia, however, has a low thermal conductivity. The cost of these materials makes any attempt at working out this plan for a brass furnace out of the question at present.

It was found later that Massip¹⁹ had suggested a similar type of furnace, but no data are available concerning it. Two brass furnaces using carborundum resistors have been described in the patent literature.

WEIDENTHAL FURNACE.

Weidenthal²⁰ describes a cylindrical tilting furnace for melting brass, bronze, and aluminum, the axis of the cylinder being vertical. The hearth is in the bottom of the furnace, and three groups of vertical-rod resistors for three-phase current are set upright around the inside of the furnace chamber, above the hearth. An arched roof reflects the heat downward upon the bath. The resistor rods are made of carborundum fire sand suitably bonded, with an outer coat of pure carborundum. When cold, carborundum is practically a nonconductor of electricity, but when raised to a white heat it conducts the current nearly as well as carbon; it is necessary as a result to provide some means for heating the resistors when starting from the cold. At least one resistor in each phase is made hollow and is filled with

¹⁹ Massip, G., U. S. Patent 1,005,725, Oct. 10, 1911.

²⁰ Weidenthal, H. G., U. S. Patent 1,804,425, May 20, 1919.

a granular carbon resistor that carries the starting current. As the adjacent resistors become heated by the radiation from the cored starting resistor they also begin to carry current and finally all become hot and carry current.

The resistors stand upon a ring-shaped carbon or graphite electrode embedded in the bottom of the furnace. The upper ends of the resistors stick out of the roof of the furnace and are there clamped to the conductors. On account of the liability of such a resistor to break, the system of connections has been designed to allow the ready replacement of any resistor rod that has failed.

It is not clear how particles of resistor, of refractory lining, or of other dirt are to be kept from falling down on the lower electrode when a ruined resistor rod is withdrawn for replacement, nor is it plain how such particles are to be cleared away before inserting a new resistor. The presence of such particles would prevent proper contact between the electrode and resistor. If any tests have been made on this furnace no results have been made public.

The question of the uniform molding of a molded resistor, such as carbon graphite or carborundum, which have negative temperature coefficients of resistivity, is involved in such a resistor. Unless they are uniform the old "hot-spot" trouble will be encountered.

The furnace walls, against which the resistors are placed, would have to be extremely refractory to stand up under the resistor temperature required to melt brass in such a furnace with reasonable efficiency.

Krieger²¹ suggests the use of carborundum plus a small amount of powdered metallic silicon (6 per cent), with or without a smaller amount ($\frac{1}{2}$ to 2 per cent) of powdered metals of the iron or silicon groups. The mixture is bonded with a temporary binder, such as tar or glucose, pressed into shape, and baked. The silicon is said to form an oxidation-resisting coating on the outside and to give sufficient electrical conductivity to the mass to allow ready starting from the cold. Such a resistor is said to be capable of operating unchanged in the air at a temperature of 1,500° C.

ERICHSEN FURNACE.

Erichsen²² suggests a tilting brass furnace in which the metal is held in a rather flat, shallow hearth, heating being done by one or two rows of carborundum rod resistors over the metal. Above the resistors is a preheating chamber, the floor of which forms a false roof over the resistors. This floor is sloped so that the preheated metal may be poked down (or run down if it melts) into the hearth through an opening in the center of the floor.

²¹ Krieger, M. L. (Soc. l'éclairage électrique), French Patent 497,805 of Apr. 4, 1919.

²² Erichsen, A. M., U. S. Patent 1,367,364, Feb. 1, 1921.

The resistors are placed on each side of the space below this opening, but not directly beneath it, to avoid breakage. The preheating chamber is roofed over and the furnace is charged through the roof, or the preheating chamber may be omitted when the roof over the resistors is the roof of the furnace. In that case charging may be done either through the center of the roof, the resistor rods being arranged as before, or the resistors may be placed all along the furnace and the charge introduced from the side, below the resistors.

The resistors are to extend clear through the furnace and are to be freely movable to allow expansion and contraction without breaking.

On account of the complicated construction in the hot zone and the many difficulties connected with the use of carborundum resistors the value of this furnace seems doubtful.

DIBZUWEIT FURNACE.

Some years ago a brass furnace of the tapping-hearth type of 800 pounds capacity was erected at the plant of the Baltimore Tube Co. by J. F. Dirzuweit. This was provided with three arched troughs for three-phase current, raised above the hearth on refractory piers and containing granular carbon resistors. Great trouble was encountered with "hot spots" in the resistor, and the furnace showed an excessive power consumption. After about 2 tons of metal had been melted the furnace was abandoned.

BAILY TILTING FURNACE.

Of all the furnaces so far described in this report many have existed only on paper, a few have had laboratory tests, still fewer have been tried out under foundry conditions, and not one has yet had any steady commercial use for brass melting. The next furnace on the list, the Baily,²³ has had commercial success. The present Baily tilting furnace is the outgrowth of the unsuccessful experiments made with the different types of former Baily furnaces previously described.

This Baily furnace, like its predecessors, uses a granular resistor. Hirsch²⁴ made an exhaustive study of granular resistors and came to the conclusion that any attempt to make a granular-resistor furnace for heating steel for forging, with the steel placed above the resistor, would fail if the steel had to be heated above 1,000° C. The lower part of the resistor became so hot that failure of the silicon-

²³ Baily, T. F., and Cope, F. T., U. S. Patents 1,176,018, Mar. 21, 1916, and 1,272,186, July 9, 1918; Baily, T. F., Resistance-type furnaces for melting nonferrous metals: Chem. and Met. Eng., vol. 21, 1919, p. 11; Anonymous, Note on operation of electric furnaces: Elec. World, vol. 71, 1918, p. 780.

²⁴ Hirsch, A., Electric furnaces for heating steel: Jour. Ind. Eng. Chem., vol. 6, 1914, p. 533.

carbide (carborundum) resistor trough ensued. Thornton²⁵ made similar experiments, found a temperature of 2,000° C. in the bottom of the resistor when the steel was heated to 1,000° C., and decided that 900° C. was the upper limit to which steel could be heated in such a furnace with safety to the granular resistor. He shows how the radiation of heat from the open top of the resistor trough forces the current to concentrate in the bottom of the trough. Dow²⁶ suggests that the resistor be so arranged with respect to the iron that the magnetic force set up would tend to bring the current to the top of the resistor.

Thomson has suggested²⁷ the use of the zigzag trapezoidal resistor previously described (p. 104) in the bottom of the heating trough, with the wider side down, to avoid concentration of current in the bottom of the trough and consequent overheating of it. Granular resistor is piled over the zigzag resistor and helps to pre-

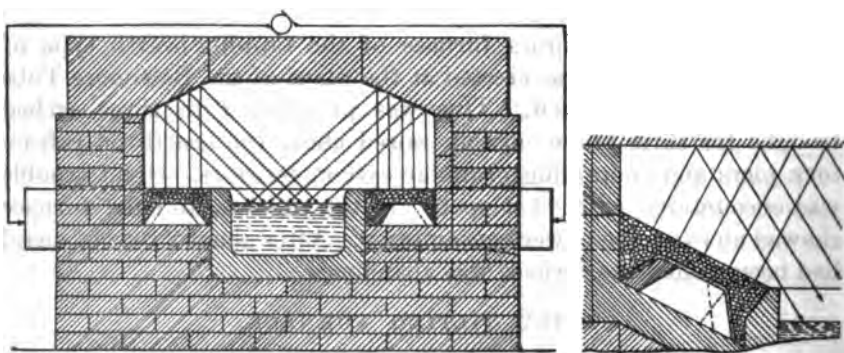


FIGURE 14.—Thomson's suggestion for improving the life of resistors in a reflected-heat furnace.

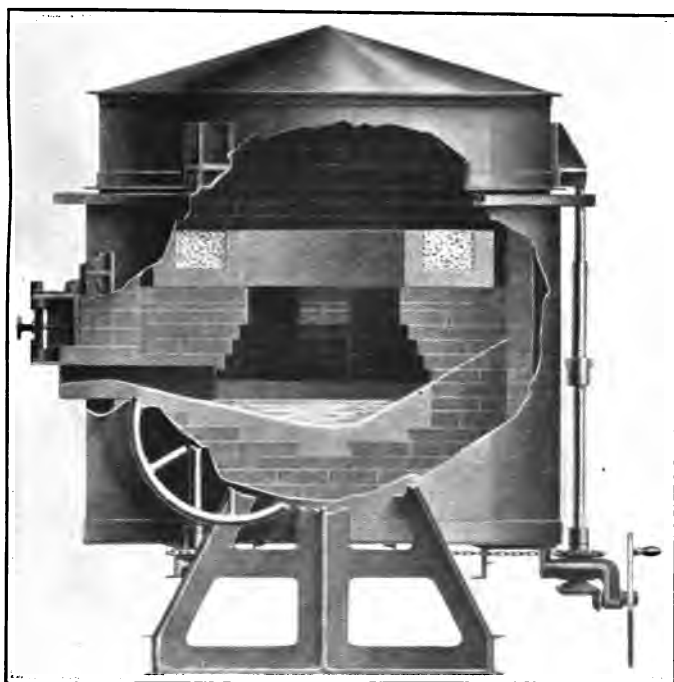
serve it from oxidation. Two adaptations of this design are illustrated in figure 14. Theoretically, this device should allow the use of a higher rate of power input without damage to the trough, and should thereby improve the efficiency of the furnace. Whether it will be a practical improvement can only be told by trial.

The standard 105-kilowatt Baily furnace, Plate VI, A, is an upright cylinder about 6 inches high by 7 inches in diameter, mounted on trunnions to tilt (a nose-tilting type is made also) and consists of the cylindrical furnace inclosure surmounted by a domed roof, which can be raised by a handwheel and gearing to allow charging fresh resistor material, and having the hearth in the bottom. A charging opening closed by a plug-type door is provided on the side, and in the lower part of this opening is the pouring spout. The source of

²⁵ Thornton, F., jr., A resistance furnace: Trans. Am. Electrochem. Soc., vol. 32, 1917, p. 257.

²⁶ Dow, A., U. S. Patent 1,161,634, Nov. 23, 1915.

²⁷ Thomson, J., U. S. Patent 1,351,977, Sept. 7, 1920.



A. STANDARD 105-KW. BAILY FURNACE.



B. TRUNNION-TYPE 105-KW. BAILY FURNACE.



NOSE-TILTING TYPE 105-KW. BAILY FURNACE.

heat is in a circular resistor trough, placed about midway between roof and hearth. Carbon electrodes enter the resistor trough from two sides of the furnace, the single-phase current thus dividing and passing through both sides of the circle in parallel. (See fig. 15.) The electrodes are large, so that the density of the current is low, and do not heat up enough to require water cooling. They are led in through a packing of carborundum fire sand to prevent an access of air to the electrode at any point where they may become hot enough to oxidize. The electrodes are stationary, being built into the refractory wall, and require no adjustment when the furnace is running. They are usually replaced whenever the resistor trough is renewed.

The fundamental point that makes possible the use of a granular resistor in the standard Baily furnace is the recognition of this crowding of the current to the bottom of the trough and arranging the trough to allow free radiation from bottom and sides of the trough as well as the top. The trough is raised off the furnace bottom by refractory piers. A free space is provided outside

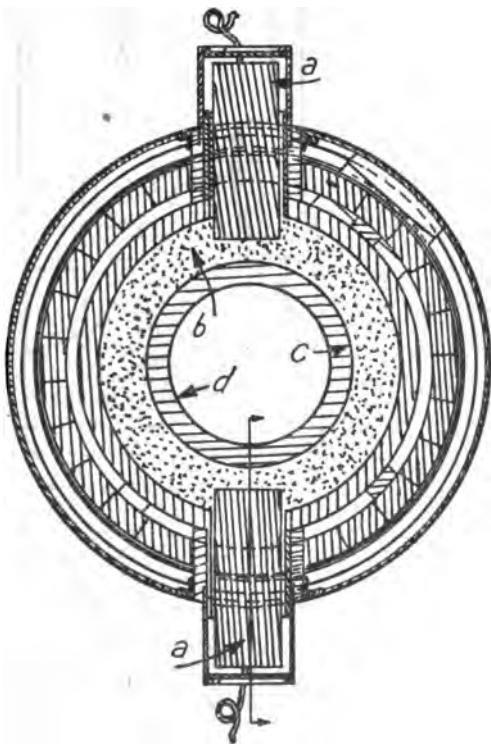


FIGURE 15.—Plan of electrodes and resistor trough of Baily furnace.

the trough between it and the furnace wall. Fairly free radiation is thus provided from all sides of the circular trough (see Pl. VI, A), although much of the radiation must be reflected by walls and roof before it can reach the hearth which lies below the resistor in the bottom of the furnace chamber. The Baily resistor has a radiating surface of more than 40 square feet. Now a granular resistor must have a fairly generous cross section; approximately 6 inches high by 8 inches wide for the inside dimensions of the trough are common dimensions, and with 2-inch walls the whole trough is about a foot wide. The resistor thus occupies a good deal of space, and as care is taken to allow free space outside the resistor the whole furnace becomes very large for the metal charge used in it, which in different

and a production on the best day's run given of 5,345 pounds of metal; but at a power input of 120 kw. the efficiency would be 74 per cent and the production would be 7,665 pounds. The Baily furnace is not supplied with a larger transformer and is not often run at a higher, more efficient power input, because the life of the resistor would be endangered and the reliability of the furnace would be decreased. It is very important that the furnace should not be overheated on account of the consequent injury to the resistor, as well as to the roof and walls, which have to reflect back so much of the radiant energy; Baily brass furnaces therefore are often equipped with a pyrometer in the roof, and the operator is rigidly instructed to reduce the power input when the roof reaches a certain temperature. The actual temperature depends on the location of the pyrometer. It may be placed through the roof to register the temperature of the chamber itself, or it may be embedded in the roof, where it will register a lower temperature. The furnaces are run at a temperature of 2,500° to 2,600° F. (1,375° to 1,430° C.) in the top of the chamber when the charge is poured at 2,000° to 2,100° F. (1,100° to 1,150° C.).

It can readily be seen that the large mass of brickwork gives a large heat storage and that the roof, walls, and resistor, being 500° F. (280° C.) hotter than the metal at the end of the heat, can continue the heating of the metal long after the power is cut off. Miller³⁰ states that it takes something like 15 minutes before the furnace begins to respond to a change in power input. The stored heat is so great that after running at full tilt all day the power can be cut down very low on the last heat of the day, the stored heat doing most of the work. This means that when the furnace is used on yellow brass, especially in rolling-mill work, where the metal must be poured nearly at its boiling point, if a delay occurs and the metal must be held in the furnace, the power must be at once cut off and scrap or gates added to the charge in order to keep the stored heat from heating the metal altogether too hot. One plant, operating on 70:30 brass poured at an average of 1,875° F. (1,025° C.), states that the first 200-pound ladle poured from an 800-pound heat averages 1,010° C. (1,850° F.) and the last 1,040° C. (1,900° F.) or higher.

Baily states³¹ that his furnaces "have an immense heat storage capacity in their refractory walls." It takes a good deal of energy to bring one of these furnaces to operating temperature. In a 10-

³⁰ Miller, D. D., The electric furnace as a medium for heating nonferrous metals: Jour. Am. Inst. Metals, vol. 11, 1917, p. 285; see also Benzinger, R. F., Are single-phase furnace loads profitable for the central station? The Electric Furnace, vol. 1, 1920, No. 3, p. 12.

³¹ Baily, T. F., The resistance-type electric furnace: The Electric Furnace, vol. 1, No. 2, February, 1920, p. 15.

hour operation the first 16 tons of yellow brass melted in the Bailly furnace installed at the plant of White & Bro. averaged 880.kw. h. per ton, including the power used in preheating the furnace. The sluggishness of the furnace means that it heats up slowly, as well as cools off slowly, and after cooling 14 hours it would take so long to bring it to full running temperature that production in a 10-hour melting day would be very low. Hence it is regular practice to have the night watchman throw the power on the furnace about 3 a. m. in order to have the furnace ready to melt at 6.30 a. m. The power used at night has to be paid for as well as that used while metal is in the furnace, so that no figures of power consumption for this furnace or for any furnace of a type that requires night heating should be given without also stating whether the night power is included.

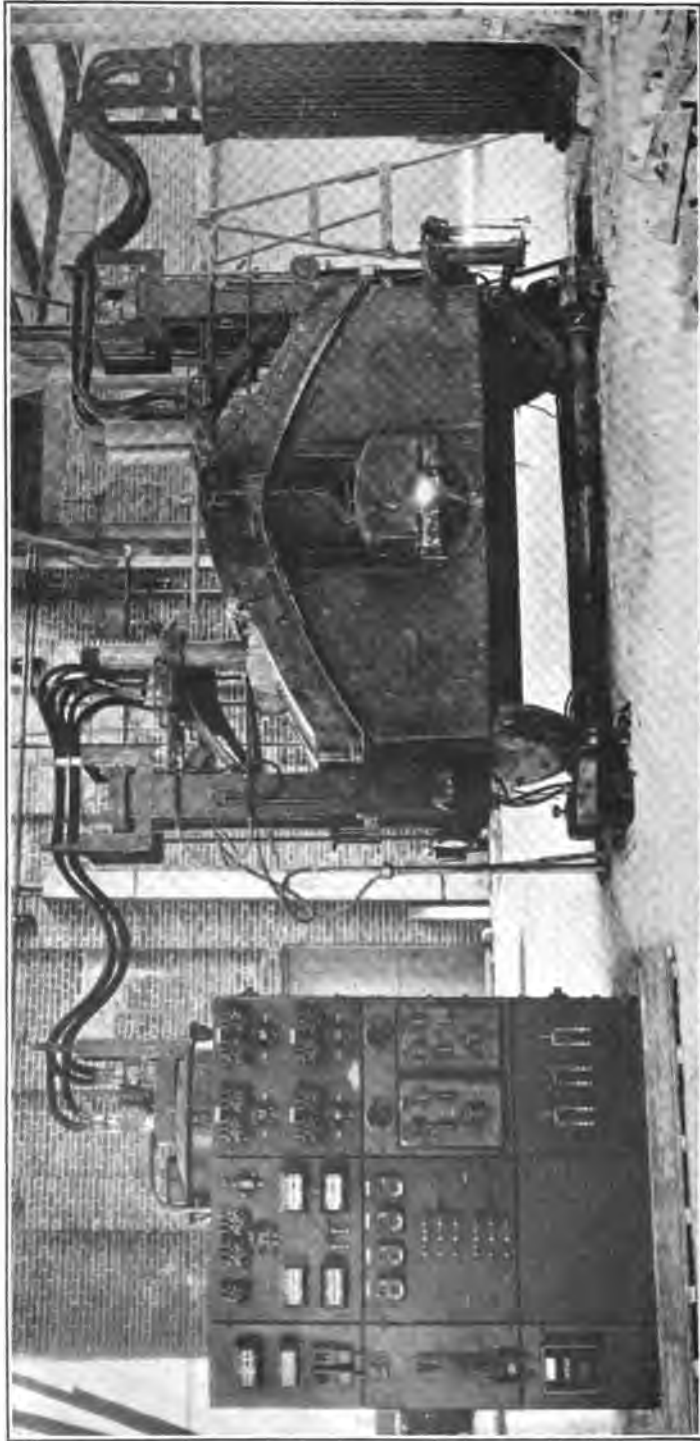
It follows that the furnace, though seldom so used in practice, is at its best on 24-hour operation and is under a decided handicap when used only on day shifts.

Besides the standard 105-kw. tilting furnace (see Pl. VI, *B*) mounted on trunnions—the type so far installed by most users—a nose-tilting type (see Pl. VII) of the same size has been designed for pouring direct into molds, one of which has been installed at the Washington Navy Yard, and a 300-kw. size has been designed, but no installations have yet been made. A 105-kw. “nose tilter” has been sold to Allen Everett Co., Ltd., of England.

A smaller, trunnion-mounted, tilting type of Bailly furnace of 50-kw. rating, shown at the right in Plate VIII, was put on the market in 1919. This type held up to 500 pounds maximum, having a rectangular shell and a long oval hearth. It was heated by a single straight resistor trough located directly over the bath, supported at the ends. A few furnaces of this type were installed, but the life of the resistor trough was so short that its use was found impractical and it was withdrawn from the market, a round, 50-kw., 500-pound furnace of the same general design as the 105-kw. size being supplied instead. Since this has nearly as much wall area as the 105-kw. furnace, its losses in radiation will be nearly as great, and the lost heat will form a large proportion of the 50 kw. supplied. The efficiency of this size, therefore, may be expected to be very low. No operating figures for this size on brass are yet available, but the relative behavior of different sizes of this type of furnace may be seen from data on melting aluminum (No. 12 alloy): The 105-kw. size at the Dayton Engineering Laboratories Co., on 425-pound heats—length of working day not stated, but apparently about 8½ hours actual melting, plus 1 hour preheat—gave an output of about



BAILY FURNACES SET UP FOR DEMONSTRATION. TWO TRUNNION-TYPE 105-KW. FURNACES AT THE LEFT, A NOSE-TILTING 105-KW. FURNACE AND CASTING TABLE IN THE CENTER, AND THE OBSOLETE RECTANGULAR 50-KW. FURNACE AT THE RIGHT



A 2,500-POUND 400-KW. GENERAL ELECTRIC RECTANGULAR FURNACE.

1½ tons per day at 675 to 625 kw. h. per ton. The 50-kw. rectangular size at the plant of Landers, Ferry & Clark, on 150-pound heats, 9 hours actual melting, 4 hours preheat, gave an output of ½ ton per day at 850 kw. h. per ton. The net metal loss is given as 0.7 per cent.

Since the 105-kw. size runs at around 600 kw. h. per ton of brass on a 9-hour melting day, it would be expected by analogy with the results given above on aluminum that the 50-kw. size, working on brass, with a 9-hour melting day, would require around 750 kw. h. per ton.

One plant, melting red brass in the rectangular 50-kw. Baily furnace, on a 7-hour working schedule, the furnace being preheated at night, averaged three heats of 525 pounds, or about ¼ ton per day, at a total power consumption of 730 kw. h., i. e., about 900 kw. h. per ton. The gross metal loss is given as 1 per cent and the life of the resistor trough as only 100 heats.

The Baily furnace is also built in a stationary tapping type, with a hearth lying between two straight resistor troughs. One furnace of 500-kw. rating has been in use on aluminum, and one of 1,000 kw. has been used in melting down electrolytic zinc cathodes. A 600-kw. furnace of this type for brass has been designed, but none yet installed. All these furnaces are supplied with energy through a variable-voltage transformer having a number of taps on the secondary, to which connection is made by an oil switch operated by hand. Lehr²² has patented a system by which the regulation of such a furnace may be made automatic. Short²³ describes an automatic control as supplied by the Westinghouse Co. to 40 furnaces, each of 1,100 kv. a., for zinc smelting in Norway. (These are not Baily furnaces.) Hand control of the Baily furnace is simple, and it is doubtful if the application of an expensive automatic control system to the 105-kw. size would normally be justified.

METALLURGICAL RESULTS IN THE BAILY FURNACE.

METAL LOSSES.

Baily²⁴ cites some tests on yellow brass (68 per cent Cu, 32 per cent Zn) made up of half scrap, half new metals, in which the net metal loss, taken by weighing the charge, deducting the nonmetallic in the charge, and weighing the product, was 1.39 per cent when the zinc was added with the rest of the charge at the start of the heat and 0.56 per cent when it was added at the end in the usual way.

²² Lehr, E., U. S. Patent 1,320,071, Oct. 28, 1919.

²³ Short, P. B., Stop-induction regulator for electric furnaces: Trans. Am. Electrochem. Soc., vol. 37, 1920, p. 35; see also Pierce, R. T., U. S. Patent 1,367,089, Feb. 1, 1921.

²⁴ Baily, F. T., work cited.

In a test reported by one foundry on a leaded bearing bronze (79 per cent Cu, 16 per cent Pb, 5 per cent Sn, trace P) there was charged:

	Pounds.
Ingots.....	728
Gates carrying some sand.....	5, 198
Lead.....	192
Copper.....	938
Tin.....	72
Total.....	7, 126
Metal poured.....	7, 009
Gross loss (1.63 per cent).....	117

The foundry estimates that 26 pounds (5 per cent) of sand adhered to the gates, making the gross loss 91 pounds on 7,100 pounds metallic charged, or 1.28 per cent corrected gross loss. Ninety-two and one-half pounds of skimmings, from which it is estimated that 40 per cent, or 37 pounds metal, would be reclaimed were also obtained, making the net loss 54 pounds on 7,100 pounds of metallic charged, or 0.76 per cent. The furnace replaces an open-flame oil furnace and makes decided metal savings.

Other tests, cited in the advertising literature of the Electric Furnace Co. are as follows:

TABLE 28.—*Results of runs with Baily furnace.*

Material.	Metal charged.	Metal poured.	Loss.	Loss.
	Pounds.	Pounds.	Pounds.	Per cent.
Yellow brass, gates, ingot borings.....	23, 429	23, 361	78	0.30
Yellow brass, scrap.....	181, 304	179, 833	1, 471	.81
Do.....	28, 320	27, 941	379	1.30
Phosphor bronze.....	35, 733	35, 686	87	.24
Do.....	6, 033	5, 991½	41½	.68

One plant reports that in regular operation the total metal loss in melting bronze of 89 per cent Cu and 11 per cent Sn in the Baily furnace, including all losses between charge and finished product, whether in melting or not, is 2.5 per cent. Another plant making an ordinary red brass from all scrap and borings declares that the net metallic loss, by test, is 1 per cent and is convinced that the furnace runs regularly at that figure. This plant takes care to plug and lute the spout and door tightly to avoid oxidation and volatilization. It asserts that on the material used the loss in coke fires can not be kept below 5 per cent in regular operation, though that figure can be surpassed on careful tests.

Another plant, running on 70:30 brass for sand castings made from scrap and borings, estimates the metallic loss to be 1 per cent

against about 5 per cent in coke fires. In still another plant which makes a cast yellow brass with about 20 per cent Zn, largely from scrap, the gross loss between dirty scrap and hot metal weighed in the ladle was 2.8 per cent on the first few heats, while the new hearth was becoming saturated, and fell on the next 20 tons melted to 1.25 per cent. It is estimated by this plant that there was 0.25 per cent dirt, oil, etc., in the scrap, and an additional 0.25 per cent would have been recovered had all the spillings been picked up. The true metallic loss therefore was not over 0.75 per cent. In coke-fired practice on this alloy the average net loss was said to be 3 per cent, and some actual tests showed 3.44 per cent.

UNIFORMITY OF PRODUCT.

The behavior of the furnace as regards metal losses is excellent, but is not so good as regards uniformity of composition of the product. Baily³⁴ gives figures for 30 heats of yellow brass (about 69 per cent Cu, 31 per cent Zn), the copper in which was determined by sampling the first and last bars poured from a 750-pound heat. The copper averaged 0.70 per cent higher in the last bar than in the first. In three heats there was no variation, in only five others was the difference below 0.25 per cent, in two between 0.25 and 0.50 per cent, in eleven it was from 0.50 to 1 per cent, in eight it was between 1 and 2 per cent, and one heat showed a variation of from 68.67 per cent in the first bar to 70.71 per cent on the last, or 2.03 per cent.

These figures were obtained on a test of one of the first Baily furnaces commercially installed, made by the makers of the furnace and not by the rolling mill. The metal was poured colder than it would have been had a regular caster done the pouring, and the importance of stirring was evidently not understood. The figures do not mean that uniform metal can not be poured from the Baily furnace or from other hearth furnaces similarly heated by radiation from above, but they do indicate a tendency to imperfect mixing and show that it is important for the metal to be thoroughly stirred before pouring. At the plant where these tests were made differences of more than 2 per cent in copper between the first and the last of the pour have been noted even with careful stirring, and uniformity is considered harder to obtain than in crucible practice.

According to Hess,³⁵ in melting steel in the Heroult direct-arc furnace there is little circulation, and heavy alloys are likely to settle, causing segregation and lack of homogeneity, which, he says, should be guarded against by systematic and vigorous stirring. As there is even less tendency toward circulation in furnaces like the Baily

³⁴ Baily, T. F., Work cited, p. 291.

³⁵ Hess, H. L., Electric furnaces as applied to steel making: Mech. Eng., vol. 41, 1919, p. 245; Chem. Abst., vol. 14, 1920, p. 16.

that melt by radiated heat than in direct-arc types, Hess's advice appears worth heeding in the operation of brass furnaces that do not stir the metal automatically, as well as in the operation of steel furnaces.

QUALITY OF PRODUCT.

The quality of the metal, from the testimony of three different users, is no worse and no better than good metal from coke-fired crucible furnaces. If careful attention is paid to stirring, the metal is thought to average more uniform, merely because the use of a larger charge than in a crucible tends to even up its regularity in composition of scrap used. In one case the metal was thought to machine more easily.

ADAPTABILITY.

The furnace can be drained completely and successive heats of different alloys made. On account of the necessity for keeping down the chamber temperature and the power input, in order to safeguard the resistor, the furnace will be less efficient on and less adaptable to the alloys that have to be poured at a high temperature. Few Baily furnaces regularly produce metal that would be poured above 1,160° C. (2,100° F.).

The furnace will melt any ordinary brass or bronze (probably not Monel metal), but is at its best on yellow brass and on red brasses and bronzes that do not have to be poured too hot.

Two different plants report that on charges of all yellow borings, especially if dirty and oily, the Baily furnace does not give satisfactory results, because such charges require too much stirring and rabbling. Charges containing not over 50 per cent yellow borings can be handled.

Although criticizing the Baily furnace for low efficiency—that is, for excessive power consumption per ton—one refining plant asserts that the Baily furnace should have a use in a refining plant in handling light or “irony” brass, as the melt can be skimmed to remove iron without interrupting the current.

RELIABILITY.

The reliability of some of the early installations of Baily furnaces was affected by the inability of the transformers to take their rated load without heating up, but this defect has been overcome.

The life of the lining and roof (corundite brick) is good, provided care is taken to keep the chamber temperature below a safe point. In one furnace, supplied with a pyrometer in the roof, that was inspected after 18 months operation and after the melting of more than 1,000,000 pounds of 70:30 brass, the lining was still in good shape. The resistor trough had to be renewed after about 750,000 pounds had been melted.

Another furnace, on 16-hour operation on red brass, had required no repair in melting about 600,000 pounds. Another, on 9-hour operation on yellow brass, had required one new hearth lining in melting 350,000 pounds, the hearth probably being damaged by throwing in heavy ingots, but no other repair besides patching the spout. The hearths are usually rammed up from carborundum fire sand, plus fire clay and water glass or some similar mixture.

One user of the Baily furnace, melting a leaded bronze with 10-hour operation, concluded that the hearth and resistor trough must be replaced after melting 1,000,000 pounds, and calculated the upkeep of the furnace, at 1919 prices, to be 50 cents per ton. Still another user has said that the trough averages a life of 250,000 pounds, but that with greater care in operation this figure can probably be doubled.

Other users had great trouble with resistor troughs when they were rammed up in a form inside the furnace from carborundum fire sand with a water-glass binder, the troughs often failing after only a very few heats; sectional troughs are now molded up and baked by the makers and sent to the user to be set up like so much brickwork. These appear to be giving good service.

Electrodes are usually replaced whenever a trough is replaced. The granular resistor material—electrode carbon broken to $\frac{3}{8}$ -inch or $\frac{1}{4}$ -inch mesh—slowly burns away and must be replenished, about 25 pounds being normally added every Sunday according to one user and at the rate of 10 pounds per day according to another. This granular carbon costs in the neighborhood of 5 cents per pound. The resistor carbon used must be as free as possible from ash or the ash will gum up the resistor, interfere with the flow of current, and endanger the trough.

In some plants where much skimming and rabbling has been done the piers (usually silica brick or corundite) have occasionally been damaged by the operator's careless poking. Generally speaking, the reliability of the furnace, when operated under pyrometric control, is good.

RATES OF PRODUCTION AND THERMAL EFFICIENCY.

As the rate of production and the thermal efficiency are closely related, they will be considered together.

Baily³⁰ has given figures on a week's run of a furnace at the Baltimore Copper Smelting Rolling Co. on 68:32 brass which show the following averages:

Average number working hours per day-----	8 $\frac{1}{2}$
Average number heats per day-----	5 $\frac{1}{2}$
Average charge-----pounds--	752
Average kw. h. per heat (not including night heating)-----	128.4

³⁰ Baily, T. F., Resistance-type furnace for melting brass: Trans. Am. Electrochem. Soc., vol. 32, 1917, p. 155; Brass World, vol. 13, 1917, p. 337.

While not stated in this paper both Mr. Baily and the superintendent of the plant²⁷ state that power was kept on the furnace all night at the rate of 44 kw.

This gives:

Average kw. h. used at night (15½ hours), at 45 kw., per night.....	871
Average kw. h. used in day (8½ hours), at 77 kw., per day....	685
Total kw. h. per 24 hours.....	1,856
Average output, 8½-hour working day..... pounds..	4,011
Average kw. h. per ton.....	675

In the regular operation of this furnace on 70:30 brass, poured at 1,025° C. (1,875° F.), with power kept on all night at 45 kw., the plant reported an output of 4,100 pounds (five heats, 820 pounds), with an average kw. h. consumption of 650 per ton. Individual heats at the end of the day, when the furnace was fully hot, were made with 130 kw. h., that is, at the rate of 810 kw. h. per ton for continuous 24-hour operation. With the furnace fully hot, copper poured at 1,200° C. (2,200° F.) required 333 kw. h. per ton, with an average power input of 100 kw. The rate of production on 24-hour operation would be about 15,000 pounds brass a day when the furnace is run at a 77-kw. input and 12,000 pounds copper a day when running at a 100-kw. input. The transformer supplied with this early furnace would not give 100 kw. steadily without undue heating.

This particular furnace is no longer in operation, the plant having gone out of the brass business entirely. The company asserts²⁸ that the furnace can not be considered for the melting of copper in competition with the regular reverberatory furnaces taking 200 to 250 tons per charge; though the application of the principle involved in the melting of large charges is interesting, it introduces too many problems for serious consideration.

A Baily furnace was tried but discarded by the Bridgeport Brass Co.

Another user gives the following figures for bronze (89 per cent Cu, 11 per cent Sn) poured at 1,150° C. (2,100° F.). Average weight of charge, 800 pounds; seven heats in 10 hours, 5,500 pounds output per day; power used, 1,100 kw. h. in 10 hours; average output, 110 kw., 550 kw. h. in 14 hours (night); average input, 39 kw.; total kw. h., 1,650, or 600 kw. h. per ton.

Another plant working on 70:30 brass made from scrap and borings for sand castings reports, on 9-hour operation (6 a. m. to 4 p. m.), power input held at 100 to 105 kw. (power off eight minutes while pouring) with night heating at about 75 kw. aver-

²⁷ Personal communications.

²⁸ Rouse, E. W. Personal communication, Jan. 10, 1920.

age from 2.30 a. m. to 6 a. m., week days, except Saturday, and from 6 p. m. Sunday to 6 a. m. Monday; the output can be brought, as a maximum, to five heats of 1,500 pounds each. The average over-all power consumption is said to be 425 kw. h. per ton.

One man operates two furnaces in this plant by staggering the operation of the two furnaces, so that two heats are not ready to pour at the same time. The charges are brought to him and the metal taken away for him.

At another plant, operating two 8-hour shifts on leaded bronze, the furnace being preheated four hours besides—that is, power being off only four hours per day—the average output in the 16-hour melting period is 4 tons, and the power consumption runs from 500 to 575 kw. h. per ton.

Still another plant, operating two 10-hour shifts, with three hours preheating, on red brass, gets an average output of 4½ tons, though up to 6½ tons have been obtained. The power consumption averages 550 kw. h. per ton.

One plant, melting 60:40 brass 24 hours a day, gets an output of 8 tons per day at 310 kw. h. per ton, while another has obtained a power-consumption figure on yellow brass, 24-hour operation, of 264 kw. h. per ton, but states that this is lower than can be expected as an average.

One plant, making sand castings of yellow brass (20 per cent Zn) from scrap got six heats of 1,000 pounds in 10 hours by using 400 kw. h. from 2 a. m. to 6.30 a. m. and 1,100 kw. h. from 6.30 to 4.30, or 1,500 kw. h. for 3 tons, or 500 kw. h. per ton. This figure is rarely reached because of delays in pouring, etc., 550 kw. h. being more common and 600 kw. h. the average. The best possible figure on 10-hour operation at this plant is thought to be 450 kw. h. per ton.

At another plant, on a test to see what maximum output could be made on an alloy of 79 per cent Cu, 5 per cent Sn, 16 per cent Pb, trace P, charge 73 per cent gates, 10 per cent ingot, 17 per cent new metal, the following figures were obtained:

TABLE 29.—*Test of Bailly furnace on leaded bronze.*

Heat No.	Time.	Weight of charge.	Kw. h. used.
Preheat.....	2.00 a. m. to 6.30 a. m.	<i>Pounds.</i> (a)	375
1.....	6.30 a. m. to 8.00 a. m.	1,102	200
2.....	8.00 a. m. to 10.00 a. m.	1,220	200
3.....	10.00 a. m. to 11.40 a. m.	1,200	175
4.....	11.40 a. m. to 1.00 p. m.	1,202	175
5.....	1.00 p. m. to 3.00 p. m.	1,200	175
6.....	3.00 p. m. to 5.00 p. m.	1,202	175
Total.....		7,126	1,475

a Empty.

Average kw. h. per ton.....	413
Average time to charge and pour.....minutes.....	20
Average power input (preheating, 4½ hours).....kw. h.....	83½
Average power input (working time, 10½ hours).....kw.....	105

The last heats are at the rate of 290 kw. h. per ton on 24-hour operation. The transformer temperature rose from 35° C. at 2 a. m. to 59° C. at 5 p. m. On another day's run, using 1,500-pound charges of leaded phosphor bronze, the following result was obtained:

TABLE 30.—*Test of Baily furnace on leaded bronze, 1,500-pound charges.*

Heat No.	Time.	Weight of charge.	Kw. h. used.	Kw. h. input.
	H. M.	Pounds. (a)	b 375	
Preheat.....				
1.....	2 20	1,516	246	105
2.....	2 0	1,509	232	126
3.....	1 35	1,505	203	128
4.....	1 30	1,503	206	137
Total.....	7 25	6,033	c 1,282	122

^a Empty.

^b Data taken from advertising literature of Electric Furnace Co. Power used on preheat not given, assumed same as on preceding test. The last two heats indicate 205 kw. h per 1,500 pounds, or 275 kw. h. per ton on 24-hour operation, run partly on stored heat from the lining.

^c 425 kw. h. per ton.

A recent account²² of the operation of a Baily furnace on yellow brass (75 per cent Cu, 20 per cent Zn, 4 per cent Pb, 1 per cent Zn), poured at 1,200° C. (2,200° F.) declares that by preheating from 3.30 a. m. to 6 a. m. four heats of 1,500 pounds each are taken off by 3.30 p. m. One more heat could be taken by running to 5 p. m. The power consumption over a month's run (70 tons) was 572 kw. h. per ton. The furnace is said to give best results if run at 120 kw., a higher power input shortening the life of resistor trough and roof and a lower one increasing the power consumption per ton. The resistor trough and hearth last about six months; that is, for the melting of some 400 to 450 tons.

In a comparison of melting costs between crucibles and the electric furnace the upkeep of the Baily furnace is put at 60 cents per ton. The labor cost for electric melting is half what it was for crucible melting, and the metal loss is reduced from 5 per cent to 1 per cent. The costs are given as \$19.20 per ton in crucibles (of which \$6.40 is due to the loss of 4 per cent more zinc) and as \$15.20 in the electric. Power cost 1.41 cents. per kw. h., or \$8.06 per ton. It is the saving in metal that makes the furnace economical. It should be noted that this cost tabulation does not include interest and depreciation on the investment in the furnace.

The red brass melted at this plant is still melted in crucibles, because the metal loss in the crucibles is only about 2 per cent, so that

²² Anonymous, Electric melting shows economy: Foundry, vol. 47, 1910, p. 845; Anonymous, Operating data on electric brass furnace: Elec. World, vol. 75, 1920, p. 11.

the electric furnace would save only 1 per cent and the cost balance, which is swung for the electric on yellow brass solely by the saving in metal losses, is against the Baily furnace on red brass.

FURNACE OF ANACONDA COPPER MINING CO.

At Great Falls, Mont., a standard Baily 105-kw. furnace is installed in the brass foundry of the Anaconda Copper Mining Co. This firm uses large amounts of electric power generated by water power for general purposes, for the operation of ferro-alloy furnaces, and for the electrodeposition of zinc. When hydroelectric power is bought under such circumstances it is usually paid for, not by the kw. h. actually used, as is the case in most brass foundries, but by the kilowatt year; that is, by the most simple form of power contract a certain amount a year is paid for each kilowatt the plant in question is equipped to draw from the hydroelectric station.

After the furnace at Great Falls had been used intermittently for 15 months the lining was still in excellent condition, one resistor-trough replacement having been made and the second being nearly ready for renewal. The melting rate, after the furnace is fully hot, is 1,800 pounds in $3\frac{1}{2}$ hours; that is, 515 pounds per hour on the basis of continuous operation. As high as 2 tons have been charged into the furnace at one time. The metal loss is under 2 per cent.

The power consumption averaged 2,900 kw. h. per ton during four months' intermittent operation, in which 42.4 tons were melted. It is said that if the furnace were operated continuously 24 hours per day it could probably equal the guarantee, made on the basis of continuous operation, of 400 kw. h. per ton. The furnace is said by the Anaconda Co. to be very satisfactory for their foundry work. The work is intermittent, only about 15 tons of metal being melted per month, on the average, and only about 10 tons per month in the period on which the figures of power consumption were computed.

Power is kept on the furnace all the time, the increased power consumption being offset by the time saved in bringing this sluggish type of furnace up to working temperature if it is allowed to cool.

At the rate of 515 pounds per hour, the four months' melt of 84,800 pounds would take only 165 hours out of 2,880, or 6.6 full 24-hour days out of 120 days.

To keep the furnace hot all the time would require an average input of about 40 kw. for the whole 2,880 hours, and if the power input was raised by 50 kw. to an average of 90 kw. during the 165 hours of actual melting the power consumed would be $(40 \times 2,880) + (50 \times 165)$, or 123,650 kw. h. used to melt 84,800 pounds, or 2,900 kw. h. per ton.

Had this amount of power been paid for on the kind of a power contract used by most purchasers of a single Bailly furnace it would probably cost about 1.4 cents per kw. h., or about \$40 per ton for power alone.

On so large a connected load as that at Anaconda, where there would be several times 100 kw. leeway between the connected kw. of the high-tension transformers and the average load, it would make no difference in the actual cost of power whether the Bailly brass furnace were run or not. Even for cost accounting it might be calculated that only the 40 kw. used all the time should be charged against the furnace if the foundry took care to do the actual melting at a time when the rest of the plant was not drawing its full load.

Forty kw. for 4 months at, say, \$25 per kw. year would cost \$333.33 for 42.4 tons, or \$7.85 per ton on a melt of 10.6 tons per month, a figure which is less than the \$8.06 per ton in the furnace last commented on, which used 572 kw. h. per ton at \$1.41 per kw. h. on a melt of 70 tons per month.

All these data tend to show that where power is high the Bailly furnace should be used for melting 16 to 24 hours per day, as it is at a disadvantage on account of its high-heat storage on 10-hour melting; where power is extremely cheap, it can be satisfactorily used on 10-hour melting or even as intermittently as the Anaconda installation. The resistor troughs have had to be replaced twice in 21 months, while melting an average of 15 tons per month or, say, 157½ tons average per trough, on this intermittent service.

Omitting the Anaconda furnace on account of the unusual operating conditions, the commercial results on the 105-kw. Bailly furnace can be summarized as in Table 31.

TABLE 31.—Summary of Bailly furnace operation.

Alloy.	Firm.	Kind of operation on which figures are based.	Hours furnace pre-heated per day.	Hours furnace used per day in actual melting.	Weight of charge.	Output per day.	Kw. h. per ton.
Yellow brass ^a	Baltimore Copper Smelting & Rolling Co.	Average.....	15	9	Pounds. 800	Tons. 2	680
Do.....	McRae Roberts.....	Best.....	2½	9	1,500	3½	435
Do.....	Michigan Smelting & Refining Co.	do.....	0	24	(^b)	(^b)	c 264
60:40.....	United States Copper Products Co.	Average.....	0	24		8	310
Brass, 20 per cent zinc.	Hays Manufacturing Co.	One month, 70 tons.	2½	9½	1,500	3	572
Red brass.....	Capital Brass Co.....	Best.....	0	16	1,500	46	(^b)
Lead bronze.	Buck Motor Car Co....	Test.....	4½	10½	1,200	3½	413
Do.....	Nolte Brass Co.....	Average.....	4	16	1,500	4	500 to 575
Bronze ^c	Lumen Bearing Co.....	do.....	14	10	800	28	600
Red brass.....	Penberthy Injector Co.	do.....	3	20	1,500	74½	530

^a Metal poured at 1,025° C.

^b Not stated.

^c This power figure is said to be lower than can be expected as an average.

^d Output given is maximum.

^e Metal poured at 1,150° C.

^f Up to 6½ tons have been gotten out in two 10-hour shifts.

The standard Baily furnace rated at 105 kw. is thus seen to produce, on 10-hour operation, on metal poured at or below $1,150^{\circ}\text{C.}$, from 500 to 800 pounds per hour, $2\frac{1}{2}$ to 4 tons per day, and to use 425 to 600 kw. h. per ton. On 24-hour operation it would produce 6 to 10 tons at 275 to 350 kw. h. per ton. These variations depend partly on the pouring temperature needed and on the rate of power input, but even more so on the efficiency of operation and the speed of charging and pouring.

It has been found, in the practice of one plant, highly essential to operate the Baily furnace without a slag. If a dirty charge is used containing a lot of nonmetallic material, such as sand adhering to gates, so that as soon as the metal is fluid, it becomes covered with a layer of fluid or crusty slag, the furnace should be opened and the slag scraped off. If the slag or crust is left on it prevents the rapid absorption of heat reflected on it by the roof, slows up production, and tends to require running the roof too hot. It acts, in baffling the flow of heat, much as a crucible wall would act.

ELECTRICAL CHARACTERISTICS.

The single-phase 105-kv. a. transformer is usually arranged to give 55 to 110 volts by 5-volt steps. The furnace, in normal running, after it is heated up, takes about 80 volts, 1,300 amperes. Control is obtained by a selective oil switch to the transformer taps. Continual changing of voltage is not necessary, but the watt meter must be watched in order not to allow the power input to rise or to fall off too much, as the furnace has a general drift toward taking a higher or a lower power input, with any impressed voltage, according to whether the resistor temperature is rising or falling:

The power factor is close to unity, and there are no sudden surges on the line, the variation being between zero and 100 kw., approximately, when the switch is opened and closed to change the voltage.

COST OF THE BAILY FURNACE.

The prices of the Baily furnace complete in April, 1920, were as follows: 105-kw., about \$9,400; 75-kw., about \$6,500; 50-kw., about \$5,000.

BAILY FURNACES MELTING ALLOYS OTHER THAN BRASS OR BRONZE.

The Lumen Bearing Co. uses one of the 105 kw. regularly and another intermittently on Lumen metal, a bearing alloy of about 85 per cent Zn, 10 per cent copper, 5 per cent Al. This is poured at about 700°C. ($1,300^{\circ}\text{F.}$). An average of $5\frac{1}{2}$ heats of 1,000 pounds each is obtained per 10-hour working day, 660 kw. h. being used during the

day and 330 during the night to keep the furnace hot. The furnace now regularly used for Lumen metal was bought secondhand, having been tried out by the Aluminum Co. of America for making up aluminum alloy ingot from pig aluminum and discarded, largely, it is understood, because of its being too small in size for such purpose. A 75-kw. Baily furnace has recently been sold to the Norsk Aluminum Co., Norway, for melting aluminum. No data are available on this.

Miller⁴⁰ describes a 500-kw. tapping-type Baily furnace used for remelting aluminum pig poured at about 850° C. (1,560° F.). It has two straight resistors, one on each side of the hearth, which holds 3 to 4 tons of aluminum. Such a furnace may be made to take three-phase current if desired.⁴¹

The furnace was operated 24 hours a day; capacity of production was 20 to 24 tons. Since it is rated at 500 kw., it could use around 12,000 kw. h. per day, or at the rate of about 500 kw. h. per ton.

Miller, however, claims that 265 kw. or 64,000 kw. h. per day, 320 kw. h. per ton, will do the work on 24-hour operation. From the data of Richards⁴² it appears that to raise aluminum to about 700° C., or 150° C. below the temperature given by Miller, will take 300 kw. h. per ton, at 100 per cent thermal efficiency.

Laschtschenko⁴³ shows that at about 700° C. the total heat in the molten aluminum is about 300 calories per gram; that is, about 320 kw. h. per ton. The presence of 8 per cent of copper in the usual No. 12 alloy would reduce the heat needed somewhat, but it is evident that, roughly, 300 kw. h. per ton of No. 12 aluminum represents 100 per cent efficiency. In a furnace of 60 per cent efficiency 500 kw. h. per ton would actually be required. It therefore seemed very doubtful if the 320-kw. h. figure were correct. Hence the matter was referred to the superintendent of the plant operating the furnace, who replied as follows:⁴⁴

Under best conditions and continuous operations we can melt 1 ton of aluminum per hour with a power consumption of slightly less than 500 kw. per ton.

We have had some difficulty with the furnace due to inherent defects or objectionable features in the design. The resistor troughs are a source of trouble, failing frequently. The transformer supplied with this furnace was built by the Electric Furnace Co., and has given us some trouble. It is

⁴⁰ Miller, D. D., The remelting of aluminum pig in the electric furnace: *Chem. and Met. Eng.*, vol. 19, 1918, p. 251.

⁴¹ See Baily, T. F., and Cope, F. T., U. S. Patent 1,322,749, Nov. 26, 1919.

⁴² Richards, J. W., Electric power required to melt metals; *Trans. Am. Brass Founders' Assn.*, vol. 4, 1910, p. 93.

⁴³ Laschtschenko, P. N., Specific heat of aluminum: *Jour. Russ. Phys. Chem. Soc.*, vol. 10, No. 56, 1914, p. 311; see also Merica, P. D. Aluminum and its light alloys: *Bureau of Standards Circ.* 76, 1919, p. 27; see also Wüst, E., Meuthen, A., and Durrer, E., Thermal constants of technical materials, *Instrumentenkunde*, vol. 39, 1919, p. 294; *Chem. Abstr.*, vol. 14, 1920, p. 1241.

⁴⁴ Vail, A. E. personal communication, Jan. 21, 1920.

"Number One" in their series and I am inclined to think could be considerably improved upon in design. We purchased this furnace primarily for making high zinc alloys of aluminum. We found that there was a noticeable decrease in the zinc losses over an oil fired or coal fired open hearth furnace. On the regular run of aluminum we could see no appreciable decrease in the melting loss. It was only in the case of zinc alloys that this difference was appreciable.

Our operating conditions are such that we could not keep the furnace in continuous operation. It has a very high heat capacity and requires a good deal of power to bring the furnace up to the proper operating temperature. When operated intermittently this is a decided drawback, since a large proportion of the power used is consumed in heating up the furnace. Over a period of time we found that under such conditions the power consumption was approximately 700 kw. per ton of metal melted. As a consequence we have practically abandoned the use of this furnace for melting aluminum. At the present moment we are seriously considering rebuilding the furnace for use as a pre-heating furnace for heating ingots for the rolling mill. The furnace has the decided advantage of close heat control and when operated at temperatures in the neighborhood of 1,000 to 1,200° F. it will stand up and give good service for a considerable period of time. We have an annealing furnace operating under such conditions which is giving very satisfactory results, but we can not say that we have been well satisfied with the performance of the melting furnace.

In a test, not made primarily for taking the figures of power consumption, on a standard 105-kw. Bailly furnace, which was attended by a representative of the bureau, the furnace had been kept hot for some 12 hours beforehand, a power input of about 30 kw. being used in that period. The roof temperature at the start was 1,780° F. (970° C.); 100 pounds of No. 12 alloy (92 per cent Al, 8 per cent Cu) were melted and raised to 850° C. in 1 hour and 25 minutes, the roof temperature then being 1,825° F. (995° C.).

The furnace was run at about 40 kw. The power was then cut off, some metal poured, and the furnace left without power for 1 hour and 10 minutes, when the roof temperature was 1,850° F. (1,010° C.) and the metal was at 910° C., showing the great heat storage of the furnace. The power was then put on again, 50 pounds more of the alloy added, more metal was then poured, and after 1 hour and 40 minutes the charge was at 980° C.

In the whole 4 hours and 25 minutes, 192 kw. h. were consumed in melting and superheating 150 pounds of the alloy, in supplying radiation losses while the furnace was held at about the temperature required in melting aluminum regularly, and in heating the furnace, at the end, to somewhat above the initial temperature. The actual melting of the aluminum would take some 22 kw. usefully applied. Hence 170 kw. supplied the radiation losses for about $4\frac{1}{2}$ hours. If it thus takes some 40 kw. to hold the furnace at the proper temperature without doing work, a 100-kw. furnace should utilize about 60 kw.; that is, should melt about 400 pounds per hour, as 300 kw.

usefully applied theoretically melts a ton. We should then expect on 24-hour operation that the 100-kw. Baily furnace would use 500 kw. h. per ton of aluminum and that on 10-hour operation it would require at least 600 kw. h. per ton.

Figures have already been given on the performance of the 105-kw. Baily furnace, showing 675 to 625 kw. h. per ton, and on the rectangular 50-kw. furnace, showing 850 kw. h. per ton, both on a 9-hour melting day. The net metal loss on 105-kw. furnace, melting No. 12 alloy, was estimated, but not actually determined, at 0.3 per cent, and the average net loss in the 50-kw. furnace was 0.7 per cent.

BAILY FURNACES FOR MELTING ZINC.

A 75-kw. Baily furnace has recently been sold to the Ilen Smelt-
werk, of Norway, for melting zinc. No operating data are available.

The 1,000-kw. tapping-type furnace installed by the Anaconda Copper Mining Co., for melting electrolytic zinc cathodes is claimed to operate, on 24-hour service, at 75 kw. h. per ton. In January, 1918,⁴⁵ out of 5,112,335 pounds melted, 4,846,710 pounds were poured, the dross being 5.20 per cent. In four days in January, 1919,⁴⁶ 1,115,580 pounds were melted and 1,088,397 pounds poured, or a loss of 2.40 per cent.

The Anaconda Copper Mining Co. states⁴⁶ that the life of the resistor troughs in the zinc furnace is about three to four months; that the most economical tonnage of the furnace is 75 tons per 24 hours, the power consumption 100 kw. h. per ton, and that the Baily furnace is not considered a success for melting zinc cathodes, its use having been discontinued in favor of a coal-fired reverberatory, which is cheaper and makes less dross. A net recovery of practically 100 per cent of the zinc charged was made, but it was desirable to avoid the necessity of having to recover the metal from the dross, which averaged 6.5 per cent. The General Electric type and the Ajax induction-type furnaces are also said by the Anaconda Copper Mining Co. to be unsatisfactory for melting zinc.

Later reports⁴⁷ give fuller reasons for the failure of the electric furnace on zinc cathodes.

The cathodes form dross in melting in any furnace, and the blanket of dross prevents efficient transfer of heat to the charge in the electric furnace. To get anywhere near the rated production of the furnace (170 to 200 tons per day) the rate of energy supply must be so rapid and the resistor temperature so high that there is volatilization of zinc over $3\frac{1}{2}$ per cent (about 6 to 7 tons per day). At a production

⁴⁵ Data from advertising literature, Electric Furnace Co.

⁴⁶ Anaconda Copper Mining Co., private communication, Dec. 16 and 22, 1919.

⁴⁷ Laist, F., Frick, F. F., Elton, J. O., and Caples, R. B., Electrolytic zinc plant of Anaconda Copper Mining Co.: Mining and Metallurgy, Bull. No. 1,028, 1920, p. 34.

rate of only 100 tons per day the resistors must run at so high a temperature that their life is too short. At 70 tons per day—that is, at a low rate of power input, so that heat conduction through the charge can prevent surface overheating and volatilization and so that the resistors do not run so hot—the furnace compares favorably, metallurgically, with coal-fired reverberatories, but the item of interest on investment at the low rate of production is so high as to make the furnace uneconomical.

A 50-kw. nose-tilting Baily furnace for melting silver has recently been sold to Johnson, Matthey & Co. (Ltd.), London.

MELTING CAST IRON AND STEEL IN THE BAILY FURNACE.

Experimental melting of cast iron has also been done in the Baily furnace.⁴⁸

It is also stated⁴⁹ that in a Baily furnace in which the usual hearth had been given an inner lining of magnesite 1,000 pounds of boiler-plate scrap was melted in seven and one-half hours, with the furnace running at about 90 kw., the heat being started with the furnace at full steel-melting temperature. It took about 75 kw. to hold the empty furnace at this temperature. It is argued from this test that high-speed steel can be handled in the Baily furnace.

The rate of melting is very slow, and if it be increased by raising the power input the life of the resistor trough would undoubtedly be very short. It is doubtful if the resistor trough would stand up long enough to make the furnace practical on tool steel even at the usual low rate of power input used on brass on account of the small leeway between the furnace temperature and the temperature at which the resistor trough fails.

SUMMARY ON BAILY FURNACE.

The standard 105-kw. Baily furnace is said by the makers to be capable of melting 2:1 yellow brass with 400 kw. h. per ton, 50 cents per ton furnace upkeep, and 1 per cent net metal loss. According to reports from users, it appears that with careful operation of the furnace this statement is correct, but that the figure of 400 kw. h. needs amplification. It can be reached by operating 16 hours per day, or lowered by operating 24 hours per day, but it would be difficult, if not impossible, to reach it in regular 10-hour operation.

Working conditions, as to coolness, cleanliness, and convenience are good, as are also the reliability and refractory life. Its electrical characteristics are satisfactory; it is one of the simplest and easiest of furnaces to operate; it gives a low metal loss, and most of its

⁴⁸ Anonymous, Melting cast-iron scrap: *Iron Age*, vol. 107, 1921, p. 1686.

⁴⁹ Anonymous, Melting steel in a nonferrous electric furnace: *Iron Age*, vol. 108, 1921, p. 472.

users consider that it shows a decided saving over the fuel-fired furnaces it has replaced. A dozen firms have each installed more than one Baily furnace.

The furnace is flexible in that it can handle alloys high or low in zinc and can change from one alloy to another. The furnace is not capable of a very high rate of production and is not so susceptible to close temperature control, nor does it mix the metal as well as some other types. These, with its rather poor performance as to power consumption on 10-hour operation, are its greatest disadvantages.

GENERAL ELECTRIC FURNACE.

A furnace similar to the Baily in that the bulk of the heat is reflected onto the hearth from the roof but differing in the way the heat is generated has been developed by the General Electric Co. This is described by Valentine⁵⁰ and by Collins.⁵¹ The principle of this furnace can be seen from figures 17, 18, 19, and Plate IX (page 125).

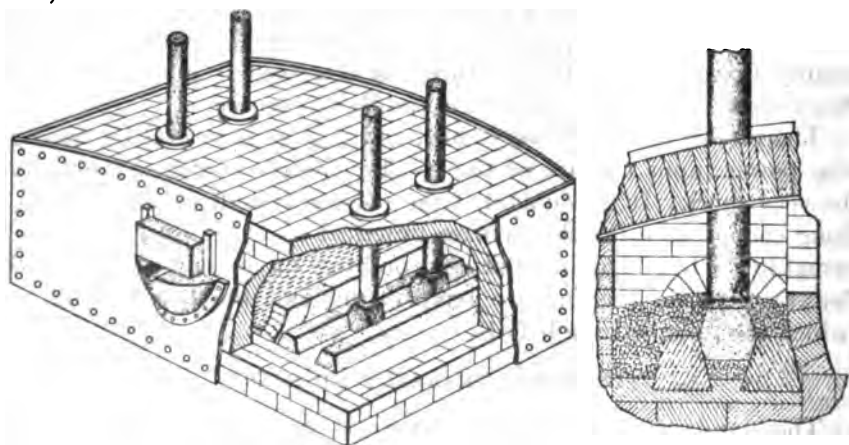


FIGURE 17.—Diagram of design of General Electric furnace, with detail of heating element.

A hearth is provided in the central part of a large furnace enclosure. Each side of the hearth, at each end of the furnace, is a bed of carbon slabs over which is piled granular resistor carbon. The granular material is held up by the carborundum end walls of the hearth. Two vertical electrodes (graphite, 5 or 6 inches in diameter) project through water-cooled rings in the roof at each end (four electrodes in all). The electrode holders are also water-cooled. The tips of

⁵⁰ Valentine, I. R., U. S. Patent 1,242,275, Oct. 9, 1917.

⁵¹ Collins, E. F., U. S. Patent 1,271,280, July 2, 1918; *Electric furnace for melting nonferrous metals and their alloys*: Jour. Cleveland Eng. Soc., vol. 11, 1919, p. 293; *Foundry*, vol. 47, 1919, pp. 284, 329; *Metal Ind.*, vol. 17, 1919, p. 221; *Chem. Met. Eng.*, vol. 21, 1919, p. 673; Anonymous, *Electric furnace replacing crucibles*: *Iron Age*, vol. 104, 1919, p. 913; see also St. John, H. M., *Discussion of above paper*: *Chem. and Met. Eng.*, vol. 22, 1920, p. 148.

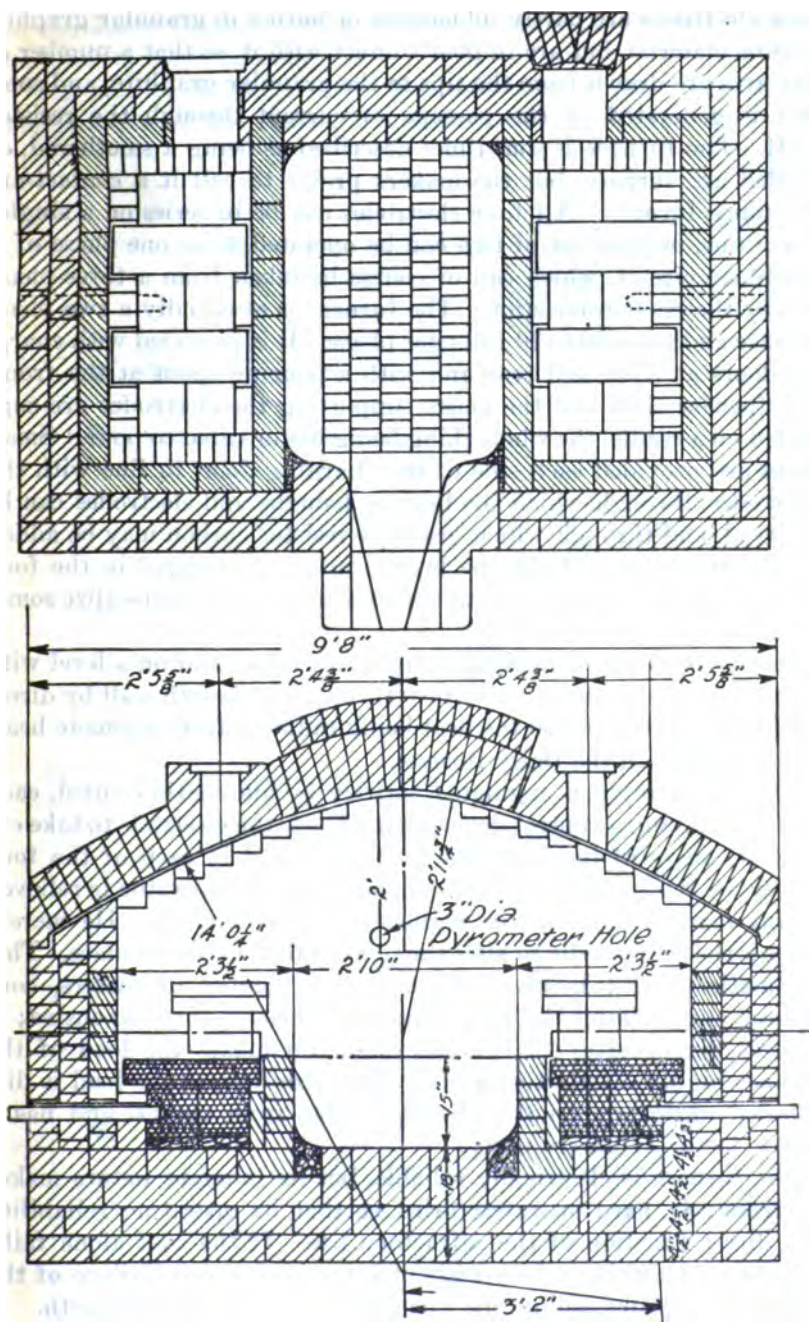


FIGURE 18.—Diagram of General Electric rectangular furnace.

these electrodes are partly submerged or buried in granular graphite resistor material, but are in poor contact with it, so that a number of tiny arcs are drawn from the tips to the granular graphite, and some heat is generated by the passage of current through the resistor itself. The furnace is sometimes described as being a smothered, or muffled arc furnace, but the makers prefer to call it a contact-arc resistance furnace. All four electrodes can be in series on a single-phase line, or each set of two can be operated from one phase of a two-phase circuit, which can of course be taken from a three-phase line by the Scott connection. The furnace is essentially a two-phase furnace and takes 60 to 70 volts per phase. It is provided with charging doors at front and rear and with a pouring spout at the front. The furnace shell and the masts supporting the electrodes are supported on a cradle, the whole thing being motor tilted to pour. Small doors are provided each side of the charging door, in line with the electrodes, through which the heating troughs and electrodes can be observed, and through which granular resistor carbon may be added as it burns away. While the heat is mainly generated in the four arcs, the conducting bed of granular resistor serves to delocalize somewhat the heat sources.

The location of the heating troughs in contact and on a level with the hearth allows some heat to flow through the hearth wall by direct conduction instead of all the heat being radiated from a remote heating element as in the Baily furnace.

The power input is regulated entirely by automatic control, each electrode having a control device that adjusts the electrode to take exactly its share of the load; the heat generated in each of the four corners of the furnace is thus kept equal and the operator is relieved of any manual labor in the regulation of the furnace. He merely turns a rheostat handle to alter the power taken by the furnace. This automatic control involves the use of a number of meters, contactors, etc., making the fully equipped furnace very expensive; it reduces the operator's labor, however, and makes the load of the furnace highly satisfactory to the central station, as the load is distributed uniformly on the phases, is free from surges, and has a power factor of nearly unity.

The furnace is electrically reliable, but its inherent nature makes it excessively hard on refractories, so that its operating reliability is a direct function of the refractory life. If the roof alone fails, a spare roof can be put on without a long delay, but failure of the hearth necessitates cooling the furnace and rebuilding the hearth.

The furnace in its usual size has a capacity of 1,500 to 2,500 pounds and is provided with a total transformer capacity of 400 kv. a., in the 1½-ton size, about 300 to 325 kw. being used normally. This type allows a very much higher rate of power input than in the case

of the Baily furnace, which is rated at only 105 kw. for a capacity of 1,000 to 1,500 pounds.

The source of heat is more localized than in the Baily, and the temperature of this source is higher. Though the arcs are to some

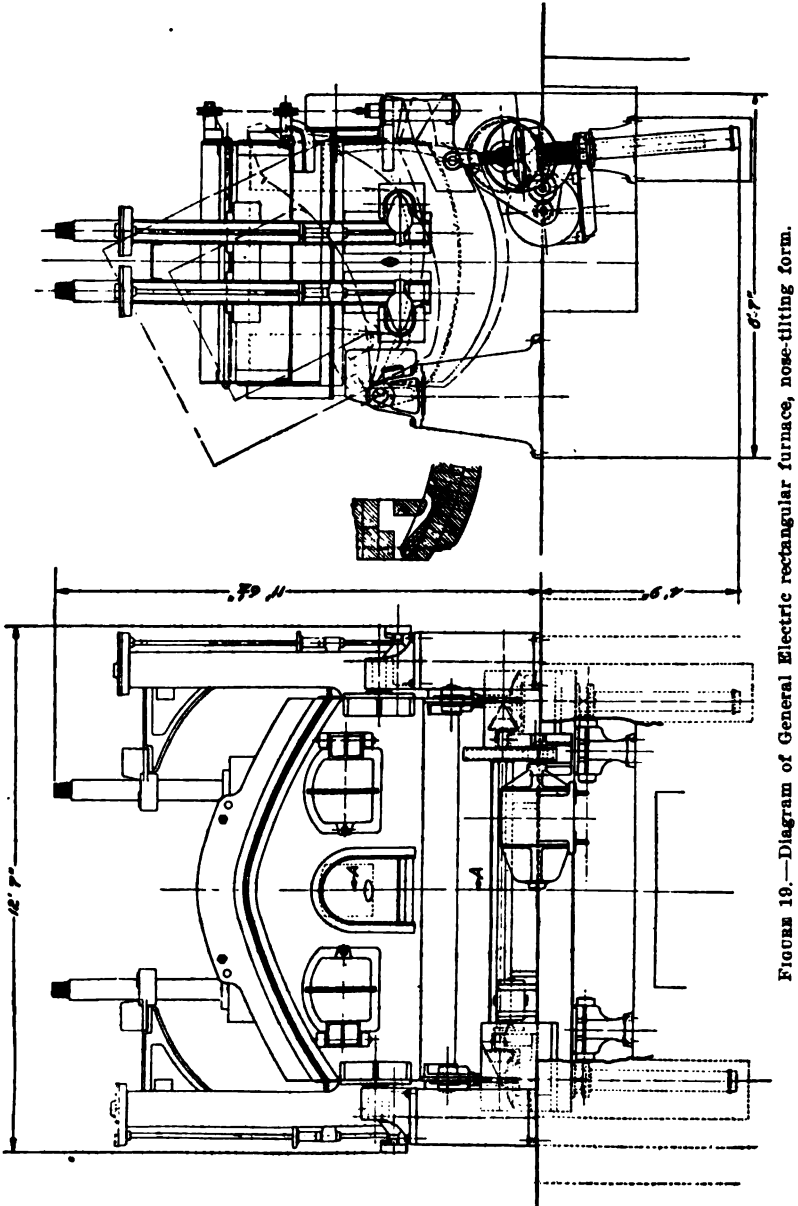


FIGURE 19.—Diagram of General Electric rectangular furnace, nose-tilting form.

degree smothered, the temperature around the tips of the electrodes is not far below that of the arc. The high temperature energy must

be reflected back mainly from the roof and end walls before it can reach the hearth. Almost no part of the heat goes to the bath by direct, unreflected radiation, as it does in the ordinary indirect-arc furnace.

In order to maintain the roof, it has been raised and its shape altered, from the original design shown in the patents to a form in which the heat is reflected more directly to the bath. The roof has no heat insulating lagging at the ends, directly over the heating troughs, and only a little in the center, as the roof fails if it is insulated.

Munroe fire brick, chromite, and Hybrand fire brick have all shown too short life in the roof. The chromite roof was reduced by the CO atmosphere.

A 1,000 to 1,500 pound furnace, taking an average of about 170 kw., was tried out in the General Electric plant at Schenectady in 1917 on 1,000-pound heats of a red brass of 80 to 85 per cent Cu (sheet punchings), 10 per cent Zn, balance Sn and Pb (new metals), 10-hour operation, no night heating. According to reports the furnace melted about 8 heats (4 tons) per 10-hour day, at a power consumption of about 350 kw. h. per ton under the most favorable conditions.

The average metal loss was said to be about 0.75 per cent. It was stated that only about 3 pounds of electrodes were used in 10 hours.

Collins⁵² tabulates the kw.-h. consumption necessary for melting in a 2,000-pound capacity 400-kw. General Electric furnace as follows:

TABLE 32.—*Claims made by General Electric Co. for performance of 1-ton furnace.*

Alloy.	Composition.	Kw. h. per ton.	Metal loss.	Pouring temperature.
			<i>Per cent.</i>	<i>Degrees.</i>
Red brass.....	80 Cu, 10 Zn, 6 Sn, 4 Pb, from scrap and new metal.	250 to 400	0.25 to 0.7	
Do.....	Yellow scrap and Cu or red brass and Zn, with gates and chips.	225 to 300	.15 to .50	1,300
Yellow brass.....	60 Cu, 39½ Zn, ½ Pb, from scrap and new metal.	220 to 350	.50 to 1.0	
Do.....	Scrap, gates, and chips.....	260 to 275	.40 to 1.0	1,100
Cupronickel.....	Scrap and billets.....	500 to 750		a 1,600
Nickel-silver.....	Scrap Cu, scrap Ni.....	500 to 750		a 1,650

* These pouring temperatures appear abnormally high to the writers.

He gives a chart "to show how foundry handicaps prevent the electric furnaces from making red brass on minimum power consumption," which assumes that the furnace itself is capable of melting at 275 kw. h. per ton plus 25 kw. h. per ton lost in transformer

⁵² Collins, E. F., Electric furnace for melting nonferrous metals: Foundry, vol. 47, 1919, p. 329; Jour. Cleveland Eng. Soc., vol. 11, 1919, p. 298.

and secondary leads; that is, 300 kw. h. is taken as the standard of what the furnace should do under perfect foundry conditions. The chart shows that the actual power used, with these handicaps, was about 350 kw. h. per ton.

No statement is made as to whether or not this chart includes pre-heating the furnace. Another chart shows that the furnace is pre-heated empty about three hours before charging, and the first heat was charged about half an hour before the foundry 10-hour working day begins, the production averaging $4\frac{1}{2}$ tons per day, or about 825 pounds per hour, under imperfect foundry conditions, although in the text the melting rate is once given as 1,500 pounds per hour, and again as 1,700 pounds per hour. These latter figures probably refer to 24-hour operation.

GENERAL ELECTRIC FURNACE AT PLANT OF CRANE CO.

A General Electric furnace was installed at the plant of the Crane Co., Chicago, for commercial test by the General Electric Co., a representative of the General Electric Co. being present whenever the furnace was run. Intermittent tests were made throughout 1918, continual running being impossible because of the repeated failure of the roof which necessitated replacement and redesign. The furnace was run on valve metal, ladle poured. Much trouble was found in making cored valve bodies to stand hydraulic test, the percentage of "leakers" being far above that from crucibles, according to the Crane Co., but according to the General Electric Co., when the metal was poured with a properly preheated ladle, defective castings from the electric and the crucible furnaces ran neck and neck. One source of trouble was the structural weakness of the hearth, the carborundum walls of which were under the pressure of the molten metal on one side and served to retain the hot resistor on the other. Metal seemed to leak through a crack in the hearth and to collect silicon that had been produced either by reduction of silicates in the ash from the resistor, or by the decomposition of carborundum at a hot spot in the resistor. At any rate, silicon was picked up, the product from the furnace being often ruined by silicon, although the same charge melted in crucibles did not pick up silicon. There was also some question whether this class of work could be satisfactorily made with ladle-poured metal, owing to the fact that there was more oxidation of the stream of metal in pouring from furnace to ladle and then from ladle to mold than in the single pour from a crucible.

At any rate, this furnace, as operated, did not impress the Crane Co. by quality of product. No difference was seen in metal losses

between this electric furnace and crucibles. The average power consumption per ton was excessively high during the periods of roof troubles, but came down under the most favorable conditions to about 400 kw. h. per ton for 10-hour operation.

The furnace was finally dismantled and returned to the makers in 1919. Detailed data on the experience with this furnace at the Crane Co. would be very instructive, as they might indicate whether the inapplicability of the furnace to the type of work was due to the furnace, to its method of operation, or whether no electric furnace would handle that work. By the terms of the contract under which the General Electric Co. had supplied this furnace for test, the Crane Co. was not able to give out detailed data.

The rejection of the furnace does not seem to be due to any prejudice of the Crane Co. against electric furnaces, as the firm is well satisfied with its Heroult steel furnace and is at present trying other types of electric brass furnaces.

Two other General Electric furnaces are installed at the McNab & Harlan Manufacturing Co. To an inquiry addressed to this company reply was made that the furnace had never been used for melting brass, but only for a higher melting, patented alloy, "Aterite," and that the company would object to furnishing any data or information, as the alloy is made under a formula which they do not disclose.

It is reported from other sources⁵³ that the alloy is a nickel alloy somewhat similar to an 18 to 30 per cent zinc-nickel brass (German silver), but low in nickel and carrying iron and lead.

Metal losses as low as 1 per cent and power figures as low as 400 kw. h. per ton on 16-hour operation have been claimed operating on alloys of this type poured at about 1,400° C. (2,550° F.) and perhaps even up to 1,600° C. (2,900° F.).

It is a pity that full details on the lining used and its life can not be obtained, for if the furnace shows reasonable reliability as to linings on such service it ought to be almost free from lining troubles on brass or copper.

GENERAL ELECTRIC FURNACE AT UNITED STATES COPPER PRODUCTS CORPORATION.

Another of these furnaces is installed at the plant of the United States Copper Products Corp. This is a 1½-ton furnace, run 24 hours a day at 300 to 325 kw., melting copper, which is

⁵³ Compare Collins, E. L., Melting nonferrous metals and their alloys in the electric furnace: Jour. Cleveland Eng. Soc., vol. 11, 1919, discussion, p. 316; Vickers, C., What is Aterite? Foundry, vol. 48, 1920, p. 70; Merica, P. D., Miscellaneous alloys of nickel: Chem. and Met. Eng., vol. 24, 1921, p. 652.

poured at 1,200° to 1,230° C. (2,200° to 2,250° F.). Twelve heats per day are obtained. The power is on about 1½ hours per heat, and it takes three-fourths of an hour to charge and pour. In average operation it is reported that it takes 550 kw. h. per ton of copper on 24-hour operation, though if more rapid charging and pouring were done this figure might perhaps be reduced to 300. There is no weighable loss in melting copper. On the basis of about 25 heats made with this furnace on 60:40 brass, it was thought that by hurrying the charging and pouring to cut the time the power is off to 15 minutes, eighteen 1-ton heats could be made in 24 hours.

If power were on 19 hours out of the 24, and the furnace run at 300-kw. average, the power consumption would then figure out to 310 kw. h. per ton. The metal loss on heavy yellow scrap is about 1 per cent. The representative of the General Electric Co. stationed at this plant stated the furnace did not work well on fine yellow scrap or turnings.

With a Hybrand brick roof the furnace, melting copper, gave from 125 to 180 heats. At 12 heats per day, this means putting on a new roof every 10 to 15 days. In the General Electric plant at Schenectady the longest roof life in melting brass so far reported was 350 heats. The use of corundite is expected by the makers to double this life.

Not only has the roof given much trouble on the furnace at the United States Copper Products Corp. but the carborundum hearth has failed, letting metal run into the resistor on several occasions. The design of later furnaces has been slightly altered, lowering the hearth and arranging it so that metal is less likely to work through into the resistor.

A spare roof must be kept ready to put on, just as is done with steel furnaces, because of the danger of the roof failing at any time. The furnace is purposely constructed both as to walls and roof with so little lagging to prevent the refractories getting too hot on the inside of the furnace that, in contrast to most electric brass furnaces, it is uncomfortably warm in the vicinity of the furnace, though not so hot as it would be near a one-ton fuel-fired furnace. The Washington Navy Yard, Washington, D. C., and the Ohio Brass Co., Mansfield, Ohio, have installed 1½-ton General Electric furnaces.

The smothered arcs of the General Electric furnace are very similar to a plan suggested by Rennerfelt.⁵⁴

Rennerfelt also suggests a design, evidently intended primarily for melting glass, of a furnace very similar in principle to the Gen-

⁵⁴ Rennerfelt, L., U. S. Patents 1,305,167, May 27, 1919, and 1,313,834, Aug. 19, 1919,

eral Electric furnace in which, as shown in figure 20, three smothered arcs, one for each phase of a three-phase circuit arc arranged about the furnace wall, the central part of the furnace containing either a melting hearth or crucibles. Rennerfelt emphasizes the severe treat-

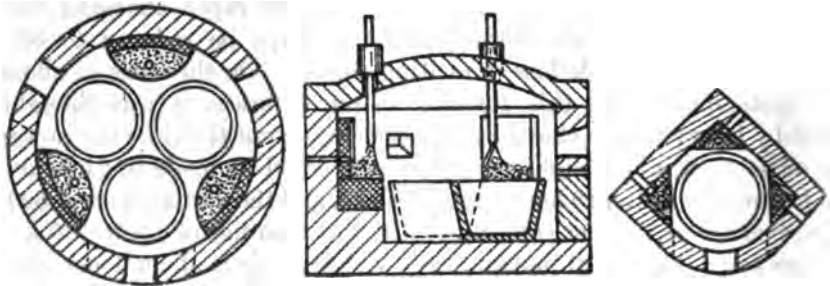


FIGURE 20.—Rennerfelt circular furnace design.

ment such an arrangement imposes on the refractories near the smothered arcs and reiterates the necessity of using extremely refractory materials at such points on this account. The General Electric Co. has recently developed a modification of its brass fur-

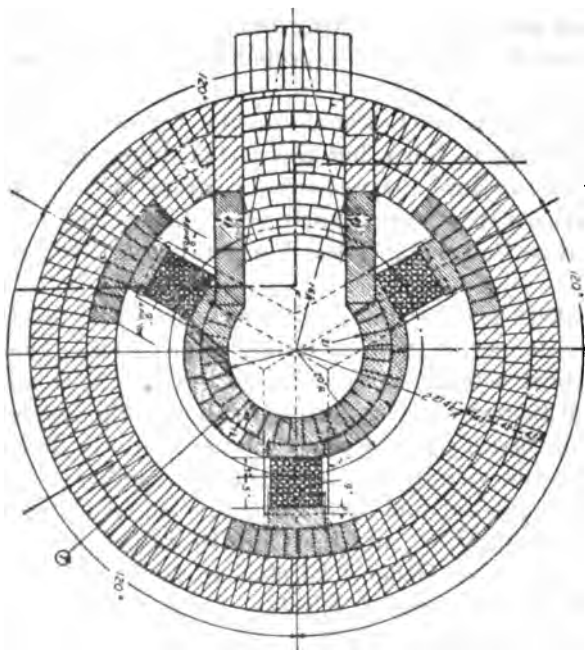


FIGURE 21.—Plan of General Electric circular furnace.

graphite over carbon wearing blocks, which are electrically connected.

nace, which, as is shown in figure 21, Plates X, B, and XI, is similar to the Rennerfelt scheme. The Rennerfelt plan uses open arcs to a bed of granular material, while the General Electric tries to avoid open arcs. In the new General Electric furnace, first tried in an 800-pound size and later developed in a 1,500-pound and larger sizes, there are three electrodes extending downward through the roof to three beds of granular



A. TWO NOSE-TILTING RENNERFELT FURNACES AT THE PHILADELPHIA MINT.



B. FRONT VIEW OF 1,500-POUND, 250-KW. HEARTH CAPACITY, GENERAL ELECTRIC NONFERROUS ELECTRIC FURNACE ARRANGED FOR NOSE TILTING; MELTING POSITION.



FURNACE SHOWN IN PLATE X, *B*, WITH ROOF LIFTED.

The furnace is circular in form, and the general arrangement of hearth and heat source is quite similar to that in the Baily furnace. Figures obtained in the General Electric Co.'s works⁵⁵ on the new type indicate that the 1,500-pound 250-kw. size will melt yellow brass at 250 to 300 kw. h. per ton, 24-hour operation, and on 9-hour operation at 325 to 375 kw. h. per ton. Metal loss on an alloy with 30 per cent zinc was 1.14 per cent on one set of heats. Electrode consumption is given as $1\frac{1}{2}$ to $2\frac{1}{2}$ pounds of graphite per ton, and about 1 pound of granular graphite per ton is required also. The life of the lining is stated to have been remarkably good on the experimental furnaces.

The prices of the General Electric furnaces, complete with transformers, meters, automatic electrode control, switchboard, etc., were, in April, 1920, as follows:

TABLE 33.—*Claims of makers of General Electric furnaces for melting brass, April, 1920.*

Capacity.	Kw.	Price.	Kw. h. per ton on brass heated to 1,100° C. 24-hour continuous operation.	Production, per hour on 24-hour operation.
Pounds.				Pounds.
3,000	400	\$17,000	250	2,250
2,000	250	16,000	275	1,500

The General Electric furnaces of the older type are said to have operated two years at Schenectady on the same linings, melting aluminum.

The same sizes as are used on brass are used for aluminum, but the transformer capacity supplied is lower.

SUMMARY ON GENERAL ELECTRIC FURNACE.

The General Electric furnace has very good electrical characteristics and can be operated at a low labor cost. It is capable of a good rate of production. The metal losses appear low, though few data on alloys high in zinc are available.

The design of the furnace is thermally inefficient, the transfer of heat from its source to the metal is very indirect, and there is a good deal of waste space in the furnace, yet this is somewhat compensated for by the high rate of power input.

The weak point of the furnace is the life of the refractories. The furnace has evidently been designed with the object of making a load desirable to the central station. That object has been fully

⁵⁵ See Winne, H. A., Recent developments in electric furnaces of the muffled-arc type: *Trans. Am. Electrochem. Soc.*, vol. 41. (In press.)

reached, but further commercial experience must be obtained with it before its relative value to the foundry itself, as compared with other commercial furnaces, is thoroughly understood.

Eastick⁵⁶ has criticized the design of the General Electric furnace.

RENNERFELT OR HAMILTON AND HANSELL REVERBERATORY FURNACE.

A more recent use of the smothered arc as a heat source is in the Rennerfelt reverberatory type.⁵⁷

This type, shown in figure 22, would usually be made tilting in smaller sizes, and tapping in the larger; vertical electrodes, usually

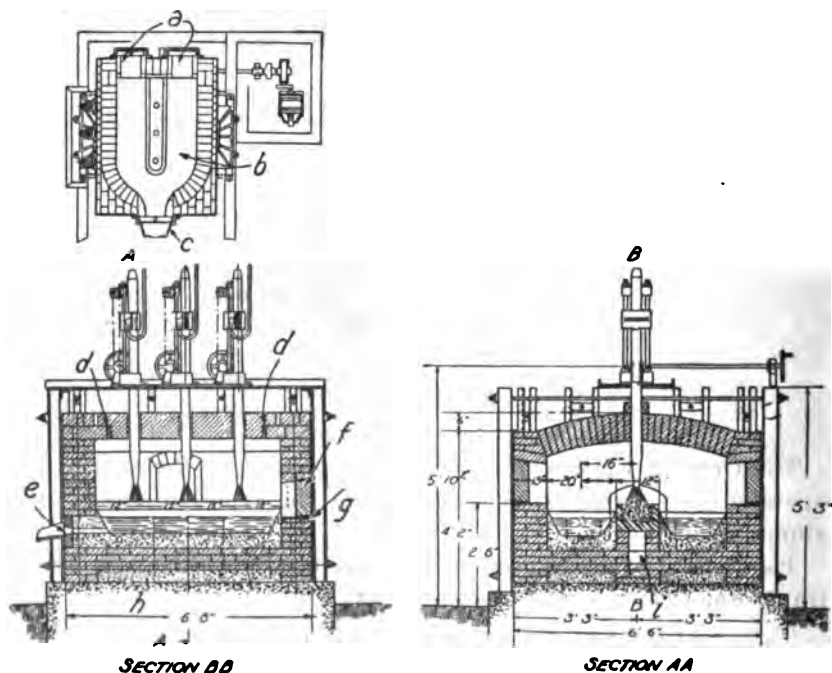


FIGURE 22.—Diagram of Rennerfelt reverberatory-type furnace: *a*, charging door; *b*, melting chamber; *c*, spout; *d*, wooden strips to allow for expansion; *e*, tap hole; *f*, skim door; *g*, skim plate; *h*, sand tightly packed; *i*, air duct.

three, draw arcs to a heating trough of carborundum, containing granular carbon or graphite. The voltage used is 100. The trough runs through the center of the furnace, with hearths for metal on each side. This design, using arcs, avoids the limitation on power input which makes the Baily inefficient as to power consumption. It should be more efficient than either of the General Electric types, as the heat is generated in the middle of the furnace and not near the end walls, so that it does not have to travel so devious a path before

⁵⁶ Eastick, T. H. A., Letter to editor, *Metal Industry*, vol. 19, 1921, p. 241.

⁵⁷ Rennerfelt, I., U. S. Patent 1,818,834, Aug. 19, 1919; DeFries, H. A., Rennerfelt electric reverberatory furnace: *Chem. and Met. Eng.*, vol. 22, 1920, p. 280.

it reaches the metal. As the electrodes enter at the center of the arch instead of at the ends, and the roof and side walls are both farther from the source of heat, the refractory life should be much increased over that in the General Electric furnaces.

The two hearths might be separate, and handle two different alloys at the same time, so that the furnace could operate on separate 1,000-pound charges with the efficiency of a 1-ton furnace. All the advantages of polyphase power and of automatic control shown by the General Electric furnace would be present in this.

In the opinion of the writers the location of the heating trough, embedded between the two hearths and below the metal level, is a source of weakness, as analogous construction in the General Electric furnace caused trouble from metal working its way into the trough. There also seems to be danger of leakage into the air duct provided below the trough to cool it. The reliability of the furnace might be largely increased if the air duct were replaced by a solid carborundum brick wall between the two hearths, thus making the construction of the whole hearth solid, and if carborundum bricks were placed at intervals across the carborundum wall above the metal level to support the trough, leaving spaces for ventilation and radiation from the bottom of the resistor. When the trough of the present design needs repair, both hearths will have to be partly torn out and rebuilt; the proposed design would allow replacing the trough, which is doubtless the weakest point, without interfering with the hearths.

Some alteration of this nature was, in fact, found necessary when the furnace was put to an actual test, since the makers report that in a trial of a 75-kw., 300-pound furnace at the Chicago Faucet Co., early in 1921, it was found desirable to raise the carborundum trough, which is now entirely exposed. On brass containing 60 per cent Cu and 40 per cent Zn the furnace, when fully hot (as in 24-hour operation) has melted at the rate of 330 kw. h. per ton, which is a good performance for so small a furnace. The metal loss is about 1 per cent on heavy scrap and 2 per cent on borings. The electrode consumption varied between 3 and 4 pounds per ton, and that of granular carbon in the resistor trough was about 2 pounds per ton. In one day's run 6 heats of 300 pounds each were made, the furnace being preheated for an hour in the morning. The first heat required one and one-half hours, later heats not over one hour each. The makers claim 360 to 480 kw. h. per ton on red brass, 8-hour operation. The net loss is said to be one-half of 1 per cent on red brass.

ARC FURNACES.

Direct-arc furnaces in which the arc plays directly on the charge have found but little use on copper alloys because of the intense local heat of the arc, which supplies heat more rapidly to the surface

of the charge than it can be carried through the body of the charge by conduction. This results in local overheating and in consequent loss of volatile constituents.

Over 30 years ago Slawianoff⁵⁵ described an apparatus for melting metal by the arc for such purposes as filling up blowholes in castings. He says: "Experiments show that brass, after electrical casting, becomes materially altered chemically in consequence of the burning out of the zinc."

HEROULT-TYPE FURNACE.

Hansen⁵⁶ made an experimental heat of 5,000 pounds of copper in a Heroult-type steel furnace, using a slag 1 to 1½ inches deep. Although the average temperature of the metal was not over 1,300° C., copper was either volatilized as a gas or tiny particles of metallic copper mechanically ejected, and the workmen were made ill. There were two cases of copper poisoning at the Titanium Alloy Manufacturing Co. when making titanium copper in a direct-arc furnace, two men being nearly killed before proper ventilation was secured. There is evidently a real danger in operating a direct-arc furnace on copper, at least under some conditions. In any experimental work on the direct-arc melting of copper good ventilation should be provided and the workmen supplied with suitable respirators or masks, at least until it is proven that no dangerous fume will be given off under the conditions of operation.

To avoid the volatilization of copper from a direct-arc furnace it would probably be necessary to hold the power input down to a rather low rate, so as to allow time for the heat to become more uniformly distributed through the melt. Perhaps some method of moving the furnace to stir the charge would work in this case, provided the stirring did not alter the distance from electrode to bath sufficiently to cause the arc to break.

There seems to have been no trouble due to copper fume, in the operation of direct-arc furnaces on Monel metal or bronze.

A three-phase, 1,500-pound furnace of this type was used in 1915 and 1916, but probably not thereafter⁵⁷ at the plant of the Canadian Brakeshoe Co. Red brass was melted under a slag of molten glass. The metal loss was stated to be under 2 per cent. The furnace was not used on material of high zinc content.

⁵⁵ Slawianoff, N., English Patent 16,279, of Oct. 12, 1890; U. S. Patent 577,329, Feb. 14, 1907; application filed Jan. 29, 1891.

⁵⁶ Hansen, C. A., Copper poisoning; *Met. and Chem. Eng.*, vol. 9, 1911, p. 67; *Electric melting of copper and brass*; *Trans. Am. Inst. Metals*, vol. 6, 1912, p. 110.

⁵⁷ Davy, J. E., Jr., private communications.

The Driver-Harris Co., Harrison, N. J.,⁵¹ operate two basic lined Heroult direct-arc furnaces, rated at 600 kw. and 2 tons capacity, but up to 3 tons are now charged. Three heats are made each day of 12 to 14 hours per furnace. Among the various alloys made are a 30 per cent nickel steel and one with 85 per cent nickel and 15 per cent chromium. The furnaces are therefore to be included both among those melting ferrous and those melting nonferrous alloys.

The power consumption averages 650 to 750 kw. h. per ton, and the consumption of carbon electrodes is 14 pounds per ton. The roofs last only about 25 heats. The gross metal loss is given as about 2½ per cent.

Another Heroult furnace, at the plant of the Hiram Walker & Sons Metal Products Co., Ltd., in Walkerville, Ontario, is melting nickel-iron-chromium alloys.

Direct-arc furnaces of the Snyder, Heroult, and Greaves-Etchells types are in use⁵² for melting such alloys as nickel-chromium, Monel metal, and cupronickel.

SNYDER TYPE FURNACES.

Haynes⁵³ stated in 1917 that though three electric furnaces of the Snyder type had been used in melting Stellite, a cobalt-chromium alloy, poor results had been traced back to the electric furnace, and the electric furnace had been discarded in favor of melting in covered crucibles. Mr. Haynes was careful to say that he did not mean to assert it would be impossible to use the electric furnace successfully, and that investigations were planned to try it out further.

In February, 1920, Mr. Haynes⁵⁴ said that the troubles had been overcome, and the furnaces were now giving highly satisfactory service. The power consumption, under normal conditions, runs from 800 to 1,200 kw. h. per ton. In work on Stellite the Haynes Stellite Co. has been able to get as much as 3,000 heats from a lining, an astonishing figure.

A 60-kw., 200 to 300 pound Snyder furnace has also been in use on a cobalt-chromium alloy at the Chrobaltic Tool Co.,⁵⁵ Chicago, Ill., which is said to get out about one heat per hour when the fur-

⁵¹ Compare Easton, W. H., Electric furnace for melting alloys: Elec. World, vol. 72, 1918, p. 295; see also Major, R. M., Electrical production of alloys: Trans. Am. Elec. rochem. Soc., vol. 37, 1920, p. 441.

⁵² Compare editorial, Met. Ind. (London), vol. 14, 1919, p. 111; Etchells, H., Met. Ind. (London), vol. 14, 1919, p. 113; Easton, W. H., Electric furnaces for melting alloys: Elec. World, vol. 72, 1918, p. 295; Diller, H. E., Foundry standardizes nickel alloy: Foundry, vol. 48, 1920, p. 209; Arnott, J., Monel metal: Metal Ind. (London), vol. 16, 1920, p. 327; Brooks, G. D., Discussion, Metal Ind. (London), vol. 16, 1920, p. 346.

⁵³ Haynes, E., Discussion, Jour. Am. Inst. Metals, vol. 11, 1917, p. 433.

⁵⁴ Haynes, E., private communications; see also Haynes, E., Stellite: Trans. Am. Elec. trochem. Soc., vol. 37, 1920, p. 880.

⁵⁵ See anonymous, Chrobaltic cast milling cutters: Machinery, vol. 26, 1919, p. 354.

nace is fully hot; this figure corresponds to about 600 kw. h. per ton on continuous operation. The roofs of these small furnaces are said by the makers to stand up to 400 heats.

It has been stated⁶⁶ that the Green direct-arc furnace would melt brass, but no data are given.

In 1912 the Metallurgic Engineering Co. advertised the Snyder furnace for melting brass, stating: "If you are making copper castings for conductivity purposes, the conductivity of electric furnace castings is much higher; if you are making brass, the loss in zinc is much lower. For steam and hydraulic fitting, the electrically melted copper and bronzes are more dense and will not seep under high pressure and superheat." On attempting to get full data from the advertisers it was stated that a 1½-ton, 300-kw. steel furnace was used, a charge of 1,200 pounds copper, 600 pounds zinc, 60 pounds lead, and 200 pounds brass scrap, or 2,060 pounds total, being melted in one hour, using 198 kw. h., the furnace being at steel-melting temperature at the start, and hence having a good deal of stored heat. A thin glass slag was used. No figures could be obtained on metal losses, the data as to the weight of metal poured being refused, but the statement was made that the loss was between ½ and 1 per cent. A representative of the Commonwealth Edison Co. who had witnessed the test said that dense clouds of zinc fume were evolved and the loss appeared to be high, but that owing to the presence of some molten steel in the furnace at the start the brass poured was contaminated with steel. The test was worthless for determining metal losses or power consumption, though it was apparently the basis for the advertisement. The firm later stated that the advertisement was premature and a mistake.

GREAVES-ETCHELLS FURNACE.

A Greaves-Etchells direct-arc furnace, 1,000 pounds, 300 kw., is in use at the Hoskins Manufacturing Co., Detroit, for nickel-chromium alloys.

Another Greaves-Etchells furnace has been sold to the Japanese mint for melting coinage bronze and silver, and still another has been sold to Lacheze et Fils, Dijon, France, for nonferrous alloys.

GRONWALL-DIXON FURNACE.

A Gronwall-Dixon furnace of 500-pounds capacity at the Cleveland Brass Manufacturing Co., Cleveland, Ohio, is melting "Clebrium," an alloy that has among its constituents chromium and, it is said, about 40 per cent of iron. The furnace might, therefore, be classed either among nonferrous melting furnaces or among furnaces for special steels, according to the classification of the alloy.

⁶⁶Anonymous, Green arc furnace: Iron Age, vol. 103, 1919, p. 1005.

SNYDER FURNACE OPERATING ON MONEL METAL.

A 1,000-pound, 150-kw. Snyder furnace is running on Monel metal at the Monel Metal Products Corporation, Bayonne, N. J.; its output approximates 5,000 pounds in 10 hours, and its power consumption about 600 to 850 kw. h. per ton. A slag about $\frac{1}{2}$ inch thick is used. The total loss of metal in the foundry is given as about 2 to 2½ per cent, but exact figures for the melting loss are not available. The furnace roof, of high-grade fire brick, lasts 50 heats on the average, and has lasted for 90 heats. The hearth is made of ganister and does not have much longer life than the roof.

The furnace is said to be easier to operate and to give metal of better quality than that from an oil-fired reverberatory with pantype burner. Great care is taken to operate with the furnace tightly closed to avoid oxidation. It is stated⁶⁷ that a Moore direct-arc furnace, made by the Pittsburgh Electric Furnace Co., has been ordered for the Huntington, W. Va., plant of the International Metal Co., to melt Monel metal.

LABORATORY TEST OF DIRECT-ARC FURNACE MELTING YELLOW BRASS.

In April, 1915, the Bureau of Mines built a direct-arc furnace in order to determine the metal loss in melting yellow brass. Instead of a tilting-type furnace, one was built in which an ordinary No. 30-graphite fire-clay crucible, with a double handful of chip graphite in the bottom to protect the crucible from the arc when starting, was set on an electrode built into the furnace bottom. The upper, movable electrode, from which the arc was struck to the charge, was a 2-inch diameter graphite rod. The crucible was set into a fire-brick inclosure, only enough space being allowed between crucible and walls to admit the tongs for lifting the crucible. A tight cover (fire-brick slab) with a 2-inch diameter hole was put over the furnace after the crucible was in place, the upper electrode passing through the hole. In order to avoid loss by spattering, the crucible was not filled to its full capacity of 90 pounds, only 50-pound charges being used. The furnace ran at about 50 volts, 400 to 500 amperes.

Two alloys were melted, both in ingot form, no new metal being used, one of 80 per cent copper, 15 per cent zinc, 2½ per cent tin, 2½ per cent lead, the other of 67 per cent copper, 33 per cent zinc. A couple of pounds of green bottle glass was used as slag on the former, and about 3 pounds of salt on the latter. The results are tabulated below.

⁶⁷ Editorial, *The Foundry*, vol. 49, 1921, p. 583.

TABLE 34.—*Bureau of Mines test of small direct-arc furnace melting alloys high in zinc.*

Heat No.	Time.	Condition of furnace at start.	Zinc in alloy melted.	Weight of metal charged.	Weight of ingot poured.	Weight recovered from spillings and from slag.	Total weight recovered.	Loss.	Pouring temperature.	Kw. h. used.
	<i>H. m.</i>		<i>Per cent.</i>	<i>Pounds.</i>	<i>Pounds.</i>	<i>Pounds.</i>	<i>Pounds.</i>	<i>Lbs. P. ct.</i>	<i>° C.</i>	
1.....	1 10	Cold.....	15	50.00	45.80	0.30	45.10	3.90 7.8	1,150	24½
2.....	1 0	Slightly warm.	33	51.25	45.65	1.20	43.85	4.40 8.6	1,070	19
3.....	1 5	Cold.....	33	52.35	47.25	1.05	48.30	4.05 7.7	1,050	20½
4.....	1 0	Hot.....	15	52.35	50.60	.30	50.90	2.45 4.6	1,140	19

Neither alloy was heated as hot as it would normally be heated in commercial practice. As soon as the alloy had started to melt down great clouds of zinc vapor came out about the clearance around the electrode, this being the only opening of the furnace that was not luted tight while running. The furnace was set up outdoors. Had it been inside it would have been impossible to have completed the runs on account of the fume. As losses of 4½ per cent on a 15 per cent zinc alloy and 7½ per cent on a 33 per cent zinc alloy, melting ingots, are decidedly higher than would be obtained in fuel-fired furnaces, it was evident that the direct-arc furnace was not fitted for use on alloys high in zinc.

TESTS OF SMALL SNYDER FURNACE.

In January and February, 1916, a 600-pound tilting Snyder⁶⁸ furnace was tested at the plant of the Chicago Bearing Metal Co., through whose courtesy a representative of the Bureau of Mines was present during part of the test.

The furnace was single-phase, rated at 100 kw. It had a rammed-in magnesite hearth, bonded with linseed oil, outside of which and next to the shell was a layer of rammed-in No. 26 J. M. asbestos cement. The roof was of silica brick. The lower electrode was a 1-inch diameter copper rod, water cooled outside the furnace and extending up through the refractory bottom to the hearth. The upper electrode was of 3-inch diameter graphite, coming down vertically through the dome of the roof. The roof, electrode and all, was raised and at the same time tilted back, by a handwheel, so that the furnace could be charged from the top. A sand seal between roof and furnace body made a tight joint. A pouring spout, closed with a fire-brick plug and fire-clay luting, was provided, through which, when the whole furnace, including the electrode support, was tilted by a second handwheel, the metal could be poured. Some 20 or 25

⁶⁸ See Snyder, F. T., U. S. Patents 1,100,995, June 23, 1914; 1,167,028, Jan. 4, 1916; 1,325,539, Dec. 23, 1919; and 1,327,174, Jan. 6, 1920.

pounds of metal was usually retained in the bottom of the furnace between heats. The power supply was 60-cycle, 4,400 volts, which was stepped down to 440 volts and then by a second transformer containing a good deal of reactance to a lower voltage, probably around 110 to 125. A portable ammeter, voltmeter, wattmeter, and watt-hour meter were used on the 440-volt side of the second transformer for test purposes, the only meter provided with the furnace being an inaccurate wattmeter. The readings for the test were made by the testing department of the Commonwealth Edison Co.

The alloy melted, except in a few runs on a yellow brass made at the bureau's request, contained 70 to 75 per cent copper, 15 to 20 per cent lead, and about 6 per cent tin, the balance being impurities (zinc, lead, iron, etc.). Charges were made up of about half cupola ingot or heavy scrap railroad bearings and about half heavy borings free from oil or gates. The charge was quite free from nonmetallic material, not over 0.50 per cent being present at most. Metal losses were obtained by weighing the ladles before and after pouring. Ladle skimmings and metal spilled in pouring were not weighed but charged back with the next heat. Slag scraped from the furnace was kept separate and its recoverable nonmetallic content calculated.

The power input to the furnace averaged about 70 kw. and was never over 90 kw.

RUNS ON YELLOW INGOT.

Three heats were made on a yellow brass ingot of 67.8 per cent copper, 27.5 per cent zinc, 2.9 per cent lead, 1.5 per cent tin, 0.3 per cent antimony.

Heat A.—The furnace was hot from previous runs on the regular leaded bronze, but had cooled somewhat because of trouble in making contact with the lower electrode. The furnace had been tilted clear up and scraped to get out all the leaded bronze and slag that had run over the lower electrode and frozen. Five hundred and one pounds of yellow ingot, plus 10 pounds leaded bronze borings to get contact between electrode and charge, was charged, plus limestone and a little sand to form a slag. The furnace was very tightly closed by luting with fire clay about the spout plug and electrode, and very little fume escaped during the run. Current was on 1 hour and 50 minutes, but only 96 kw. was used, the power input being only an average of 52 kw.

The furnace would not take any more power than this, the arc being very short and snappy, no matter how close the graphite electrode was brought to the bath, as the power factor was amazingly low, running from 18 to 38 and averaging 33. When the metal was poured its temperature measured in the ladle was only

980° C., far too cold for pouring castings, though the metal poured into ingot without much skulling. The roof, furnace walls, spout, and back of the spout plug were covered to a depth of one-half inch with a mixture of zinc oxide and metallic zinc which had distilled out of the charge by the local overheating of the arc. When this coating was scraped in order to bare a fresh surface, it burned with the characteristic zinc flame, showing the presence of metallic zinc. Only 428½ pounds was obtained in the ladle as spillings and as large chunks in the slag. The skimmings weighed 47 pounds and were estimated to contain 75 per cent metallic or 35 pounds. The furnace was not drained completely, but certainly not over 20 pounds was left in it.

As a total of 511½ pounds was charged and 428½+35+20 pounds obtained, the net loss was about 28 pounds, or 5½ per cent.

Heats B and C.—To check up this figure two more heats were made, the furnace being hot from a heat on bearing bronze, which had been drained and scraped out till the furnace was clean.

In heat B the arc was on two and one-half hours, 108 kw. h. being used. The average power input was 43 kw., but the power factor was not taken. The metal was poured at 1,090° C. Cupola slag, glass, and limestone were used as slag. In heat C the arc was on 1 hour and 48 minutes, 84 kw. h. being used. The average power input was 52, the power factor varying from 20 to 44 and averaging 33. The metal was poured at 1,070° C. Little zinc fume escaped during the run, but vast clouds came out while pouring. In these two heats 1,017 pounds ingot was charged, and 925½ pounds were weighed up in the ladles. The furnace was scraped clean after heat C. From the skimmings 4 pounds of large chunks were picked. The balance of the skimmings, 53 pounds, contained 47 pounds metallic. Recovery, $925\frac{1}{2}+4+47=976\frac{1}{2}$ pounds; loss, 40½ pounds, or 4.4 per cent. Analyses of metal from heats B and C showed, respectively, 71.8 per cent copper, 22.3 per cent zinc, and 69.7 per cent copper, 25.3 per cent zinc; the original ingot carried 67.6 per cent copper, 27.5 per cent zinc. Compared with this 27.5 per cent, the average zinc content, 23.8 per cent of the product, indicates a loss of about 4½ per cent.

These runs, while not giving very accurate figures, show plainly that the direct-arc furnace, causing losses of 4 to 5 per cent on yellow brass charged as ingot, can no more be considered for melting alloys high in zinc than can an open-flame oil furnace.

Smith⁸⁸ mentions a trial made in England during the war in which it was found impractical to handle yellow brass in a direct-arc furnace.

⁸⁸ Smith, K., discussion, *Metal Ind.* (London), vol. 16, 1920, p. 368.

In these three heats, yellow brass was heated to an average temperature of less than 1,050° C., in a hot furnace, at an average rate of 380 kw. h. per ton.

RUNS ON LEADED BEARING BRONZE.

Though the furnace was plainly useless for yellow brass, it worked better on the leaded bearing bronze, as is shown by the data below on two-day runs.

In order to keep the furnace hot it was filled with coke at night, and enough air was allowed to enter to keep the coke smoldering. The coke was scraped out the next morning. On each heat in the test given in Table 30, 600-pound charges were used. The charge was half ingot or heavy scrap, half heavy borings or gates, save in heats 2, 3, and 4 of February 10, when the charges were concentrates from ashes and skimmings from the foundry's reclaiming plant.

TABLE 35.—*Test of 600-pound Snyder furnace melting leaded bronze.*

Date.	Heat No.s	Power on.	Power off.	Arc on. ^b	Kw. h. used.	Temperature in ladle.
				<i>H. m.</i>		<i>° C.</i>
Feb. 9.....	1	7.00 a. m..	9.06 a. m..	1 50	132	
2.....	2	9.06 a. m..	11.01 a. m..	1 47	132	1,130
3.....	3	11.01 a. m..	1.00 p. m..	1 30	108	1,170
4.....	4	1.00 p. m..	2.55 p. m..	1 40	108	1,140
10.....	5	2.55 p. m..	4.50 p. m..	1 48	108	1,250
	1	7.04 p. m..	9.19 p. m..	2 00	132	
	2	9.19 p. m..	11.12 p. m..	1 40	108	
	3	11.12 p. m..	1.07 a. m..	1 48	108	
	4	1.07 a. m..	2.45 a. m..	1 23	108	
	5	2.45 a. m..	4.35 a. m..	1 35	108	
					c 1,150	

^a Total for 10 heats, 6,000 pounds.

^b Average time on, 1 hour and 46 minutes.

^c Average, 368 kw. h. per ton.

In the test in January, when an operator was on the job who maintained a higher power input, and hence only averaged 1 hour and 36 minutes for the time the arc was on per heat, and the metal was poured with little delay, so that six heats were made per 10-hour day; the average of several days was 350 kw. h. per ton.

In 27 heats, with the regular bearing-metal charge, 15,468 pounds were charged and 15,138 pounds metal were weighed in the ladles, a gross loss of 2.13 per cent; 431 pounds slag contained 7 per cent copper and was calculated to contain 5 per cent recoverable metal, or 21½ pounds, making the net loss 1.99 per cent.

In 17 later heats of the regular bearing-metal charge (heats on concentrates not being considered) on which data were taken by the bureau's representative, 9,978 pounds were charged and 9,767 pounds were weighed in the ladle; and 281 pounds slag were obtained, estimated to contain 14 pounds recoverable metallic, a gross loss of 2.12 per cent, or a net loss of 2 per cent.

From this 2 per cent net loss should be deducted about $\frac{1}{2}$ per cent nonmetallic in the charge, leaving the true net loss 1.5 per cent. The net losses, not deducting for nonmetallic in the charge, on the same material melted in the other furnaces at this plant, were said to be:

	Per cent.
Oil-fired crucible lift-out furnace.....	2 $\frac{1}{2}$
Tilting crucible, forced draft, coke.....	2 $\frac{1}{2}$
Open-flame oil.....	5 $\frac{1}{2}$

Out of the 5 $\frac{1}{2}$ per cent in the open-flame oil furnace 3 per cent represents extra lead added in the ladle to compensate for the lead burnt out or volatilized in the open-flame furnace to make the product analyze the same as that from the crucible furnaces.

On account of slag formation in the open-flame furnace and of much charge being dropped into the ashes in the coke furnace, from which it had to be recovered in the reclaiming plant, the distribution of the metal charged to the different furnaces was as follows, according to the plant records:

TABLE 36.—*Distribution of metal charged.*

Furnace.	In crucible or ladle.	For recovery in slag or ashes.	Lost.
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
Crucible lift-out oil.....	95	2.5	2.5
Crucible tilting coke.....	85	12.5	2.5
Open-flame oil.....	90	4.5	5.5
600-pound Snyder.....	97.9	.1	2.0

In the crucible tilting coke furnace much of the heavy gross loss was due to spilling the charge outside the crucible into the coke in charging.

The 600-pound furnace ran smoothly, was kept tight by luting the spout and the space around the electrode so that the furnace did not give off much fume, was much cooler, and was less irksome generally to operate than fuel-fired furnaces. The power factor of the furnace was uniformly low, and varied greatly with the conditions of operation, particularly with the nature of the slag.

The slag used in the yellow-brass heats was not fluid, but crusty, and was raked off rather than skimmed. In the regular bearing-metal heats, in which limestone in walnut or large size was added to form a slag with the small amount of sand on the gates and with the sand and dirt swept up with the spillings, the slag was also more crusty than fluid. The addition of limestone gave a good fluid slag in a few heats on melting concentrates, from the reclaiming plant, that carried slag from the open-flame oil furnaces and ash from the coke furnaces. The variation in power factor is shown below:

TABLE 37.—*Variation of power factor in melting different materials in 600-pound Snyder furnace.*

Material.	Average kw.	Average kv. a.	Average power factor.	Lowest power factor.	Highest power factor.	Slag.
Yellow brass.....	45	136	33	18	44	Crusty
Bearing metal.....	70	126	55	27	48	Do
Concentrates.....	69	96	72	55	89	Fluid.

In an atmosphere of zinc or lead vapor and with a crusty slag, which probably causes the arc to play directly on metal rather than on slag, the power factor obviously is amazingly low, whereas in melting concentrates, which are practically free from zinc and low in lead, with a fluid slag that can produce arc-supporting vapor, as in a direct-arc furnace operating on steel, the power factor, though low, could be tolerated.

The plant decided that a 600-pound furnace was too small a unit. The test indicated, however, that the installation of 1-ton furnaces, especially if they were run 20 to 24 hours a day, would effect notable savings through the elimination of crucibles and the reduction of metal losses; the crucible furnaces, therefore, were replaced by two 1-ton Snyder furnaces, and two 1-ton Rennerfelt furnaces were added later. The open-flame oil furnaces continue to be used when the demand for metal exceeds the capacity of the electric, or when an electric is down for relining; but the great bulk of the production comes from the electric furnaces.

The Chicago Bearing Metal Co. makes large railroad-car bearings, used as backing for babbit liners; on these bearings considerable variation in composition is allowable and they have to pass no severe tests. The question of improved quality in their manufacture was not paramount, as the open-flame oil furnaces produce metal of satisfactory quality for the purpose. Under these conditions the usefulness of the furnace lies almost entirely in quicker production, the principal and almost the sole aim in the operation of furnaces being large output.

Two single-phase 1-ton furnaces with 700-kv. a. transformers, built on the same general design as the 600-pound furnace, were installed, but it was found that on some points their action was not quite like that of the smaller furnace. Figure 23 shows the design of the single-phase furnace.

It proved impossible to lute the spout and electrode tightly enough to prevent the escape of metal vapors or to prevent the entrance of air. The pressure developed inside the furnace was so great that the luting was blown out; if the luting were held in place mechanically the bricks in the roof were loosened until sufficient vent was made for the vapor to escape.

The furnaces have regularly been run with the spout plugged loosely, if at all, and with plenty of clearance about the electrodes. As a result the metal losses have been greater than was indicated

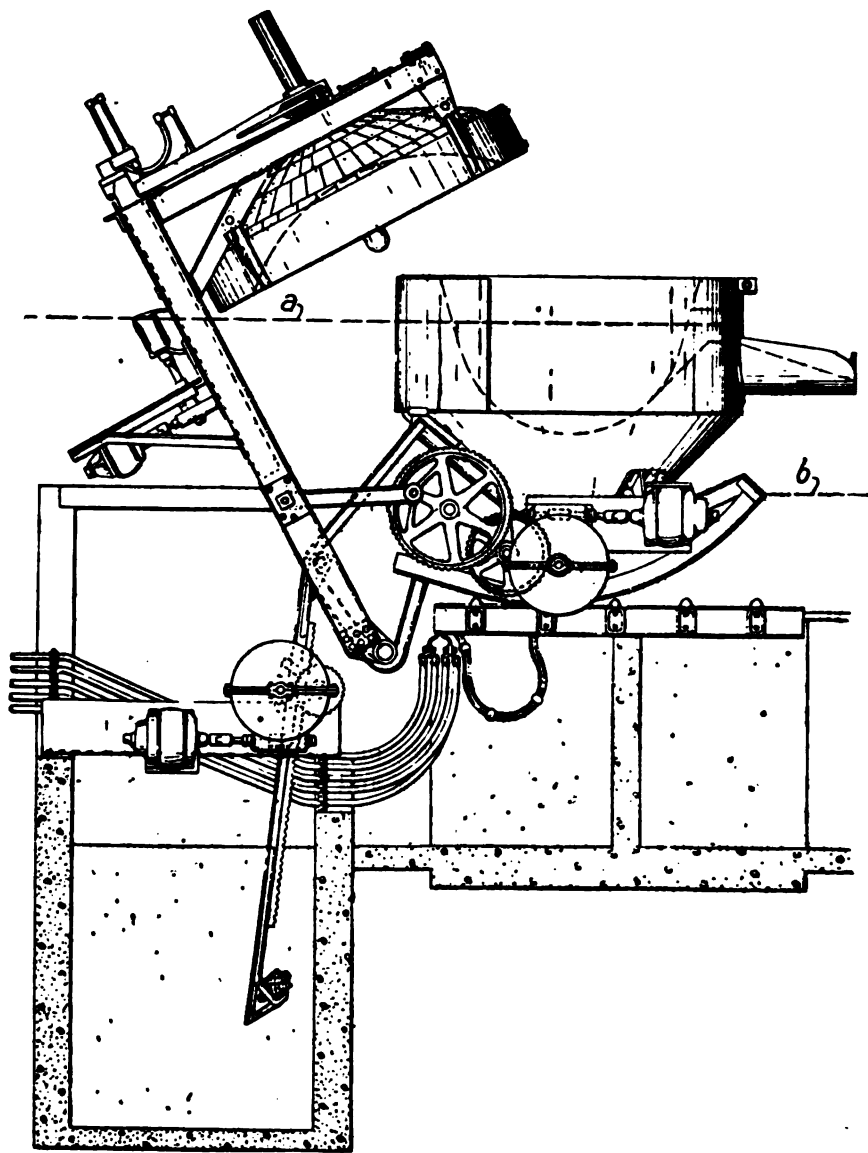


FIGURE 28.—Snyder direct-arc furnace: *a*, charging-floor line; *b*, foundry floor.

by the test of the small furnace and the working conditions as to fume and smoke have been much worse than with the open-flame oil furnaces. It has been necessary to install hoods and fans to withdraw the fumes in order to avoid danger from lead poisoning;

in the first few weeks' operation of the large furnaces several workmen were seriously, though temporarily, affected by the lead and antimony fume.

IMPROVEMENTS IN SNYDER FURNACE.

Inasmuch as power is purchased on a schedule calling for a penalty when the power factor of the total supply to the plant is under 70, and as the average power factor of the Snyder furnaces has been much below that figure, constant effort has been made to improve the power factor. An attempt at improvement was made soon after the installation of the furnaces by removing from one furnace most of the reactance that had been put into the transformer to stabilize the arc. In consequence the furnace drew such an excess of power that an almost new silica-brick roof was melted down promptly. Part but not all of the reactance was then replaced. This reduction in reactance raised the power input that could be conveniently held to about 420 kw. instead of 360 kw., which was all that could be conveniently held when the original reactance was in use. Most of the later attempts at improving the power factor dealt with the furnace itself and its method of operation rather than with the reactance of the circuit. The furnaces use an open-circuit voltage of about 205, which drops under load to an arc voltage that varies from 100 to 160 volts, depending on the position of the electrode and the resulting length of the arc, but averages about 130.

Until recently the furnaces were hand operated. An arc long enough to draw the higher voltage can not be readily maintained while the cold charge is melting down, especially if the charge is mostly made up of large chunks such as ingot or of scrap bearings. If a considerable percentage of borings, concentrates, or other small pieces is present, a pool of molten metal is more readily formed and the maintenance of an arc at the higher voltage is possible.

The operator can hold the arc at 145 volts if he gives it constant attention, but owing to the inherent reactance in the circuit and to that in the furnace transformer if he sets the arc at a length where it will burn steadily with little or no hand regulation this length draws 125 to 130 volts. St. John¹⁰ gives a curve showing that at 110 volts the power factor is 47, at 125 it is 55; if the arc voltage is held up to 160, the power factor rises to 68.

In the regular running of the hand-operated furnace the voltage averages 125 to 130 and the power factor 55 to 56. This power factor is so low that it involves an expensive penalty in the power bill. The motor load, and especially the use of the Rennerfelt furnaces, which have a power factor well above 70, increase the total plant

¹⁰ St. John, H. M., Commercial testing of metallurgical electric furnaces: Chem. and Met. Eng., vol. 21, 1919, p. 383.

power factor so that the penalty is much less than it would be if the electric furnaces were all Snyders.

By the use of plenty of coke on the surface of the melt, so that the arc was drawn between the graphite electrode and the coke, instead of between the electrode and the metal, the amount of metal vapor in the arc was reduced, the arc length increased, and the voltage raised so that a power factor of 73 could be obtained. The practice was not adopted because of the contamination of the metal by the sulphur in the coke.

The use of crushed electrode scrap, which would be free from sulphur, has been suggested, and obviously is worth trying, but so far as is known has not been tried out. The first furnace, rated at 1 ton, was at first given charges of 3,000 pounds, and turned out about 10 heats, or 15 tons in 24 hours, at a power consumption of around 315 kw. h. per ton, with a power factor of about 55. The electrode consumption was about 5 pounds per ton. The first linings of fire brick or rammed in carborundum, coke dust, and fire clay had a very short life.

On 24-hour tests of the 1-ton Snyder furnaces made in 1917 with hand operation one furnace made 38 heats of 2,000 pounds each in 48 hours at an average power consumption of 285 kw. h. per ton; average power factor, 56; the other made 40 heats in 48 hours, at an average of 290 kw. h. per ton. One furnace made 10 consecutive heats in 11 hours and 40 minutes, at an average power consumption of 271 kw. h. per ton, varying from 254 to 288. Four consecutive heats were made at an average of 261 kw. h. per ton. This period was exceptionally free from delays, such as waiting for the crane to handle the ladles, only 25 minutes being occupied by anything other than charging, pouring, or melting. The furnace ran at an average power input of 400 kw., an average voltage of 121, and an average power factor of about 53.

Throughout the whole test (52 heats, 65 hours) of the furnace the power consumption averaged 285 kw. h. per ton. The arc was on 57 per cent of the time; 20 per cent was occupied in pouring, charging, and pulling slag; 6 per cent was lost through electrode troubles, mainly breakage due to poorly fitting joints; 3 per cent through trouble with the tilting motor; and 14 per cent through delays not due to the furnace itself.

By eliminating avoidable delays it should be possible to make 21 heats in 24 hours, at 270 kw. h. per ton, on this leaded bearing metal, poured at about 1,150° C. (2,100° F.). This large production is due in part to the high power input, about 400 kw., and in part due to the rapid mechanical charging made possible by the design of the furnace roof, which may be raised rapidly and tilted back to open

completely the top of the furnace, and the charge may be dumped in from a bucket or clam shell without any handwork save a little poking to arrange the charge.

METAL LOSSES.

Metal losses were determined on 42 heats of this test, though not with a great degree of accuracy. The net loss was about $4\frac{1}{2}$ per cent; that is, much greater than the loss with the smaller furnace, more than on fuel-fired crucible furnaces, and only 1 per cent below that given by the open-flame oil furnace. This loss is due obviously to volatilization of lead and antimony, the latter being an impurity arising from the use of scrap hard lead as a part of the charge in the making of the bearings. Any lead to be added to the charge is not put into the furnace but is added in the ladle.

The metal losses on the whole year's operation of the Snyder and Rennerfelt furnaces together, calculated on an inventory basis, were somewhat lower than those found in the tests, and though the Rennerfelts are conceded to give somewhat lower metal losses than the Snyders, thus reducing the figures of metal loss when the results of both types of furnace are lumped, it is doubted whether this difference is enough to give the inventory figure if the average loss with the Snyders was as great as indicated by the tests. It therefore seems probable that the net loss for the Snyders is between 3 and 4 per cent rather than above 4 per cent.

The fact that lead may be volatilized from copper by a direct arc is utilized by Hill and Luckey¹² for rapid determination of the lead content of copper. They play a 220-volt arc 0.5 cm. long, at 10 amperes, on 0.4 gram of copper, observe the arc spectrum through a spectroscope, and record the time required for the lead lines to disappear. This time is proportional to the lead content of the copper. In two minutes 0.016 per cent lead disappears, and in eight minutes 0.216 per cent.

It is not surprising then that a direct arc playing on an alloy with, say, 15 per cent lead should cause a distinct loss of lead.

According to figures from the Chicago Bearing Metals Co., in 49 working days (two shifts of nine and one-half hours each) in December, 1917, and January, 1918, the two Snyder furnaces together melted 2,600,000 pounds, or about $13\frac{1}{2}$ heats of one ton each per furnace per 19-hour (two-shift) day.

The average cost of materials used for relining the Snyder furnaces for three months, exclusive of labor, was $21\frac{1}{2}$ cents per ton. The roofs were made of fire brick, the hearths (rammed in) of a

¹² HILL, C. W., and Luckey, G. P., The spectroscopic determination of small amounts of lead in copper: *Trans. Am. Electrochem. Soc.*, vol. 32, 1917, p. 835; *Met. and Chem. Eng.*, vol. 17, 1917, p. 659.

mixture of four parts carborundum fire sand, two parts fire clay, and one part molasses. The labor charge would probably make the cost about 35 cents per ton.

REPAIR COSTS.

The average electrode consumption for three months was 2.9 pounds per ton, which does not include losses from breakage, the broken pieces being used as stock for machining nipples for the Rennerfelt electrodes. On a one-week test, in which 163 tons were melted, 573 pounds were actually consumed and 159 pounds broken pieces were set aside as nipple stock. The total consumption was then $4\frac{1}{2}$ pounds per ton; aside from breakage, $3\frac{1}{2}$ pounds per ton. At 3 pounds per ton, at the 1918 price of 25 cents per pound for graphite electrodes, the electrode cost is 75 cents per ton. No water-cooled ring such as is common on steel furnaces is used where the electrode enters the roof, so that the electrode becomes red-hot for some distance outside the furnace and burns away on the surface above the roof. Suitable water-cooling or a suitable "economizer"¹² at the point of entrance might reduce the electrode consumption. The total cost of upkeep for refractories, the labor in relining, and for electrodes was therefore about \$1.10 per ton.

DATA ON OPERATION.

In April, 1918, through the courtesy of the Chicago Bearing Metal Co., a representative of the Bureau of Mines was permitted to take the data on the regular operation of the Snyder furnaces while melting about 140 tons of metal. Fifty-two heats were run in one furnace in four days each; heat was 1 ton (except one 1,500-pound heat) of leaded bearing bronze and 13 heats were melted in about 19 hours (two $8\frac{1}{2}$ hour shifts), 103,500 pounds were melted with 14,460 kw. h., or 278 kw. h. per ton. If the first two heats made after the cooling period between shifts be excluded, to compute results on a 24-hour basis, the power consumption required was at the rate of 265 kw. h. per ton.

On 34 heats, from 67,500 pounds of the leaded bearing metal, less 147 pounds estimated nonmetallic, or 67,453 pounds metallic charged, 64,011 pounds were poured (weighed in the ladles), giving a gross loss of 3,442 pounds or 5.1 per cent. The 2,526 pounds slag, estimated by the Chicago Bearing Metal Co. as 25 per cent recoverable metallic, reduce this loss to 2,810 pounds, or to 4.15 per cent net loss.

In the other furnace, for 36 heats of 1 ton each of leaded bearing metal, three days' run, 12 heats, averaging about 18 hours, 72,000

¹² Anonymous, Device cools gas around furnace electrodes: Foundry, vol. 48, 1920, p. 581.

pounds total were melted with 11,050 kw. h., or 306 kw. h. per ton. The reason for power consumption being more in this furnace than in the other is partly the shorter running time and partly the larger delays, not due to the furnace. It is also probable that the average pouring temperature of the output of the second furnace was somewhat higher.

The average of both furnaces for 88 heats was 290 kw. h. per ton on 18 to 19 hour two-shift operation. While the arc was on the average power input was about 310 kw. The average power factor on a week's test, made by the meter department of the Commonwealth Edison Co. at this time, was 35 for one Snyder furnace and 37 for the other. As these furnaces were designed to give a power factor of about 70, it seems probable that part of the low power factor resulted from the arc containing the vapor of volatile metals such as lead or zinc.

A power factor of 36 was obtained in this 1918 test, one of 55 in the 1917 test, and one of 50 in the 1919 test. The effect of these factors on the cost of operation can be realized by noting that power was bought under a contract having the usual demand charge, in this case \$1 per month per kw. maximum demand, as shown by the demand meter, plus an energy charge of about 1.21 cents per kw. h. for the first 30,000 kw. h., 0.68 cent for the next 70,000 kw. h., and about 0.50 cent for power above a total of 10,000 kw. h. If the power factor under average operating conditions is under 70, it is provided that the demand charge shall be multiplied by 70 and divided by the actual average power factor. The average demand of the two Snyder furnaces together is, say, 600 kw.

TABLE 38.—*Effect of power factor on cost.*

Power factor.	Monthly demand charge.	Excess monthly cost from low power factor.	Excess yearly cost for two furnaces.
70 or over.....	\$600.00		
55.....	$600 \times \frac{70}{55} = 763.63$	163.63	\$1,963.56
50.....	$600 \times \frac{70}{50} = 840.00$	240.00	4,080.00
36.....	$600 \times \frac{70}{36} = 1,166.66$	566.66	6,800.00

Suppose the two furnaces produce metal at the rate of 80 tons a day on 24-hour operation, 20 tons a day on 19-hour, two-shift operation, or 10 tons a day on 8½-hour, one-shift operation, the output per furnace per year of 300 working days would then be as follows: 24 hours, 9,000 tons; 19 hours, 6,000 tons; 8½ hours, 3,000 tons.

TABLE 39.—*Yearly excess cost from low power factor.*

Power factor.	Excess cost per furnace a year.	Approximate excess cost per ton a year.		
		24 hours.	19 hours.	8½ hours.
70.....	None.	None.	None.	None.
65.....	\$681.78	\$0.10	\$0.15	\$0.30
60.....	2,040.00	.23	.34	.68
55.....	3,469.00	.38	.57	1.13

In 1918 the Chicago Bearing Metal Co. was using the two Snyder and two Rennerfelt furnaces two shifts, and three 2,600-pound capacity open-flame oil furnaces one shift, on leaded bearing bronze. A few small oil-fired crucible furnaces were running one shift on yellow brass and manganese bronze, no attempt being made to operate the electric or the open-flame furnaces on these high-zinc alloys because of the metal losses being higher than in the crucible furnaces.

IMPROVEMENTS.

In 1919 General Electric (Seede) regulators were put on the two Snyder furnaces in order to save labor, one man (with help during charging and pouring) operating two furnaces, whereas with hand regulation one man was required per furnace. It was hoped that the average rate of power input would be increased and the power factor raised by holding a longer arc and hence taking a higher arc voltage than with hand regulation. It was found possible to hold a higher power input, so high in fact that the transformers would not withstand, without undue heating, the rate of power input that gave maximum output and thermal efficiency. In 52 days of nine hours each one Snyder furnace melted 952,441 pounds, or 476 tons, in 458 hours; the other Snyder furnace melted 874,000 pounds, or 438 tons, in 465 hours. Both furnaces together produced 904 tons in 923 hours, or about 1,950 pounds per hour, running single shifts of nine hours, with automatic control of electrodes. The power consumption was reduced to about 300 kw. h. per ton for the leaded bearing metal; that is, almost to the 1917 figure for 24-hour operation with hand control of the arc. This power input had to be reduced in order to safeguard the transformers, so this low power consumption could not be continued.

The roofs, made of cheap fire brick, last 90 to 140 heats, spare roofs being kept on hand. The rammed-in carborundum and fire-clay hearth lasts about 200 heats.

The power factor can be maintained at 50 to 55, but to raise it above those figures has not been possible, although expert electrical engineers attempted to improve it,

The makers of the Snyder steel furnace have put a two or three phase, lower voltage type of furnace on the market, which gives a higher power factor on steel than the older form of single-phase, high-voltage Snyder. Undoubtedly it is possible to find some way of constructing a direct-arc furnace and operating it on bronze or red brass so as to have a power factor high enough to avoid penalties, but so few attempts have been made to apply true direct-arc furnaces to copper alloys, high-voltage Snyder furnaces being the only example of the direct-arc type commercially melting such alloys in this country, that data on the best methods of doing this are lacking. It seems probable that the task would be less difficult in melting a true bronze, neither copper nor tin being very volatile, than in melting alloys containing much lead, antimony, or zinc, all of which are decidedly volatile at arc temperatures and tend to make an arc stream that is largely metallic vapor rather than carbon vapor.

Although the metal losses in any direct-arc furnace running on yellow brass are prohibitive and are high in the 1-ton Snyder, even on leaded bearing metal, owing to the impossibility of luting the furnace and running it tightly closed, the users consider that the metal losses are at least 1 per cent lower than for open-flame oil furnaces melting the same charge.

The field of the direct-arc furnace is seen to be very small, only that of melting alloys with very little or no zinc, and the furnace is handicapped by the metal losses on alloys high in lead. Its reliability is only fair, for the life of roof and lining is not very long; to balance this disadvantage is the possibility of putting a spare roof in place without much delay.

Because it can be charged rapidly by opening the roof and because of the relatively high power input for the 1-ton furnace, the Snyder furnace has so high a thermal efficiency and so great a rate of production that under the conditions in which it has been used it has shown lower melting costs than crucible or open-flame melting, even though the metal losses were high.

The electrical characteristics of the old-style Snyder furnace are poor, as the 1-ton furnace, taking 300 to 400 kw. at 50 to 55 power factor, draws a load of about 700 kv. a., and to avoid current disturbances to other power users on the same line it has to be put on a separate line. The excessively low power factor involves a penalty that, on single-shift operation of the furnace, may amount to an excess power charge of over \$1 per ton. Because of the large production and the use of automatic regulators the labor cost per ton is low.

The price of the 1-ton single-phase furnace or of the more modern two-phase furnace, with automatic electrode control and all auxiliary

transformers, switches, switchboard, and meters, except a watt-hour meter, on January 19, 1920, was \$17,000. For a single-phase hand-operated furnace without automatic electrode control the price was \$16,000.

WILE FURNACE.

In 1912 another furnace, the Wile, was advertised for use in brass melting, though intended primarily for smelting tin dross, melting gold concentrates, etc.¹¹ At the plant of the Great Western Smelting and Refining Co., St. Louis, some 10 years ago one of these furnaces was tried for smelting lead and tin drosses, but was not considered satisfactory and was not put into regular commercial operation. About the same time both the tilting and tapping forms were also tried out on silver at the plant of Handy & Harmon, Bridgeport, Conn., with unsatisfactory results and were not used for commercial work.

The furnace was planned to melt brass with a deep slag or molten glass, and the heating was from the resistance of the slag alone if the furnace is operated at a low voltage, with the upper electrode or electrodes buried in the slag. If operated at a higher voltage, with the electrodes not touching the slag, but drawing arcs to it, the heating is due to a combination of arc and slag resistance. One of these furnaces was used for a time by the Westinghouse Electric and Manufacturing Co. in melting cast iron for making cast resistance grids. The furnace used 200 to 300 pounds of glass slag for a 100-pound charge of cast iron and had to be run with the electrodes arcing to the slag and not buried in it. The arc heating, in this case at least, clearly overbalanced the resistance heating, and the furnace seems classifiable more properly under arc furnaces than under resistor furnaces.

The furnace is similar in principle to the general run of ferro-alloy smelting furnaces, such as those for ferromanganese. In these the charge is manganese ore, flux, coke, and metallic iron if not enough is present in the ore; the manganese, reduced from the ore in the reaction zone and alloyed with the iron, drops through the slag layer into the layer of fused alloy below. From time to time slag and metal are tapped and more charge is added, the furnace operating continuously. Such smelting furnaces are efficient and suitable for the work they have to do, but they are *smelting* furnaces, not melting furnaces, and are designed to heat the solid charge or the molten slag rather than the lower layer of fused metal.

¹¹ Wile, R. S., An electric furnace for the treatment of tin dross: Met. and Chem. Eng., vol. 10, 1912, p. 495; Reduction of tin drosses in an electric furnace: Trans. Am. Electrochem. Soc., vol. 18, 1910, p. 206; The Wile furnace: Trans. Am. Electrochem. Soc., vol. 26, 1914, p. 252; U. S. Patents 947,723, Jan. 25, 1910, and 1,070,568, Aug. 19, 1913.

It was reported that a Wile tapping shaft furnace, with a conducting bottom and one or two upper electrodes extending into the slag layer, was used for a time in smelting concentrates from brass furnace ashes, dirty brass borings, etc., the furnace operating much like an electrically heated cupola. No data on its operation has been obtained. It may have run on a commercial scale, but if so its operation was not long continued.

The Wile¹⁴ melting furnace advertised for brass was a tilting furnace having a furnace shell mounted to tilt on trunnions, and it has a "teakettle" spout for pouring the metal from beneath the slag and retaining the hot slag in the furnace for the next heat. In the single-phase form two upper electrodes, in series, project through the roof and directly underneath are two lower electrodes, also electrically connected. In starting the cold furnace arcs are drawn from the upper electrodes to the lower ones until the glass is melted. The lower electrodes are short-circuited by the molten metal and no appreciable current flows through them.

One of these furnaces was installed, for instruction purposes, at the Carnegie Institute of Technology. In 1912 two qualitative runs of this furnace were made by D. A. Lyon of the Bureau of Mines.

The furnace was rated to hold 500 pounds of brass, but the power supply available was only 50 kw. In the first run it took 1½ hours and 72 kw. h. to melt the glass slag ready to run; 22 pounds of scrap red brass were then charged and in 45 minutes 31 kw. h. were used. During this run much zinc oxide was emitted from the furnace. The metal could not be poured cleanly, free from slag, so the slag was poured also; 17.8 pounds of ingot were poured. On grinding and concentrating the slag 4½ pounds of brass shot were obtained. The slag contained 3.6 per cent zinc; 0.7 pound of metal, or about 3 per cent, was lost.

After this run fresh glass was charged into the hot furnace. It took two hours and 91 kw. h. to get the glass melted and fully fluid.

Twenty-three pounds of copper were charged and 25 kw. h. more were supplied to the furnace. The power was then cut off and 13 pounds zinc added in small pieces at intervals. Explosions took place as each piece passed through the superheated slag and clouds of zinc fume were evolved. About 15 kw. h. more were supplied and an attempt made to pour, but only 6.8 pounds ingot were obtained. The slag contained 9.4 per cent zinc.

A further test was made by one of the writers in January, 1913, to study the behavior of the furnace. As Mr. Wile could not be present

¹⁴ Wile, R. S., An electric furnace for the treatment of tin dross: *Met. and Chem. Eng.*, vol. 10, 1912, p. 495.

and sent no directions, the furnace was operated on the basis of his previous verbal instructions to Mr. Lyon.

The chrome-brick lining had been considerably eroded in the previous runs, but was patched up with magnesite cement. The furnace was started on about 150 volts and about 80 pounds of broken window glass was fed in slowly. It took 1 hour and 50 minutes and 67 kw. h. to melt the glass. In order to get the furnace to take 45 kw. it was necessary to run short arcs to the slag. When charging was begun the furnace took 150 volts, 300 amperes; 94.3 pounds of leaded bronze containing 6 to 7 per cent tin, 10 to 12 per cent lead, $1\frac{1}{2}$ to 2 per cent zinc, the balance copper, was charged in the form of ingots weighing about 15 pounds each. Each ingot was allowed to melt before the next was added in order to avoid freezing the molten mass. All the charge was added in 40 minutes, and heating was continued 30 minutes more. Sixty-seven kw. h. were used in 1 hour and 10 minutes. The metal was then poured, but could not be poured cleanly from the slag; everything was poured into ingot molds and the slag was later broken off. Eighty-seven and nine-tenths pounds of ingot were obtained and one-half pound more as buttons by crushing and jiggling the slag. The net metal loss was 5.9 pounds, or over 6 per cent. The slag contained chromium from the chromite lining. The power factor averaged about 94. The electrodes lost $3\frac{1}{2}$ pounds during the run.

Very unpleasant fumes were evolved while the glass was being melted by the arcs, the lining was eroded by the slag, and the metal, except in the middle of the pour, could not be poured reasonably free from slag; the first and last metal were badly mixed with the slag.

Mr. Wile later said that the tilting type had been abandoned in favor of the tapping type. Though the test could not give any direct evidence on power consumption, it did indicate that melting the glass on 10-hour operation, for example, would take so long and use so much power to melt the glass that the furnace would be inconvenient and impractical.

If the solid material charged floated on the molten material, or if it could be held in the slag until it melted, as in ore-smelting furnaces, an arc-slag resistance furnace might be thermally efficient; in melting brass, however, the solid material charged sinks under the molten slag and goes to the bottom of the molten metal, if any is molten. The heat generated above or in the slag is applied therefore by conduction only to the metal, and slag is a relatively poor conductor of heat. Wile furnaces have been suggested for melting steel, but

Cone⁷⁵ says that the Wile furnace has disappeared altogether from the list of active steel furnaces.

A Wile furnace was erected at the plant of the Tottenville Copper Co. in 1919 "to make experiments on certain alloys, but it did not give satisfaction, is not now in use, and has been dismantled."⁷⁶

The Haynes Stellite Co.⁷⁷ tried a Wile furnace for melting Stellite, but found the molten glass too destructive to the lining.

BENNETT FURNACE.

The Bennett furnace, the next on the list, is structurally similar to a direct-arc furnace, but is not an arc furnace because the voltage between electrode and bath is kept so low that no true arc is formed. Heat is generated by the resistance of a poor contact between electrode and metal.

The Bennett furnace has been described only in the patent,⁷⁸ and the patent describes the process rather than the furnace. Other applications on important features are still in the Patent Office. The furnace has been developed at and for the Scoville Manufacturing Co., Waterbury, Conn., and has been operating commercially for several years. Six 1-ton furnaces and one 5-ton furnace are melting alloys ranging from yellow brass to copper. The 5-ton furnace is the largest electric furnace yet built for brass.

The Bennett furnace was adopted after exhaustive experiments with many other types; hence its selection and use by the Scoville Manufacturing Co. indicate that it is suitable for nonferrous melting. It has not yet been decided, however, whether or not the furnace will be made available for other users; unless and until the furnace is put on the market it is likely that the present policy of secrecy as to the exact design of the furnace and as to its performance will be maintained.

According to the patent the furnace is three-phase, the 1-ton size taking 2,500 amperes per electrode at a voltage of from 18 to 20 between each electrode and the charge, and the 5-ton furnace taking 4,200 amperes and 32 to 40 volts. In a three-phase furnace, with the probable power factor of a furnace of this type, this figure amounts to a power input of about 150 kw. in the 1-ton and 500 kw. in the 5-ton.

An arrangement of the electrodes similar to that used in the Heroult steel furnace is advised in order to effect the slight circulation of the metal and the mixing of the charge that can be obtained by utilization of the electromagnetic field. That the cir-

⁷⁵ Cone, E. F., The status of the electric steel industry: Iron Age, vol. 105, 1920, p. 75.

⁷⁶ Personal communication, Jan. 8, 1920, from Tottenville Copper Co.

⁷⁷ Personal communication, Feb. 12, 1920, from Elwood Haynes.

⁷⁸ Bennett, M. H., U. S. Patent 1,337,305, Apr. 20, 1920.

culation in the Heroult is not enough for mixing steel containing difficultly alloying elements has, however, been pointed out by Hess,⁷⁹ and this would not be sufficient to avoid local overheating.

Bennett lays great stress on holding the rate of power input down to a point where the thermal conductivity of the charge can take care of and distribute the heat without causing excessive local superheating near the electrode tips. The rate of power input used is about half that used in furnaces of similar sizes in other types. The ratio of heat lost by radiation to that usefully employed will therefore be high, and the efficiency will probably not be high, though the generation of heat in contact with the charge itself gives less chance for thermal loss than in many other types of furnace, as this factor makes for efficiency. On the other hand, the use of lower voltages and higher currents than are required by most types means a greater loss of heat in electrodes and leads.

Figure 24 shows the furnace as described in the patent, but the diagram probably represents the form actually used only as closely as was deemed necessary for the purposes of the patent.

The electrodes are probably controlled automatically by electrical devices which maintain the electrodes at such a distance from the charge that the desired voltage value is not exceeded. The auxiliary electrode shown in the tap hole does not carry power but allows connection of meters and control devices in order to maintain the proper voltage between the true electrodes and the bath. It is easy to see how the furnace can operate in this way after the charge is molten, but some difficulty might be expected in its operation on solid material before the charge has melted.

The furnace is said to operate without a slag, to be tightly closed, to be charged at the start with the whole charge, including the zinc, and to give a net metal loss of less than 1 per cent, even on alloys high in zinc and lead. The power consumption is said to compare favorably with many of the other commercial electric brass furnaces.

The furnace is of much interest theoretically, and the fact that it has operated under rolling-mill conditions in commercial production indicates that it has future commercial possibilities.

The generation of heat at voltages lower than are commonly supposed to be necessary to hold an arc on alternating current is not entirely unknown.

⁷⁹ Hess, H. L., Electric furnaces as applied to steel making: Mech. Eng., vol. 41, 1919, p. 245; Chem. Abstr., vol. 14, 1920, p. 16.

SCHEMMANN AND BRANN FURNACE.

The Schemmann and Brann⁸⁰ furnace for melting ferromanganese for addition to steel operates as a contact-resistance furnace. The patent states that arc furnaces operating at the usual voltage of

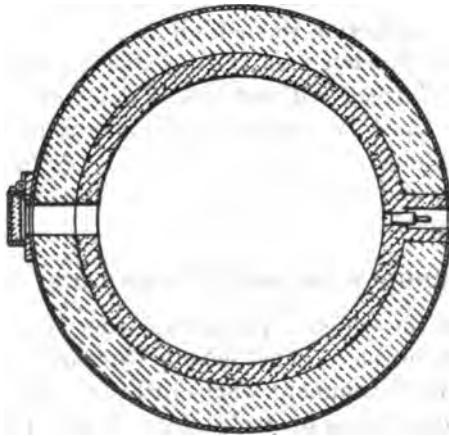
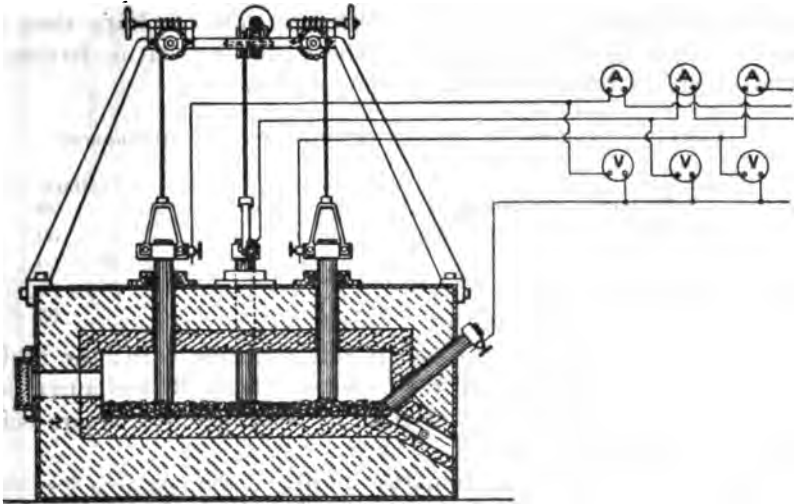


FIGURE 24.—Diagram of Bennett furnace.

steel furnaces—that is, 45 to 75 between electrode and bath—cause volatilization of manganese; if, however, the electrode is lowered so that it just touches the surface of the bath and the voltage is 16 to 18—that is, below the counter electromotive force of the arc, which is generally taken to be about 30 volts—it is possible to ob-

⁸⁰ Schemmann, W., and Brann, J., U. S. Patent 1,056,456, Mar. 18, 1913.

tain, by the contact resistance thus produced, an ample heating of the bath without noticeable loss of manganese.

According to Rodenhauser⁸¹ the Schemmann and Bronn furnace is built much like an ordinary three-phase Heroult, but the voltage between each electrode and the bath is 25. Nearly all the furnaces melting ferromanganese run at a distinctly lower voltage than the steel furnaces of the same make. The voltages used on ferromanganese melting furnaces differ as shown below:

TABLE 40.—Voltage of electric furnaces for ferromanganese.

Furnace:	Voltage.
Schemmann and Bronn-----	25
Keller-----	30
Girod-----	40 to 50
Heroult-----	45 to 55
Nathusius-----	65

The furnaces all carry a layer of slag over the bath; the higher the voltage used the thicker the layer of slag. Rodenhauser considers that an arc is present even at 25 volts, the arc stream being made up of metallic vapor.⁸²

Lemp⁸³ states that on automatic welding machines arcs less than $\frac{1}{16}$ -inch long can be held, and sometimes there has been maintained continuously an arc so short that there hardly seemed to be any actual separation. Lemp concludes that much is to be learned as to current action and current conduction in such an arc.

Bardwell⁸⁴ quotes Korten⁸⁵ as stating that to melt ferromanganese in a direct-arc furnace without loss of manganese by volatilization a low voltage must be employed, the arc must be short, and the hearth offer as much surface as possible. A uniform heating over a large surface and not an intense heating over a small surface area should be sought.

DENOLLY-GRAMMONT FURNACE.

Another furnace, on which little information is available, is the deNolly-Grammont furnace. The deNolly furnace as shown in figure 25 was used at the Grammont works in France for melting ferromanganese for fluid addition to steel. According to the deNolly pat-

⁸¹ Rodenhauser, W., *Ferromangan als Desoxydationmittel*, 1915, p. 26.

⁸² Compare McLeenan, J. C., *Low-voltage arcs in metallic vapors*: *Proc. Phys. Soc.*, vol. 81, 1919, p. 80; *Chem. Abstr.*, vol. 13, 1919, p. 2311.

⁸³ Lemp, H., discussion, *Welding mild steel*: *Bull. Am. Inst. Min. Eng.*, May, 1919, p. 829.

⁸⁴ Bardwell, E. S., discussion, *Bull. Am. Inst. Min. Eng.*, No. 143, Nov., 1918, p. 1652.

⁸⁵ Korten, R., *Stahl und Eisen*, vol. 32, 1912, p. 426.

ent²² this furnace also bears much resemblance to a direct-arc steel furnace, but the voltage used is very low and very large electrodes are

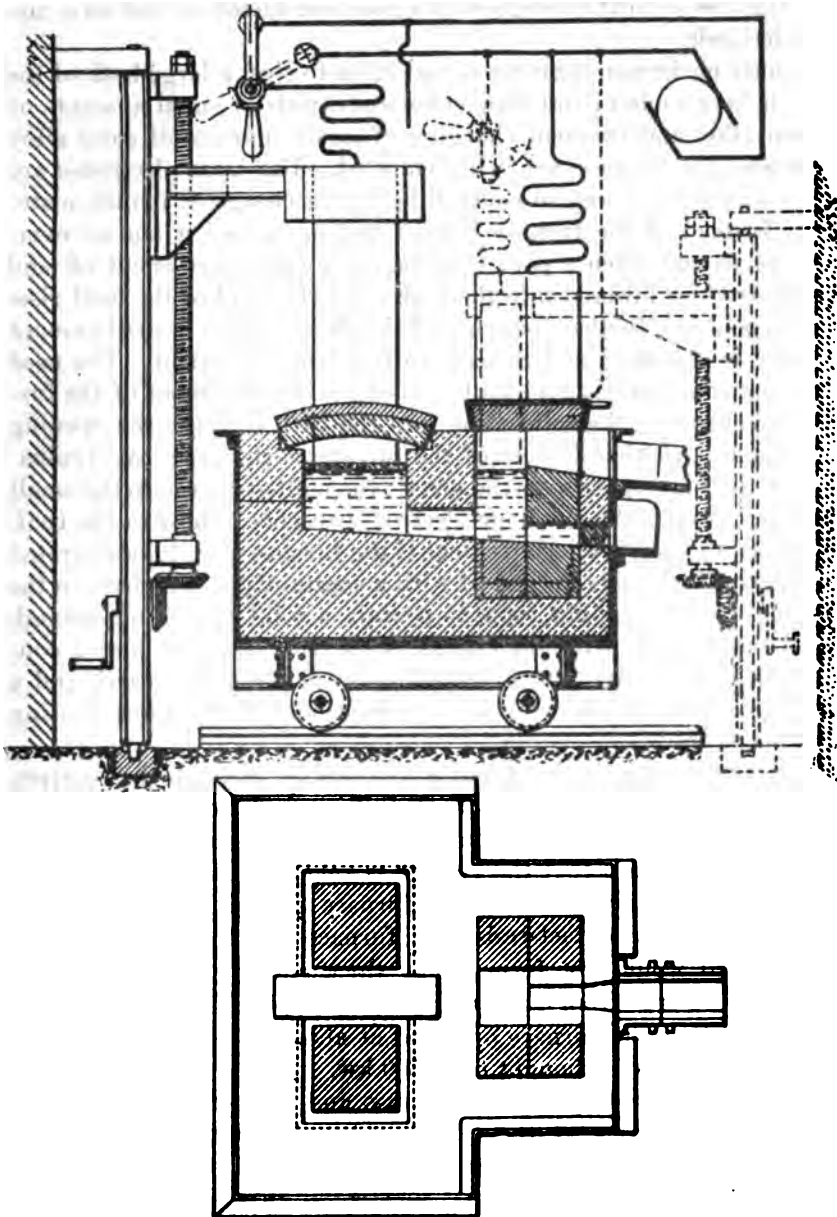


FIGURE 25.—Diagram of deNolly furnace.

used in order to distribute the heat generated and to help reduce local overheating. The carbon electrodes are so large (approximately 20 by

²² deNolly, H., U. S. Patent 1,216,061, Feb. 20, 1917.

20 inches) that they almost eliminate any real roof to the furnace. This arrangement, though avoiding roof troubles, must involve a material loss of heat through such a good conductor of heat as a carbon electrode.

In melting ferromanganese for addition to steel a large bath of the alloy is kept molten, and frequent withdrawals of small amounts of molten alloy and frequent charging of small amounts of solid alloy are made, the furnace not being emptied. The huge electrodes are utilized in a novel way in controlling the discharge of molten material. Instead of the furnace being tilted or tapped it has an overflow spout, and when a pour is to be made the power is cut off and the electrode is lowered into the bath until the level of the melt rises to the spout; by further lowering of the electrode the desired amount of alloy is displaced and is made to run from the spout. The need for a charging door is avoided by raising an electrode out of the furnace by charging the solid alloy into the bath through the opening thus made. Another feature of the furnace is the practical elimination of any open space over the metal; the molten charge in the small space not covered by the huge electrodes reaches almost to the roof. This feature would be inconvenient if the furnace were to be operated for intermittent heats instead of with a constantly molten bath in the furnace, for there is not room to contain a solid charge big enough to fill the furnace after the charge is molten. There is also a constriction in the hearth, or a channel between the main hearth and a sort of a forehearth, in which a certain amount of resistance heating is claimed.

The information on this furnace was supplied through the courtesy of Capt. M. Altmeyer, of the French Technical Commission. Unfortunately he did not have data on the voltage and current used. Though it is certain that the voltage is very low, it is impossible to determine definitely whether the furnace should be classed as a direct-arc or as a contact-resistance furnace like the Bennett. The borax added would not form a slag thick enough to give any appreciable heating from slag resistance. Capt. Altmeyer had the impression that the furnace operated at about 30 volts. The available power supply is thought to be 300 kw.

During the war France needed brass and was short of fuel. As the deNolly furnace at the Grammont works used hydroelectric power, the furnace was tried for melting brass, and two such furnaces were put into regular operation.

Data on the operation of one furnace from December 19, 1916, at 8.35 a. m., to December 21, 8.52 a. m., are given in Table 41. To form a slag 85 to 40 kg. or 75 to 90 pounds of borax were charged with each heat. The charge was 60:40 brass scrap, with the addition of zinc to compensate for loss in melting. As a deoxidizer, 2 kg.

of aluminum per heat were added. The metal was poured direct into ingot molds placed on a turntable in front of the spout.

TABLE 41.—*French test of deNolly-Grammont furnace melting yellow brass.*

Heat No.	Date.	Time. ^a		Charge.		Zn.	Total.	Ingot poured.	Dross.	Power used.
		Charged.	Poured.	Turnings.	Crop ends.					
		<i>H. m.</i>	<i>H. m.</i>	<i>Kg.</i>	<i>Kg.</i>			<i>Kg.</i>	<i>Kg.</i>	<i>Kw. h.</i>
1.....	19.....	8 35	15 00	1,001	1,082	60	2,093	1,913	110	1,090
2.....	19.....	15 05	21 00	1,000	1,084	100	2,134	1,899	104	1,105
3.....	19-20.....	21 00	3 00	1,330	701	37	2,118	2,075	105	1,090
4.....	20.....	3 00	9 15	1,330	702	70	2,102	1,902	130	955
5.....	20.....	9 20	15 00	1,330	703	90	2,123	2,015	105	1,062
6.....	20.....	15 05	20 55	1,330	702	86	2,118	2,046	109	1,083
7.....	20.....	20 55	2 55	1,397	736	76	2,209	2,119	65	1,082
8.....	20-21.....	3 00	8 52	1,330	701	70	2,101	1,998	51	1,012
Total.....						589	16,998	15,957	779	8,429

^a Total time, 48 hours and 17 minutes. French time; same as 3 p. m.

The gross metal loss was 1,036 kg., or 6.1 per cent. No data are given as to whether the turnings contained oil or dirt or concerning the recoverable metal. If it be assumed that there was 2 per cent nonmetallic, oil and dirt, on the turnings and that 25 per cent of the dross was recoverable metal, the net loss would drop to about 4 per cent, a high figure, but not so high as to prevent the use of the furnace in the emergency. It will be noted that on the average $3\frac{1}{2}$ per cent excess zinc, on the basis of the total charge, was added to make up for volatilization losses, the charge being calculated for 58 per cent copper and 42 per cent zinc, in order to give the desired 60:40 brass.

The power consumption and rate of production are not good as compared to American practice, since a furnace taking 175 kw. average load, on 16,998 kg., about 37,480 pounds, or 18 $\frac{1}{2}$ tons, required 8,429 kw. h.; that is, 60:40 brass, doubtless poured at about 1,050° C., was melted on 24-hour operation, at the rate of 450 kw. h. per ton of metal charged, or 480 kw. h. per ton of metal cast.

The production was only some 780 pounds per hour with an average power input of 175 kw. The average charge was 4,650 pounds and the average time per heat six hours. The furnace seems underpowered for so large a charge, though doubtless an increase in the rate of power input, especially if obtained through an increase in voltage, would have resulted in increasing the loss of zinc.

The electrode consumption is given as 10 kg. of carbon electrodes per metric ton of brass, or 20 pounds per short ton. It is said that one man can operate one furnace, six other men helping during casting the ingots, or one man can operate two furnaces with a crew of 10 for aid in casting. With the heats coming out six hours apart it

would seem that by staggering the heats a crew able to handle the pouring of one furnace should suffice for two.

GRAMMONT MODIFICATION OF DENOLLY FURNACE

Grammont⁸⁷ has patented a modification of the deNolly furnace which is evidently designed to make the design more suitable for work on copper and its alloys. Capt. Altmeyer stated that he thought M. Grammont's interest in the furnace concerned more the melting of copper than of brass.

The Grammont modification eliminates the forehearth and channel of the deNolly furnace, being a two-phase furnace with conducting bottom and two large, square carbon electrodes, which almost cover the hearth, as in the deNolly, and require very little refractory roof.

The furnace is rectangular, with charging doors at each end, the pouring spout being in the middle of a long side of the furnace. The overflow pouring scheme of the deNolly furnace seems to be retained. Above the conducting bottom, which rests on the lower metallic electrode built into the furnace, is a nonconducting hearth, with several (five shown in the patent) holes through it. The molten metal in these holes is in contact with the lower electrode and is supposed to be heated by its own resistance to a temperature higher than the body of the bath. This appears to be an attempt to combine heating below the bath in resistor tubes similar to the Hering "pinch-effect" furnace, with heating above the bath by a low-voltage arc or contact-resistance heating.

Now the Hering principle, as will be seen later in the description of the Hering furnace, requires such extremely high currents and low voltages that carrying these currents in carbon electrodes, even the huge ones of the deNolly-Grammont design, is impracticable because most of the heat developed would be in the electrodes, outside the furnace, rather than in the resistor holes or tubes. These holes in the hearth of the Grammont furnace therefore serve merely to make connection between the lower electrode and the bath and can not develop any appreciable proportion of the total energy developed in the furnace. The Grammont furnace must consequently be classed as a direct-arc or a contact-resistance furnace (the voltage to be used is not mentioned in Grammont's patent) rather than as a Hering type resistance furnace, and its power consumption and metal losses should not differ much from those of the deNolly furnace, although in construction (aside from the holes in the hearth) it is rather more simple than the deNolly. The holes in the hearth require either that the furnace be made to tilt (which is seemingly not contemplated), so that all molten metal could be drained from them if the furnace

⁸⁷ Grammont, E., French Patent 487,605, July 17, 1918.

were to be shut down, or else require that the furnace be kept hot continuously, so that the metal will not freeze in the holes and damage the hearth. Evidently the furnace would be simpler and more practical if the lower carbon electrode were extended up to the hearth instead of being connected to it by tubes full of molten metal.

It should be pointed out that for a given power input the low-voltage furnaces, whether they use direct-arc or contact-resistance heating, require high currents and consequently are subject to greater heat losses in leads and electrodes than are furnaces operating at higher voltages, so that the thermal efficiency of low-voltage furnaces tends to be lower than that of higher voltage furnaces using electrodes.

The question whether in these furnaces that structurally resemble direct-arc furnaces local superheating and consequent metal losses can be avoided by using very low voltages and utilizing contact resistance instead of a true arc or by using very large electrodes and attempting to spread the arc over a large surface instead of localizing it can not be answered by the available data. The big electrodes will not wear off smoothly to a perfectly flat, level tip, from which a short but very fat arc low in energy density—that is, generating few calories per unit of a very large area—can be drawn, but will wear off irregularly, with the result that the current will jump from the tiny projection nearest the charge, and thus the heating will be localized.

It is possible that by using a slag, like the borax used in the deNolly furnace, the electrodes might be slightly below the normal level of the slag and the slag be slightly blown away from it by the arc stream, for something of this nature is often noted in an ore-smelting furnace running at a low voltage. At a certain point there is no open, flaming arc, yet the slag does not wet the electrode even though the electrode is buried in it. Such a condition is usually called a submerged arc, and while such an arc is not as hot as a high-voltage open arc, its temperature is still certainly above the boiling point of zinc in brass. Moreover, this condition does not hold when the furnace contains a charge of solid metal at the beginning of a heat. Then the heating is essentially and necessarily localized, and though the metal may sometimes drip away from the arc as fast as melted and before it can be superheated, yet it must often collect in pockets and be held in the arc long enough to permit material losses of volatile constituents, unless the furnace is so tightly closed that the metal vapor can not escape. It would be difficult in the deNolly-Grammont furnace to make a vapor-tight joint between the huge carbon electrodes and the openings through which they enter and still provide for the necessary adjustment of the electrodes.

It is probable that a low-voltage direct-arc furnace might not have quite as high metal losses as the ordinary direct-arc furnace, and that the metal loss might be held low enough that, under some conditions, the simplicity of construction of the direct-arc type might make it pay to accept the metal loss, but under most American conditions it would seem that the chances were against such a furnace being useful. Until such a type of furnace has been shown to give much lower metal losses than those of the de Nolly test, or with that metal loss, to give materially better production, power consumption, etc., than other types of furnaces that show low metal losses, no form of direct-arc furnace should be used for melting alloys high in zinc. Whether a low-voltage direct-arc furnace would do good work on a red brass, bronze, leaded bearing metal, etc., is an open question, as only the high-voltage Snyder has been used for them.

VON SCHLEGELL-FLETCHER FURNACE.

A suggestion for minimizing the metal loss in a direct-arc brass furnace is found in the von Schlegell-Fletcher patent,⁸⁸ whereby the brass is to be melted down by a direct arc, and after it is melted an extra electrode is stuck in through the pouring spout and the heating is continued by an indirect arc, that is, one between carbon or graphite electrodes above the melt.

To reduce further the surface overheating that occurs even with an indirect arc, the hearth of this furnace is made deep and of small cross section in order to present as little surface as possible to the direct heat from the arc over it. As heated metal stays at the top, this deep hearth will defeat its own purpose; to get the metal in the bottom of the hearth hot enough to pour, the surface layer must be even more violently overheated than it would be with a wide and shallow hearth, so that the scheme would invite rather than prevent metal losses. The deep hearth also offers every opportunity for the composition of the melt to vary between the first and the last metal poured, owing to separation of heavy constituents by gravity. So far as is known this furnace has not been tried or constructed.

DIRECT-ARC FURNACES IN WHICH THE ARCS ARE IN MOTION.

VON SCHATZL FURNACE.

Von Schatzl⁸⁹ designed a furnace consisting of a large upright hoop-shaped trough mounted to rotate like a wheel on a jacked-up axle. The electrodes are set at a slight angle so that their tips enter the trough at the lower part of the hoop; the arcs drawn from the

⁸⁸ Von Schlegell, F., and Fletcher, C. B., U. S. Patent 1,294,837, Feb. 18, 1919.

⁸⁹ Von Schatzl, G., German Patent 231,734, Feb. 28, 1911; von Schatzl, G., and Krieger, E., Austrian Patent, 49,448, Aug. 25, 1911.

electrode to the charge heat the latter, which lies below the electrodes in a small sector of the trough. The rotation of the hoop is designed to mix the charge and to promote homogeneity. Among the possible uses is mentioned the melting of steel. It is not known whether this furnace has been actually constructed or not.

BUCKMAN FURNACE.

Buckman⁹⁰ describes a direct-arc furnace in which the electrodes are eccentrically located and rotate, thus distributing the heat and reducing its localization. The mechanical details as to the making of electrical connection for these moving electrodes are not given. The furnace has not been specifically suggested for brass, nor is any embodiment of the scheme in an actual furnace known to have been made.

SUMMARY IN REGARD TO DIRECT-ARC FURNACES.

Direct-arc furnaces superheat the surface of the charge so that if the charge contains much zinc, lead, or other volatile metals, the metal losses are very great. Furnaces built like direct-arc furnaces but using so low a voltage that no true arc is formed are possibilities concerning which too little is known to make them available for present general use. Direct-arc furnaces could be used on pure copper (if the problem of copper poisoning by fumes is solved), on true bronze, aluminum bronze, or red brass not too high in zinc or lead, but can not be satisfactorily used, even for occasional heats, on yellow brass. Only one commercial application of true direct-arc furnaces to copper alloys is being made in the United States at present.

This type of furnace is the accepted standard for steel and nickel-chromium alloys and works well on Monel metal. It might work on aluminum or aluminum-copper alloys, but no actual trials have been recorded. Aluminum is so prone to absorb gases at high temperatures that the direct arc might not produce a good quality of metal.

STATIONARY INDIRECT-ARC FURNACES.

MOLDENKE FURNACE.

Indirect-arc furnaces do not superheat the surface of the melt so severely as direct-arc furnaces, and the indirect-arc type therefore, has a wider application to brass and bronze.

Perhaps the earliest suggestion of the use of an indirect-arc furnace for brass, though the main object was melting steel, was that of Moldenke,⁹¹ in 1896. The charge was to be placed on an inclined sur-

⁹⁰ Buckman, H. H., U. S. Patent 1,092,764, Apr. 7, 1914.

⁹¹ Moldenke, R. G. J., U. S. Patent 569,221, Oct. 13, 1896.

face, preheated by heat transmitted to this inclined surface by a fuel-fired furnace, melted by the heat radiated from one or more arcs, and then allowed to run into a sump, where it was to be heated to pouring temperature, and from which it was to be tapped. Since this suggestion was made before the first Stassano patent (1898) or the first Heroult patent (1900) at a time when electric-furnace design was in an embryonic state, it is not remarkable that the idea was not carried to fruition. It is more than possible that the principle of preheating the charge by fuel and of finishing the melt by electric heat may be found desirable, though such a development is still in the future.

CONTARDO FURNACE.

In 1901 Contardo⁹² described a tilting indirect-arc furnace more nearly approaching modern furnaces in the relative size and arrangement of parts.

HANSEN'S TEST OF WEEKS'S ZINC FURNACE ON COPPER.

Hansen made a heat or so of pure copper in the Weeks indirect-arc zinc-smelting furnace, the furnace being stationary, and has described the furnace and given data on the test.⁹³

The furnace had the form of a horizontal cylinder with electrodes entering each end through stationary heads, which did not move when the furnace was melted to pour. Starting with a cold furnace, 2,000 pounds of copper ingots were melted with 630 kw. h. in 3 hours and 50 minutes, and 1,926 pounds of copper were recovered, giving a gross loss of 3.7 per cent, part of which was doubtless due to soakage into the fresh lining. From this test Hansen calculates that an indirect-arc furnace should melt brass at less than 300 kw. h. per ton. Having made no tests on brass in the Weeks furnace, he does not commit himself very definitely as to metal losses.

Weeks⁹⁴ has discussed the possible application of his zinc furnace to brass, but the discussion is not based on actual test.

SHIPTON FURNACE.

Shipton⁹⁵ has suggested a modification of the ordinary indirect-arc type especially intended for brass melting. In the Shipton furnace the metal is charged, for preheating, on a tiltable refractory shelf located above the arcs, but below the roof. After the metal is preheated it is dumped into the melting chamber proper by tilting the shelf. This is an attempt to utilize, more directly than merely

⁹² Contardo, R. C., U. S. Patent 688,393, Dec. 10, 1901.

⁹³ Hansen, C. A., Electric melting of copper and brass: *Trans. Am. Inst. Metals*, vol. 6, 1912, p. 110.

⁹⁴ Weeks, C. A., Melting nonferrous metals in an electric furnace: *Met. and Chem. Eng.*, vol. 9, 1911, p. 363.

⁹⁵ Shipton, J. D., U. S. Patent 1,313,746, Aug. 19, 1919.

by reflecting it back from the roof, the heat radiated upward from the arcs. Such a construction, however, violates the principle that no complications are permissible in the hot interior of a furnace.

The patent leaves to the imagination the selection of a material suitable for the construction of this movable shelf.

The upkeep and repair of such a part of the furnace would necessarily be so expensive, difficult, and troublesome, as to nullify any advantage that might accrue from more directly utilizing the heat radiated upward from the arcs. It is unlikely that this furnace has ever been operated.

INDIANAPOLIS FURNACE.

Late in 1915 an indirect-arc furnace operated in a stationary position, but tilted to pour, was advertised by the Indianapolis Electro Metals Co. for melting brass. This furnace was single-phase, built in the form of a horizontal cylinder, with electrodes entering at each end, but the electrodes tilted with the furnace, thus avoiding the joint between electrode support and furnace body that was a weak point in the design of the Weeks' zinc furnace. Certain claims were made as to the performance of this furnace, and photographs of the furnace, which had a capacity of perhaps 500 pounds, taken during the pouring of a heat of copper, were sent in.

The data and photographs were lost in the fire that destroyed the Bureau of Mines office at Cornell University in 1916. Letters addressed to the firm thereafter were returned undelivered, so the data and photographs could not be replaced. The furnace was never in commercial use.

STASSANO-TYPE FURNACE TESTED BY HANSEN.

The first indirect-arc furnaces to attain any real commercial use were those designed by Stassano, so indirect-arc furnaces are often referred to as being of the Stassano type, just as direct-arc furnaces are referred to as being of the Heroult type, because Heroult was the first to make commercial furnaces of that type. Siemens^{**} had invented and described both types long before, but as the term "Siemens furnace" does not tell which type is meant the terms "Heroult" and "Stassano" are often loosely used as a convenient shorthand to designate the direct-arc and the indirect-arc type of furnaces.

In 1913-1914 Hansen, instead of continuing with the Weeks furnace, designed a Stassano-type furnace at the Schenectady works of the General Electric Co. for melting brass. This furnace was three-phase, holding 3,500 to 4,000 pounds, and was operated without the

^{**} Siemens, W., English Patents 4,208 of 1878 and 2,110 of 1879.

oscillating motion of the Stassano. When it was tried on yellow brass borings the surface overheating of the charge was so great that the furnace could not be kept closed, clouds of zinc vapor escaped, the metal losses were high; the product analyzed 27 per cent zinc when the borings charged contained 33 per cent, and the furnace was evidently not suitable for yellow brass. The furnace was later used for other purposes and no detailed records of its operation on brass are available.

The Scoville Manufacturing Co. also tested a Stassano-type furnace on yellow brass and encountered such difficulties from superheating of the surface, from excessive zinc loss, sticking of the electrodes, and attack of the electrode coolers, due to condensation of zinc, that they abandoned it.

According to Cone⁹⁷ there was in January, 1920, only one Stassano furnace—that one being used for steel—in the United States. In Italy, however, a large part of the steel furnaces are of the Stassano type, as according to recent figures⁹⁸ there are 40 Bassanese, 26 Stassano, and 22 Angelini furnaces, or 88 indirect-arc-type furnaces, to 14 Heroult and 10 Keller, or 24 of the direct-arc type. The Bassanese⁹⁹ and Angelini furnaces are Stassano-type furnaces said to be made by former employees of Stassano.

Figure 26 shows the principle of a single-phase Stassano-type furnace, and figure 27 the plan of a three-phase Stassano steel furnace.

STASSANO-PETINOT FURNACE.

Schmelz¹ describes Petinot's² modification of the Stassano furnace, which mainly consists in making the roof so thin that enough heat is lost to keep the roof fairly cool and hence increase its life. This furnace was tested at the plant of the Titanium Alloy Manufacturing Co., but the only information given on the test is that 1,000 pounds of copper was melted in a 120-kw. furnace in less than two hours. No tests are cited on brass and no metal losses given.

BUREAU OF MINES TESTS ON SMALL STASSANO-TYPE FURNACE.

In order to get an idea of the metal losses in an indirect-arc furnace the Bureau of Mines made a few tests, starting in March, 1913, in a small furnace built in a 100-pound Schwartz open-flame oil-furnace shell, loaned through the kindness of the Hawley Down-

⁹⁷ Cone, E. F., *The status of the electric steel industry*: Iron Age, vol. 105, 1920, p. 75.

⁹⁸ V. D., *1,000 fours électriques à acier dans le monde*: Jour. du four électrique, vol. 28, 1919, p. 113.

⁹⁹ Bassanese, F., English patent 1,173 of 1915; see also Hvidsten, T. M., and Ingelard, A. H., U. S. Patent 1,359,087, Nov. 16, 1920.

¹ Stassano, E., and Petinot, N., U. S. Patent 1,093,494, Apr. 14, 1914.

² Schmelz, E. M., *Electric furnace for medium temperature*: Trans. Am. Inst. Metals, vol. 8, 1914, p. 261.

Draft Furnace Co. Two graphite electrodes at an angle of about 120° entered through holes cut in the shell; they were protected by water coolers outside the shell and were adjusted by a crude device. The regular fire-brick lining was first coated with a layer of asbestos cement and later with a mixture of magnesite and alundum cement. The furnace was run at about 35 kw. On eight heats of about 100 pounds each of yellow brass ingot, about 30 per cent zinc, poured at an average of $1,050^\circ\text{C}$. ($1,925^\circ\text{F}$.) the following results were obtained: 804.5 pounds were charged; 775½ pounds ingot and 6½ pounds spillings and shot reclaimed from slag were obtained—782 pounds

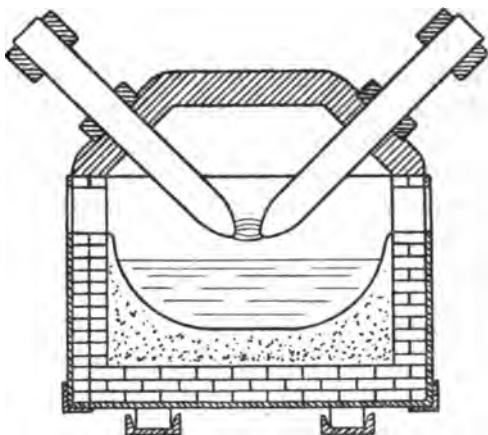


FIGURE 26.—Diagram showing principle of single-phase Stassano-type furnace.

total, a net loss of 22½ pounds, or 2.8 per cent.

In seven heats of an alloy of about 78 per cent copper, 18 per cent zinc, and 4 per cent lead, poured at about $1,100^\circ\text{C}$. ($2,000^\circ\text{F}$.), 686 pounds ingot were charged, 655½ pounds ingot poured, and 14 pounds metal recovered from spillings and slag—total 669½ pounds, a net loss of 16½ pounds, or 2.4 per cent. The use of salt, glass, etc., for a slag seemed to have no measurable effect in reducing the loss.

The furnace was kept tightly closed while running, and, although metal-

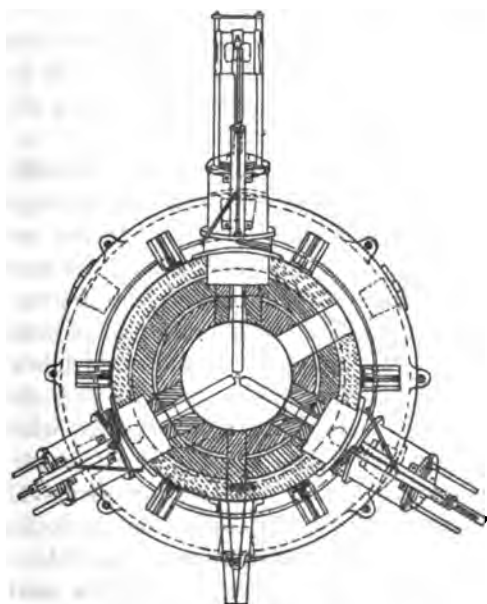


FIGURE 27.—Plan of three-phase Stassano steel furnace.

lic zinc condensed about the electrodes, in the furnace walls, and in the pouring spout, no fume escaped, save that sometimes toward the end of a run the pressure of zinc vapor became strong

enough to blow out the luting about the spout plug. At such times or when the spout was opened to pour, a hissing stream of zinc vapor would shoot out of the spout for several feet.

In order to avoid loss from this cause, the temperature of the melt was judged by the readings of a pyrometer situated in the furnace wall and separated from the charge by about one-half inch of refractory. This reading, of course, showed a temperature lower than the actual temperature of the metal, and only approximate extrapolation to the true temperature was possible.

In attempting to melt yellow brass borings many difficulties were encountered. In order to keep the borings well away from the arc only 56.6 pounds (oil-free basis) were charged in one heat into the furnace warm from a previous heat, though the furnace would hold over 100 pounds of metal, and the furnace was tilted slowly back and forth at the end of the heat in the hope of stirring the melt. When the pyrometer in the hearth read 900°C ., which corresponded to about $1,040^{\circ}\text{C}$. in the metal itself, the furnace was opened. Though the bottom of the melt was hardly hot enough to pour, the top was boiling violently, and on it was much fine, powdery dross. Only 46.25 pounds of ingot were obtained, or about 18 per cent gross loss; 5.5 pounds of dross containing about 90 per cent, or 5 pounds, metallics were recovered, bringing the net loss down to $9\frac{1}{2}$ per cent.

The original borings contained 61.4 per cent copper, 34.9 per cent Zn., 3.7 per cent Pb., and the ingot analyzed 68.5 per cent copper, 28.2 per cent Zn., 3.3 per cent Pb.

Another heat of 100 pounds of borings (97.5 per cent metallic, oil-free basis) with $3\frac{1}{2}$ pounds salt for slag gave $80\frac{1}{2}$ pounds ingot and $11\frac{1}{2}$ pounds metal from slag, a gross loss of about 18 per cent and a net loss of nearly 6 per cent, with the same difficulties as to surface overheating and nonuniformity of temperature as before.

As no watt-hour meter had been installed, accurate figures of power consumption were not obtained. No runs were made on red brass because in 1913 the prices of fuel and crucibles were so low that the field of electric furnace seemed limited to yellow brass, where reduction of metal losses might bring the total cost of electric melting below that of melting with fuel. On yellow brass the indirect-arc furnace showed metal losses equaling or exceeding those of fuel-fired furnaces, and hence did not seem suitable, at least without vital modifications, for such melting. Without modification the local superheating of the surface of the metal, directly beneath the arc, made the indirect-arc furnace of little use for melting alloys high in zinc, although the superheating and the melting losses on the same class of material were not quite as bad as in the direct-arc type. It was realized that if an indirect-arc furnace were greatly underpowered, so that the rate at which heat was

generated by the arc was very low, the thermal conductivity of the charge might more nearly keep pace with the generation of heat, and surface overheating might be minimized. But such an underpowered furnace would be thermally inefficient, its power consumption per ton of metal melted would be high, and its rate of production low, so that its use, even if the excessive metal loss could be thus prevented, would not be economical. The use of a number of very small arcs instead of one large one, in order to distribute the source of heat, is possible, but tends to complicate the furnace and to increase its cost.

RENNERFELT FURNACE.

Only one stationary indirect-arc furnace, the Rennerfelt,³ has found commercial application to nonferrous metals.

According to Cone,⁴ out of 148 Rennerfelt furnaces in operation in the world in January, 1920, 32 were melting nonferrous alloys, 8 of these being in the United States and Canada.

The Rennerfelt furnace differs but slightly in principle from the Stassano furnace, the main difference lying in the shape of the arc. The Stassano is normally a three-phase furnace, using three electrodes placed 120° apart and at an angle of some 15° to 45° from the horizontal, the three electrodes thus pointing toward the center of the furnace, but inclining slightly downward. The arc then plays from any one electrode to the other two, forming a sort of Y lying on its side, as shown in figure 28.

The Rennerfelt is essentially a two-phase three-wire furnace, the two-phase current being taken from a three-phase line by the Scott connection. One electrode is vertical and serves as a common return for both phases. The other two electrodes are horizontal (or in some of the furnaces are movable in a vertical plane so that they may be

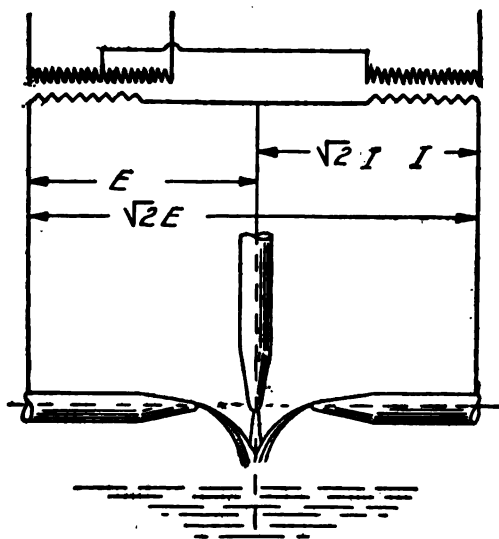


FIGURE 28.—Arrangement of connections to electrodes of Rennerfelt furnace.

³ Rennerfelt, I., U. S. Patent 1,076,518, Oct. 21, 1913; Herlenius, J., U. S. Patent 1,338,681, Apr. 27, 1920. For a modification see Groeger, L. C. H., U. S. Patent 1,383,914, Mar. 16, 1920.

⁴ Cone, E. F., The status of the electric steel industry: *Iron Age*, vol. 105, 1920, p. 75.

at an angle of some 15° above or below the horizontal). The arc plays from each horizontal electrode to the vertical electrode, but the mutual repulsion of the current in the various electrodes throws the arc downward so that it has, roughly, the form of a fleur-de-lis. Thus the shadow of the vertical electrode protects the center of the roof from the direct radiation of the arc, whereas in the Stassano the roof is open to this radiation. Therefore the roof in the Rennerfelt, other things being equal, lasts longer than that in the Stassano. Although the Rennerfelt is a much newer furnace than the Stassano, it has been used much more in this country for steel melting, although for this purpose it is used much less than direct-arc furnaces. In recent Rennerfelt steel furnaces the side electrodes are capable of being tilted so far downward from the horizontal that, if desired, the furnace may be operated with a "free burning" arc—that is, as an indirect-arc furnace—while the cold charge is melting down, thus making the arc stable at the time when a direct arc is unstable. By tilting the side electrodes downward the arcs may thereafter be drawn to the surface of the bath, the furnace being thus finally operated as a direct-arc furnace.

Running the Rennerfelt on brass in this way would result in worse superheating of the surface than running as an indirect-arc furnace, and the tilting of the electrodes would preferably be upward in order to keep the arc away from the top of a new cold charge. As the charge melted the arc might be lowered. However, this tilting involves the maintenance of a refractory ball-and-socket joint or else the cutting away of the refractory about the electrode to allow clearance, thus weakening the furnace and increasing the loss of heat. Most of the furnaces used on nonferrous alloys do not have this complication, and its value for such use is questionable.

The early furnaces had the form of horizontal cylinders, but later furnaces are usually built as upright cylinders, as shown in Plate XII, A (p. 204), the domed roof being removable for easier maintenance.

The Rennerfelt furnace was described by vom Baur,⁵ who claimed power-consumption figures of 168 kw. h. per metric ton (153 per short ton) on red brass and 197 per metric ton (179 per short ton) on copper and less than 160 per short ton on leaded bearing metal. These figures were obtained in a steel furnace hot from a melt of steel, and much melting was done by stored heat, so that the figures were unduly optimistic and have not been attained in regular operation, as might be expected from their representing about 100 per cent of the theoretical efficiency.

⁵ vom Baur, C. H., The Rennerfelt electric-arc furnace: *Trans. Am. Electrochem. Soc.*, vol. 20, 1916, p. 497; Rennerfelt electric furnace operation: *Trans. Am. Electrochem. Soc.*, vol. 31, 1917, p. 87.

SWEDISH DATA ON THE RENNERFELT FURNACE.

The first account of the operation of the Rennerfelt furnaces on brass was given in 1914,^{*} when it was stated that in a hot 60-kw. furnace 209 pounds copper were melted in 30 minutes at the rate of 290 kw. h. per ton and 164 pounds of 60:40 brass (from copper and zinc) in 20 minutes at the rate of 244 kw. h. per ton. According to this account the brass was of excellent quality and there was practically no loss from oxidation. As this report suggested that the furnace was applicable to yellow brass, inquiry was at once made for further details. The reply said that the copper was first melted, and the zinc then added and heated for seven minutes, and that "very little zinc oxide was noticed flying around the furnace." The charge was 44.5 kg. copper and 30 kg. zinc. The alloy obtained analyzed 38.1 per cent zinc. On the assumption that there was no loss of copper, the zinc in the alloy amounted to 27.4 kg. and the total weight of alloy was 71.9 kg., a loss of 2.6 kg., or 3.5 per cent.

In 1916 some further information was obtained on the operation of a 50-kw., 600-pound furnace at the Nordiska Armatur Co. in Sweden. The power consumption (with furnace hot, that is, 24-hour basis) on 60:40 brass made from copper and zinc was about 225 kw. h. per short ton. The metal loss ran from 6 per cent to 4 per cent on 60:40 brass, according to the nature of the charge. On red brass melted by the Atvibaberg Co. the average power consumption was 365 kw. h. per ton and the metal loss under 1.5 per cent.

In a similar furnace running at about 65 kw., at the Svenska Metallverken, the power required to melt copper, with the furnace fully hot at the start, was 220 kw. h. per ton, but if the metal had to be superheated (temperature not given) it took 340 kw. h. per ton. The gross metal loss on scrap copper was under 1 per cent and corresponded with the amount of nonmetallic present, the true metal loss being practically zero. In a 1,200-pound furnace on a run of 254 tons the average power consumption on pure copper was 365 kw. h. per ton, but this probably refers to 24-hour operation. The metal loss was 0.86 per cent, no deduction being made for dirt on the charge, which was mainly old scrap. The electrode consumption with carbon electrodes was 10 pounds per ton, but it is stated that this would be halved with graphite electrodes.

The furnace was reported to melt copper, bronze, and red brass much better than yellow brass, the zinc fumes condensing on the electrodes in melting yellow metal and care being required in cleaning the electrodes from time to time.

As experience was gained with the furnace its agents saw that it was not applicable, under American conditions, to alloys high in zinc,

^{*} Anonymous, A small-sized electric-arc furnace: *Met. and Chem. Eng.*, vol. 12, 1914, p. 275.

and in their advertising literature carefully point out that attempts to melt alloys with much more than 20 per cent zinc are not desirable.

Johnson⁷ has shown that at the normal pouring temperatures the pressure of zinc vapor for the various brasses is as follows:

TABLE 42.—*Partial pressures of zinc vapor for different brasses.*

Alloy:	Partial pressure of zinc.
Cu 60, Zn 40-----	atmospheres__ 0.78
Cu 65, Zn 35-----	do_____ .71
Cu 70, Zn 30-----	do_____ .63
Cu 80, Zn 20-----	do_____ .33

On account of the high volatility, at pouring temperature, of zinc in the alloys rich in zinc it is obvious that not much local overheating of the surface will be needed to cause a large loss of zinc, and that there is more leeway on the 80:20 alloy. Commercial practice with the Rennerfelt agrees with the theory. As a matter of fact, even on a 20 per cent zinc alloy the loss in an indirect-arc furnace is higher than in other types of furnaces that are freer from surface overheating, and the operation of the indirect arc furnace is troublesome.

Notwithstanding the metal loss, the furnace has been used in Sweden on alloys high in zinc. The data on a test made in June, 1919, follow:

MELTING YELLOW BRASS IN A LOW-POWERED RENNERFELT.

The furnace was of about 3,000 pounds capacity, operated at about 100 volts, and had a 225-kv. a. transformer. The power input was, however, held down to about 110 kw. (93 to 127) in order to avoid surface overheating as much as possible. After a heat was poured the charge of scrap with 42 to 45 per cent zinc was charged, pouring and charging taking about an hour. The furnace was fully hot at the beginning of the test, and was operated 23 hours a day, not being run from midnight to 1 a. m.

Eight heats of 2,860 pounds (2,750 pounds scrap plus 110 pounds zinc added at the end of the heat), or 11.44 tons, were melted in 38 hours elapsed time, less two midnight periods, or 36 hours, 3,478 kw. h. being used, or 304 kw. h. per ton. The rate of production was only 635 pounds per running hour, leaving out the idle hour at midnight. The electrode consumption was 4½ pounds per ton of metal melted; 5-inch diameter jointed carbon electrodes with graphite nipples were used. The power factor was about 85. Cast iron instead of copper or brass water coolers were used about the electrodes, as the volatilized zinc deposits on the nonferrous coolers and attacks them.

⁷ Johnston, J., The volatility of the constituents of brass: Jour. Am. Inst. Metals, vol. 12, 1918, p. 21.

The net metal losses are given as $2\frac{1}{2}$ to 3 per cent on the metallic basis, oil and dirt on the charges being allowed for. To 2,750 pounds of scrap 110 pounds of zinc, or 4 per cent of the weight of the scrap, are added to compensate for the volatilization loss and maintain the composition.

Such a loss is greater than that in good fuel-fired crucible practice in American rolling mills. Under Swedish conditions, however, melting yellow brass in the Rennerfelt is cheaper than in fuel-fired furnaces, even at the low power input used, which gives a low rate of production and a high power consumption, considering that the furnace was large, the melting point of the alloy low, and the furnace was operated 23 hours a day. Probably the prices of zinc and of crucibles do not differ greatly in Sweden and in the United States; but at the time of this test coal was costing \$50 to \$75 per ton in Sweden and coke was correspondingly high, whereas electric power could be had at 1 cent or less per kw. h. Melting with coke therefore cost so much more for fuel than melting with electricity—say \$15 to \$20 per ton of brass for coke melting against \$3 per ton for electric melting—that the loss of even 3 per cent more zinc or 60 pounds per ton, at, say, 10 cents per pound, or \$6, was far more than made up by the \$12 to \$17 lower cost of power than of coke, plus the saving through elimination of crucibles.

Under American conditions melting 60:40 brass in a stationary indirect-arc furnace would be decidedly uneconomical. A firm normally melting red brass in such a furnace could, of course, make an occasional heat on an alloy high in zinc, at the cost of some extra metal loss, by reducing the normal rate of power input and thus lessening surface overheating. If, however, the plant were to produce any notable proportion of alloys high in zinc, a type of furnace giving better results on such alloys should be chosen.

DATA ON A 100-KW. RENNERFELT FURNACE AT GERLINE BRASS FOUNDRY.

Through the courtesy of the Gerline Brass Foundry Co. a representative of the Bureau of Mines was allowed to watch the 100-kw. Rennerfelt furnace in operation at that company's plant and was given full data on the first 117 heats made with it in 1917, though metal loss determinations had been made on some 50 heats only.

The arcs normally operate at about 100 volts, but by throwing switches another transformer tap can be used, giving 70 volts. Each phase contains a two-coil reactance, and by switches all, half, or none of the reactors may be used. If the arcs are snappy when starting up from the cold, all the reactance is thrown in; with the furnace at full temperature and the arcs stable it is all cut out. The power factor varies from 75 to 85.

The furnace shell is about $4\frac{1}{2}$ by $4\frac{1}{2}$ by 4 feet high, and the furnace tilts on large hollow trunnions, through which the side electrodes, which are not capable of vertical adjustment, enter the furnace body. The electrodes are graphite, the side ones being 2 inches in diameter, and the top one, which is the common return for both phases and hence carries more current than the side ones, being 3 inches in diameter. The side electrodes carry about 600 amperes each, in normal operation. By means of rods and gears the side and top electrodes are all regulated by handwheels at the back of the furnace, from which position the switchboard and meters can be seen. The electrodes pass through water coolers, but the electrode holders themselves are not water cooled. The electrode control does not allow the electrodes to be drawn completely out of the way while charging, but the electrode holders have to be unscrewed to allow the electrodes to be pulled back out of the way, as is regularly done when charging.

The furnace holds about 650 pounds of metal. The bottom of the hearth is $10\frac{1}{2}$ inches below the horizontal electrodes. As the arcs are deflected downward, the tip of the arc flame comes rather close to the surface of the charge. The shell is lined, first with 1 inch of asbestos, within which is a $4\frac{1}{2}$ -inch layer of Woodland fire brick, and within this a $2\frac{1}{2}$ -inch layer of silica brick on the sides, which is covered by the rammed-up material, ganister plus a cement of fine silica sand bonded with linseed oil and water glass, that also forms the bottom. The arched roof is made up of a $4\frac{1}{2}$ -inch layer of clay-bonded carborundum brick, covered by $4\frac{1}{2}$ inches of fire brick. The center of the roof is not lagged above the fire brick, but the edges are covered with infusorial earth brick and cement.

A 15 by 15 inch charging opening is covered by a sliding door of fire brick in an iron frame, which is wedged up to tighten it. The pouring spout in the bottom of the charging opening is closed by a fire-brick plug. The furnace is kept as tightly closed while running as possible.

This furnace is used on jobbing work and it has seldom been necessary to force the furnace to its maximum production. Often the furnace has had to be held back while waiting for molds to be put up, and smaller charges than those on which it would show maximum efficiency have often been used. It has been operated only on one shift a day, usually about 10 hours, never on a 24-hour basis. The output has been normally from 1,500 to 2,500 pounds a day.

The first 117 heats averaged not far from 550 pounds each, or about 32 tons, and the total power used for them, including all pre-heating of the empty furnace, was 18,156 kw. h., or about 575 kw. h. per ton.

From the results obtained in regular practice, without pushing the furnace to its limit, it seems that if pushed the furnace can melt

on 10-hour operation about 3,500 pounds of red brass at about 475 kw. h. per ton. Though so small a furnace would probably not be used for 24-hour operation, it should give, if operated 24 hours a day, with no unnecessary delays, nearly 5 tons a day at about 380 kw. h. per ton. The electrode consumption, exclusive of breakage, was about $3\frac{1}{2}$ pounds per ton. Very few electrodes were broken when operating on red brass, but the breakage was high when melting an alloy of 20 per cent zinc. The carborundum roof appeared good for more work after 400 heats.

The furnace has been used on a few heats of Monel metal, turning out 500 pounds in four hours, at the rate of about 1,200 kw. h. per ton, and as there are no volatile metals in Monel metal the loss ranged from zero to one-tenth of 1 per cent. Very good Monel castings were made from metal melted in the furnace.

Pure copper has been melted in the furnace, starting hot, at the rate of 500 kw. h. per ton and with a loss of two-tenths of 1 per cent. On 16 heats of red brass, made from scraps, gates, ingot, and new metals, in which 8,669 pounds were melted, 8,469 pounds ingot or castings were recovered, giving a gross loss of 2.3 per cent, no deduction being made for metal recoverable from slag or for spillings. In the coke fires the gross loss on this material was said to be about 4 per cent.

On 26 heats of "half-yellow half-red"—that is, an alloy made up of half-red scrap, ingots, or gates and half-yellow ingots, the mixture containing 20 to 22 per cent zinc—13,689 pounds were melted and 13,347 $\frac{1}{2}$ pounds were obtained, giving a gross loss of 342 $\frac{1}{2}$, or 2.46 per cent. The gross loss in melting this material in coke fires was given as 4.5 per cent. On three heats from 1,700 pounds of all borings (20 per cent zinc) 1,621 pounds were obtained, a gross loss of 79 pounds, or 4.6 per cent. On two heats from 1,191 pounds of all-yellow ingot (about 36 to 40 per cent zinc) 1,071 pounds were obtained, the furnace being run at a low power input to reduce local overheating, a gross loss of 48 pounds, or 4 per cent. The loss on this in coke fires is given as 3.5 per cent.

On one heat of all-yellow borings 308 pounds were charged. The furnace could not be held tight, though the power input was kept down to 60 kw., and a flame 3 feet long shot from the furnace throughout the latter part of the heat. Only 153 pounds ingot could be poured from this heat, the rest of the metal being volatilized or coming out as powdery dross.

Even on the 20 per cent zinc alloy, melting half-red scrap and half-yellow ingots, there was much trouble from zinc vaporizing, condensing on the electrodes and making them stick in the furnace walls or water coolers, so that they could not be adjusted. In taking them out to clean them from the zinc deposit much time was lost, and in

18 consecutive heats on the 20 per cent zinc alloy nine electrodes were broken. The Gerline Co. states that the furnace does wonderful work on Monel metal, excellent work on copper and red brass, works satisfactorily, all things considered, on the 20 per cent zinc alloy, but the company would not recommend it "under any circumstances or to anyone" for yellow brass. On one heat of an alloy high in lead the last few castings from the last ladle were said to be nearly pure lead, but it was also stated that this trouble could be largely overcome by careful stirring. In general, the quality of the metal is just as good as that from pit fires, and on account of the high prices of crucibles in 1917 the Gerline Co. was of the opinion that the melting cost was decidedly low. Although this furnace was very satisfactory, it has not been operated recently because the central station supplying power has been too heavily loaded to carry even this small furnace.

OTHER RENNERFELT FURNACES IN USE.

A Rennerfelt furnace of one of the older types of about 600 to 800 capacity, 90 kw., was installed by Titanium Bronze Co. a couple of years ago, but its installation was delayed. The furnace was said to be not well arranged mechanically and melting practice at this plant became established in oil; the electric furnace was used only on a few trial heats and never thoroughly tested. Another Rennerfelt of 1,000 pounds capacity, as well as a Snyder of similar capacity, are installed at the laboratory of the Chile Exploration Co., New York City, but both are used for purely laboratory purposes on many materials, not for regular production.

Small Rennerfelts are also installed for laboratory or test purposes only at the Driver-Harris Co., Haynes Stellite Co., and General Chemical Co. The Haynes Stellite Co. says that its furnace is not considered very satisfactory.

A small Rennerfelt was in use at the Acieral Co.'s plant for making "hardeners" for use in aluminum alloys before that plant was closed, and that furnace has been installed for a similar purpose at the Bausch Machine Tool Co., Springfield, Mass.

A 1,000-pound Rennerfelt has been installed by the Hardite Metals Co., Long Island City, N. Y., for use on nickel-chromium alloys.^a

PERFORMANCE OF RENNERFELT FURNACE AT PHILADELPHIA MINT.

In July, 1917, a 1,000-pound, 125-kw., 110-volt Rennerfelt began operation at the Philadelphia Mint. The director of the mint^b has published some data on the operation of the furnace and has kindly

^a Cone, E. F., Status of the electric steel industry: Iron Age, vol. 107, 1921, p. 69.

^b Baker, R. T., Annual report of the director of the mint for the fiscal year ending June 30, 1918: Treas. Dept. Doc. No. 2622, p. 11.

furnished other information which is included below. The furnace usually ran 24 hours a day, and in some 50 weeks of regular operation produced over 1,300 tons of coinage alloys—cupronickel, bronze, and silver. The average power consumption per net ton of product in this period was 500 kw. h. for cupronickel, 300 for bronze, and 200 for silver, these figures including all operation, whether on one 8-hour shift or on 24-hour operation.

CUPRONICKEL.

On cupronickel, which is 75 per cent Cu and 25 per cent Ni, a typical 24-hour run shows 10 heats of 1,000 pounds, or 5 tons melted at 440 kw. h. per ton. On 8-hour operation, three heats, $1\frac{1}{2}$ tons can be made at 650 kw. h. per ton.

The net metal loss on cupronickel is one-fourth of 1 per cent. While there are no volatile metals in this alloy to be lost by vaporization, in order to get the best quality of metal it is essential, notwithstanding that the atmosphere of the furnace may generally be reducing, to keep the furnace tightly closed against the entrance of air.

It was thought that the heat loss and the oxidation due to double pouring made desirable rigging the furnace to pour direct into the bar molds, though a ladle with a "teapot" spout helped somewhat. It was decided that Rennerfelt furnaces to be installed thereafter at the mint should have automatic electrode control and should be built to tilt around the spout for direct pouring.

The cupronickel melts proved to be homogeneous without stirring, as in this alloy both components are mutually soluble and have no tendency to segregate.

The melting cost on cupronickel is given as \$9.45 per net ton, which is less by 50 per cent than the cost in gas-fired crucible furnaces.

COINAGE BRONZE.

On coinage bronze, which is 95 per cent Cu and 5 per cent Sn and Zn, about 4 per cent Sn and 1 per cent Zn, a typical 24-hour run shows 12 heats of 1,065 pounds each, or about $6\frac{1}{2}$ tons melted at 325 kw. h. per ton.

On 8-hour operation the output is about 3,500 pounds at 400 kw. h. per ton. The metal losses seem to run slightly higher than the 0.3 per cent loss regularly obtained in the mint crucible-furnace practice, going up to one-half or three-fourths of 1 per cent, perhaps owing to greater volatilization of zinc, but it is thought that with experience this can be kept down to the 0.3 per cent figure. Such a figure in either type of furnace would hardly be obtained in ordinary foundry practice; it was obtained at the mint because of the workmen's skill, gained by working on the precious metals, in saving every drop or

scrap of metal. The mint considers that the furnace would give an excessive loss of zinc on high-zinc alloys.

No trouble is experienced in getting a homogeneous product on coinage bronze, and even coinage silver (90 parts Ag and 10 parts Cu) has shown uniformity in samples taken from top and bottom of the melt. No segregation would be expected in the bronze alloy, though the handling of coinage silver to avoid segregation is more of a task.

The furnace roof and lining was silica brick; outside the 9-inch silica brick on the sides is 2½-inch fire brick, then 1-inch asbestos cement. The hearth is of ganister bonded with water glass. The life of the lining—1,277 heats for the furnace body and 800 to 1,000 heats for the roof—is amazingly good in view of the pouring temperatures of cupronickel, the material chiefly melted.

The long life of the silica roof is doubtless due, in part at least, to the furnace having been normally run 24 hours a day and never allowed to cool unless repairs were to be made.

On all of the alloys melted the mint figures a saving of 40 per cent in melting cost over former fuel-fired practice. The output per furnace tender was doubled over that from crucibles.

In a later report¹⁰ of the director of the mint further data on the operation of the 1,000-pound Rennerfelt are given. In melting silver coin the furnace handled 18,000 pounds, or 9 tons, in 24 hours. In melting 1,585 tons of silver coin the average power consumption was 177.4 kw. h. per ton, 24-hour operation, the cost for electric power per ton being over 50 per cent less than the corresponding cost of gas per ton for melting in crucible furnaces. The furnace did not require either relining or a new roof during the 3,191 heats required to melt the 1,585 tons of silver coin. The metal losses in melting silver coin were approximately one-tenth of an ounce per 1,000 ounces, one part in 10,000, or 0.01 per cent, a most remarkable figure and one lower than is obtained in crucible melting. The director of the mint says: "This is due to the fact that the furnace construction and the absence of flue prevent metal from going up the chimney, and, too, the method of pouring is cleaner and fewer mechanical losses are suffered."

After melting the run of silver coin the original 1,000-pound furnace was replaced by another having automatic electrode control for the side electrodes and tilting on the lip for direct pouring into molds. This furnace ran 286 heats of cupronickel, one heat of pure copper made to clean out traces of nickel, and 445 heats of coinage bronze, or a total of 732, when it was shut down until the installation

¹⁰ Baker, R. T., Annual report of the director of the mint for the fiscal year ending June 30, 1919: Treas. Dept. Doc., No. 2849, p. 12; see also Coleman, H. D., Melting in a Rennerfelt electric furnace: *Metal Ind.*, vol. 18, 1920, p. 406.

of another similar furnace of 1-ton capacity, as shown in Plate X, A (see p. 148), was complete. Another 1,000-pound furnace was ordered for the San Francisco Mint. The electrode consumption is $2\frac{1}{2}$ to 4 pounds per ton of metal melted, depending on the character of the charge. Though a week's run demonstrated that the lip-pouring device allowed pouring direct into molds with precision, it was concluded that ladle pouring gave better ingots and was more practical. A 200-pound ladle is put on a car, rolled under the spout, filled, and pulled to the molds, where it is lifted in a shank and hung in a bail on a hoist, the metal then being poured into molds resting on a turntable. Use of larger ingot molds, being poured simultaneously, and of larger ladles are to be tried to shorten the pouring time.

The makers of the Rennerfelt furnace, however, state that the casting table or device by which the molds are spotted in front of the spout did not work very well, retarded pouring too much, and kept the spout open too long. They consider nose tilting desirable if the furnace can be emptied very rapidly.

As the work at the mint demands the highest quality of product, this point, as well as the mint's conclusion that the furnace must be tightly closed to prevent oxidation and the resultant lowered quality of product, deserve close consideration by users of electric furnaces. The Gorham Mfg. Co., Providence, R. I., has installed a Rennerfelt for melting silver.

PERFORMANCE OF RENNERFELT FURNACES ON LEADED BEARING BRONZE.

Besides the Snyder furnaces, the Chicago Bearing Metal Co. operated two 300-kv. a., 1-ton capacity Rennerfelt furnaces on leaded bronze, 70 to 75 per cent Cu, 15 to 20 per cent Pb, about 6 per cent Sn, balance Zn, Sb, Fe, etc. The furnaces take current at 100 to 110 volts. In a test of these furnaces in 1917 in 130 heats the average power consumption was about 285 kw. h. per ton on 20-hour operation, and from the data full 24-hour operation should give a figure of about 270 kw. h. per ton. The output was lower than that from the Snyder furnaces, 10 to 11 tons being obtained a day. The rate of power input to the Snyder furnaces was considerably higher than that to the Rennerfelts. The metal losses seemed to be somewhat lower than in the Snyders.

The Rennerfelt furnaces had loosely fitting water-cooled bronze sleeves about the side electrodes at the entrance to the furnace, the side electrodes being 3-inch diameter graphite and the top one 4-inch diameter. Trouble was encountered from deposition within these sleeves of metallic zinc, or zinc oxide, and lead and antimony or their oxides. Then the electrodes would bind, and as the electrode supports and adjusting mechanism were not very rigid, a twisting stress was set up and the electrodes, especially the side or 3-inch electrodes,

broke easily. The actual electrode consumption ran at about $5\frac{1}{2}$ pounds per ton of metal melted, plus about 2 pounds per ton more lost by breakage. Damage to the water coolers was also common. Hence early in 1918 the use of the water coolers was abandoned and the electrodes passed through large holes, about 6 by 8 inches, in the furnace walls. This cut down the sticking of the electrodes, but made them oxidize more readily, the joints became weak, and the electrode consumption increased greatly, being nearly three times as much the first month after the change as it was before. In 57 heats watched by a representative of the bureau in 1918, two months after the coolers were eliminated, 33 electrode sections had to be added because of breakage, usually at the joint, which had been weakened by burning. The broken sections were not always a total loss, but had to be rethreaded into shorter sections.

The electrode consumption has since been reduced by using 4-inch electrodes on the sides as well as on the top, these electrodes being large enough to withstand the burning of the joints without becoming excessively fragile.

The Rennerfelt furnaces are lined with the same mixture of carborundum fire sand, fire clay, and molasses as the Snyders, and have fire-brick roofs. At present the hearth lasts about 400 heats of 1,500 pounds each and the roof 100 to 180 heats, on 8-hour, one-shift operation. While on 20-hour operation in 1918 the cost of materials, labor not included, for Rennerfelt roofs and linings, during the melting of 816 tons, three-months' output, two furnaces, averaged 60 cents per ton. The lining life at that time was not as good, on the basis of tons melted, as in the Snyder, whereas the later figures, given above, show it to be slightly better.

Used with large openings about the electrodes the Rennerfelt furnace can not be kept tight, and in this condition gives off as much fume as the Snyder. The metal losses in tests of Rennerfelts, operated open, were somewhat below those reported in tests of the Snyders; 18 heats of 1 ton each gave 35,043 pounds of metal and some 650 pounds slag, from which about 160 pounds metal would be recovered; the gross loss on 36,000 pounds was 957 pounds, or $2\frac{1}{2}$ per cent, and the net loss about 800 pounds, or $2\frac{1}{4}$ per cent. No account is here taken of dirt or other nonmetallic in the charge, but it was thought that the charge to the Rennerfelts in the 1918 tests was a little cleaner than that to the Snyders. Though the true difference between the net metal losses in the Snyders and the Rennerfelt, both operated unsealed and open at spout and electrodes, may not be in favor of the Rennerfelt quite as much as the test figures—4.5 per cent Snyder, 2.25 per cent Rennerfelt—it is believed that the Rennerfelt has a lower metal loss than the Snyder.

In the 1918 tests the Rennerfelts were giving only seven to eight 1-ton heats in 19 to 20 hours, while the Snyders were giving 13 heats of the same weight in the same time. The Snyder power consumption was only about 280 kw. h. per ton, whereas the Rennerfelts on 57 1-ton heats used 18,860 kw. h., or about 330 kw. h. per ton. On a full 24-hour schedule the Snyders seemed capable of melting at 265 kw. h. per ton and the Rennerfelts seemed capable of melting at 300 kw. h. per ton.

Several factors contribute to the lower output and higher power consumption of the Rennerfelts. The Snyders could be charged more rapidly on account of their suitability for mechanical charging, but the Rennerfelts have to be charged by hand. The use of small uncooled electrodes and the consequent loss of time from excess breakage handicapped the Rennerfelt, and in that furnace much heat was lost about the electrode openings. The greatest factor in the relative slowness of the Rennerfelts was, however, the low rate of power input, the Rennerfelts being run at about 200 kw. and the Snyders averaging 310 kw.

The power factor of the Rennerfelts, however, was satisfactory, being around 75 on the 60-cycle current used, whereas that of the Snyders is much lower.

Much better results as to rate of production were later obtained with the Rennerfelt furnaces, after the electrodes were all made 4 inches in diameter, and after adopting the plan of basing the pay of furnace tenders on a certain number of heats instead of paying by the hour. This plan has cut about two hours off the running time, the furnaces being operated one shift only during most of 1919, because the furnace tenders hold a much higher power input than before, take more care, and are less likely to break electrodes when responsible for the time lost in replacing them. One negro furnace tender takes care of one furnace.

Care is taken to pour the metal from the furnace as soon as it is ready, pouring taking 10 minutes. Charging is still done by hand, as the Rennerfelt furnace is not suited to mechanical charging. Two men charge in five minutes.

Eight 1,500-pound heats, or 6 tons, are now obtained in a trifle over eight hours. In 49 days one furnace made 394 heats, 296½ tons, in 395½ hours; the other made 390 heats, 292 tons, in 400½ hours, or a total of 588 tons in 796 hours, or about 1,500 pounds per hour. The power consumption in 8-hour operation, hand control, was about 320 kw. h. per ton. These figures show a lower output and a slightly higher power consumption than by the Snyder and are accounted for by the transformers at the Rennerfelt furnaces being rated at 300 kv. a., whereas those at the Snyders are rated at 700 kv. a. The Rennerfelt transformers overheat at a rate of power input to the

furnaces that will give one 1,500-pound heat, including pouring and charging, per hour at 320 kw. h. per ton, and the attempt to maintain so high a rate of production was deemed unsafe. If pouring and charging take 15 minutes, if the power is on 45 minutes per heat, and 240 kw. h. are used, then the average power input is 320 kw.; at 75 per cent power factor this input is over 425 kv. a. instead of the 300 kv. a. at which the transformers are rated. It was thought that if automatic electrode controls were installed on the Rennerfeldts one furnace tender could operate both furnaces.

Prices quoted on Rennerfelt furnaces, including transformers, meters, etc., in February, 1920, were as follows, for 60-cycle current:

TABLE 43.—*Prices of Rennerfelt furnaces.*

Type.	Capacity.	Kv. a.	Price.
O.....	300 to 500 pounds.....	100.....	\$5 380.
IA.....	1,000 pounds.....	150 to 200.....	\$12,000.
IIB.....	1½ tons.....	300 to 500.....	\$16,000 to \$20,000.
IVA.....	3 to 4 tons.....	800 to 1,200.....	\$26,000 to \$30,000.
VA.....	5 to 6 tons.....	1,400 to 1,800.....	\$35,000 to \$46,000.

Type O is hand controlled; all the others have automatic electrode control. At the prices quoted types O, IA, and IIB have central trunnions. Types IVA and VA are set on rockers. Types O and IA are tilted by hand, the rest by motor.

RE-PEL-ARC FURNACE, OR VON SCHLEGELL REPELLING-ARC FURNACE.

A recent development is the von Schlegell "repelling-arc" furnace,¹¹ manufactured by the Re-pel-arc Furnace Co., of Indianapolis, which can be operated either as a direct-arc or an indirect-arc furnace. Figure 29 shows a schematic diagram of the furnace. The furnace is normally three-phase (though single-phase current with two electrodes can be used) and is designed for 220 volts. The electrodes, instead of hanging vertically, are supported on pivots and supplied with counterweights, so that when no current is on the three electrodes hang at slight angles, touching at their tips. When the current is thrown on, the repulsive action due to the current flow moves the electrodes apart, thus starting the arc. Similarly, if the arc is broken while the current is on it will restart itself. The length of the arc, and hence the power input, is regulated by the position of the counterweights. The arc is therefore largely self-regulating.

¹¹ Von Schlegell, F., Electric furnace with repelling arc: *Iron Age*, vol. 106, 1920, p. 1556; *Metal Industry* (London), vol. 18, 1921, p. 23; U. S. Patent 1,352,541, Sept. 14, 1920.

If the cluster of electrodes is placed so far above the charge that the arc path from one electrode tip to another is shorter than that

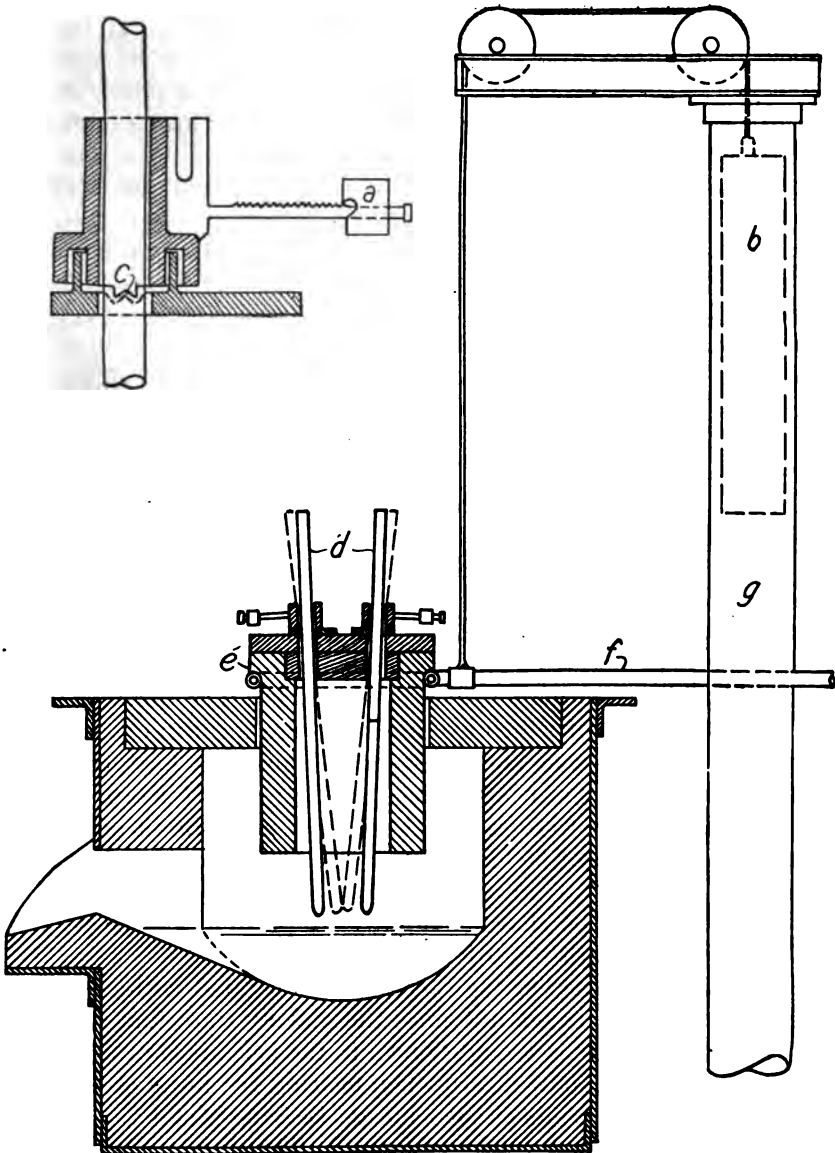


FIGURE 29.—Von Schlegell "repelling-arc" furnace: *a*, current-regulating weight; *b*, counter weight; *c*, pivoted electrode-holder showing gas-tight joints; *d*, electrodes; *e*, refractory sleeves; *f*, pipe support; *g*, mast. By raising torch arc is drawn between electrodes, instead of between electrodes and metal, thus moving center of temperature above metal without affecting operation of the arc.

from one tip to the bath and back to another tip, the arcs will be drawn among the electrodes, and the furnace will operate as an

indirect-arc furnace. If the electrode cluster be lowered so that the electrode tips are close to the bath, the arcs will be drawn to the path and the furnace will operate with a direct arc. If the electrodes burn off evenly, so that independent adjustment of each electrode can be dispensed with, this scheme may prove a simple method of direct-arc furnace operation. The electrode, of graphite, may be held above the charge to give an indirect arc while the charge is melting and be lowered to give a direct arc when the charge is melted. The electrode cluster is termed by its inventor "an electric torch," since the arc is thrown down from the electrode tips something after the fashion of the Rennerfelt fleur-de-lis arc, and it is pointed out that this torch contains all the electrical parts of the furnace, the balance being merely the hearth and shell. By placing two shells side by side, as shown in Plate XIII, *A* and *B*, one furnace can be heated while the other is being poured and recharged. A gas or oil flame could be used to preheat one furnace charge while the other is being electrically heated.

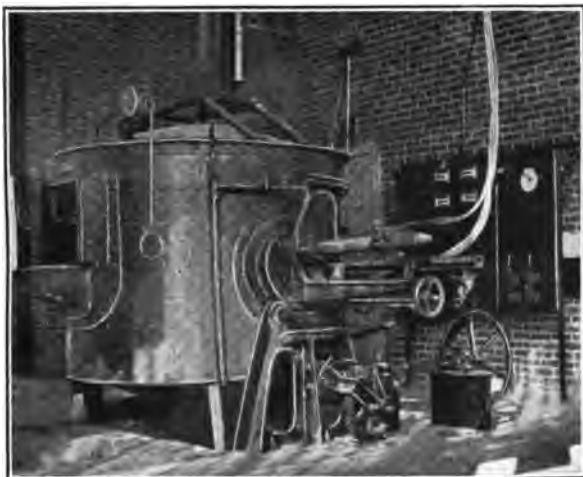
On red brass, 85:5:5:5, half heavy scrap, half borings, ~~etc~~ heats of 300 pounds were made in a day. The furnace was preheated half an hour before charging. Three hundred and seventy-five kw. h. were used, including the preheat, and 2,112 pounds of metal melted; that is, the furnace ran at the rate of 357 kw. h. per ton. Electrode loss was about 3 pounds per ton. No data are given on the life of the "sleeve" about the electrode. The time for a heat averaged one hour, of which five minutes were spent in charging and pouring. It takes one and one-fourth hours per heat on Monel metal. In another run on red brass half ingot, half scrap, 11 heats of 300 pounds each were said to have been made in 10 hours, at a power consumption of 350 kw. h. per ton, and a metal loss of 1 per cent.

The furnace can be charged rapidly by pulling out the "torch" and charging through the top of the furnace.

It is not stated whether the furnace was operated with a direct or indirect arc on red brass, but since the makers state that on Monel metal or steel the arc is brought down to the bath, it is probable that it is operated as an indirect arc when melting red brass or bronze.

One of the Rennerfelts at the Chicago Bearing Metal Co. has been changed over to take a 450-kw. "torch." This furnace is shown in Plate XII, *B*. The metallurgical behavior of the furnace may be expected to be that of any direct-arc furnace when used as a direct-arc furnace and of any indirect-arc furnace when used as an indirect-arc furnace. It could be used on bronze and Monel metal but not on yellow brass high in zinc.

All the yellow brass and manganese bronze made by the Chicago Bearing Metal Co. is still melted in crucible lift-out oil furnaces, the



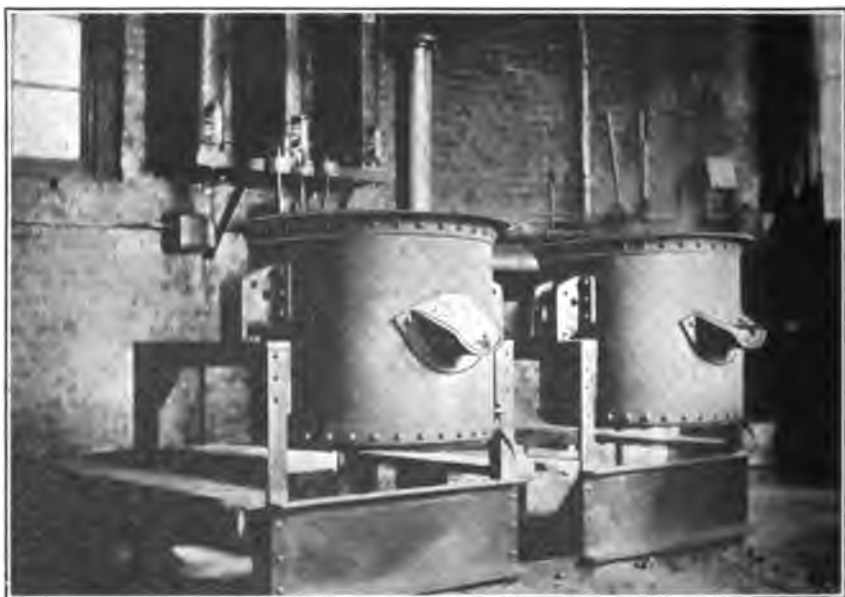
A. TRUNNION-TYPE RENNERFELT FURNACE.



B. RE-PEL-ARC ELECTRIC TORCH IN AN OLD RENNERFELT SHELL AT THE PLANT OF THE CHICAGO BEARING METAL CO



A. REPELLING-ARC FURNACE WITH TWO SHELLS, SIDE VIEW.



B. REPELLING-ARC FURNACE WITH TWO SHELLS, FRONT VIEW.

various direct and indirect arc furnaces not being of any more use for alloys high in zinc than the open-flame oil furnaces would be.

A repelling-arc furnace is being tried out by the Bridgeport Brass Co., but it is stated ¹² that it has not yet been very successful.

SUMMARY OF STATIONARY INDIRECT-ARC FURNACES.

It has been shown that although stationary indirect-arc furnaces, of which the Rennerfelt is the best known, have been used on yellow brass by running at low power input, where excessive zinc loss was a minor factor because of electric power being cheaper than fuel, yet under American conditions it would be wasteful to use such furnaces for melting alloys high in zinc. Even in the United States an alloy of about 20 per cent zinc has been commercially melted in a 600-pound furnace with a metal loss slightly less than in former fuel-fired practice, but the furnace has given trouble through the electrodes sticking from condensed zinc. By analogy with the behavior of the direct-arc furnace, the 600-pound size of which could be kept tight when melting a high lead alloy, though the 1-ton size could not, it is doubtful whether a 1-ton indirect-arc furnace can be satisfactorily operated on even the 20 per cent zinc alloy at a rate of power input that will give good figures for production and power. The 1-ton indirect-arc furnace even gave trouble when operated tightly closed on a 15 to 20 per cent lead alloy and finally had to be operated open with correspondingly larger metal losses.

Stationary indirect-arc furnaces are highly suitable for Monel metal, cupronickel, pure copper, silver, bronze, red brass, and similar alloys containing up to, say, 10 per cent zinc or 15 per cent lead. On alloys high in volatile metals this type of furnace operates at a metallurgical disadvantage, which increases with the amount of volatile material; hence operation on yellow brass is entirely impractical under American conditions, a fact that is understood and admitted by the makers of the Rennerfelt. Though the furnace has a wide field—a much wider one than that of the direct-arc type—it is not useful in the important field of rolling-mill work on alloys high in zinc.

The indirect-arc furnace ranks high as regards reliability, reasonable upkeep, and rate of production. Under similar conditions its rate of production is not quite as large as that of the direct-arc type, nor is its thermal efficiency quite as good, but in both these respects it is decidedly better than furnaces of the reflected-heat type, such as the granular resistor and smothered arc. Its field is smaller, however, than that of those types. Its electrical characteristics are satisfactory. The load is not so steady nor its power factor so high as those of the reflected-heat type, but both are better than in the

¹² Webster, W. B., personal communication, Aug. 24, 1921.

direct-arc type. The Rennerfelt takes three-phase current, and hence is free from the drawbacks of a single-phase load. All in all, the stationary indirect-arc furnace is a very useful type, provided it is used on work for which it is fitted.

MOVING INDIRECT-ARC FURNACES.

STASSANO FURNACE.

Though the earliest indirect-arc furnaces suggested were those of Pinchon (1853)^{12a} and Siemens (1878), followed by that of Moissan, used to such good advantage in the classical researches of that great Frenchman, the first commercial indirect-arc furnace was invented by Stassano. Stassano first designed (in 1898) a shaft furnace for making steel direct from iron ore; then he designed a stationary hearth furnace, but his first furnaces to attain commercial success were designed, as illustrated by figure 30, to be kept in motion.¹³

These furnaces were built in the form of an upright cylinder with its vertical axis at a slight angle from the vertical. The whole furnace was rotated, so that both the rotary motion and the tilt of the axis tended to swash the metal from side to side.

Stassano says¹⁴ that the "agitation increases the activity of the reaction producing the refining, and shortens the time, thus making possible work on a full charge without risk of overheating the chamber or damaging the lining." Stassano's prime object in rotation thus seems to have been assuring a better contact between the ore and reducing agent in smelting iron ore, or between the metal and slag in refining steel. Of course, the continuous rotation of the furnace body involved the use of sliding contacts in the electric circuits and of joints in the water-cooling connections. Such contacts in circuits carrying the high amperage required by good-sized electric furnaces are notoriously difficult to maintain and such joints in water connections are hard to keep tight. Stassano¹⁵ therefore had to substitute an oscillating motion and abandon complete rotation of the furnace in order to overcome these difficulties.

Rodenhauser¹⁶ says: "If any security is desired for a complete uniformity of the material in its several layers, it is not possible to dispense with the mechanical bath circulation."

^{12a} See Rodenhauser, W., and Schoenawa, J., translated by vom Baur, C. H., *Electric furnaces in the iron and steel industry*; 1917, p. 4.

¹³ Steel production in the electric furnace: *Electrochem. Ind.*, vol. 1, 1903, p. 247; Stassano, E., U. S. Patent 799,105, Sept. 12, 1905; application Apr. 9, 1902.

¹⁴ Stassano, E., The application of the electric furnace to siderurgy: *Trans. Am. Electrochem. Soc.*, vol. 15, 1909, p. 70.

¹⁵ Stassano, E., U. S. Patents 1,024,657, Apr. 30, 1912, application Aug. 7, 1911; 1,059,499, Apr. 22, 1913, application Nov. 29, 1912, and 1,105,859, Aug. 4, 1914, application Oct. 15, 1913.

¹⁶ Rodenhauser, W., and Schoenawa, J., translated by vom Baur, C. H., *Electric furnaces in the iron and steel industry*; 1917, p. 118.

Schmelz¹⁷ says that an oscillating furnace with the double-trunnion suspension, like that of a ship's compass, does away with the former complications of the cable and water connections of the rotary

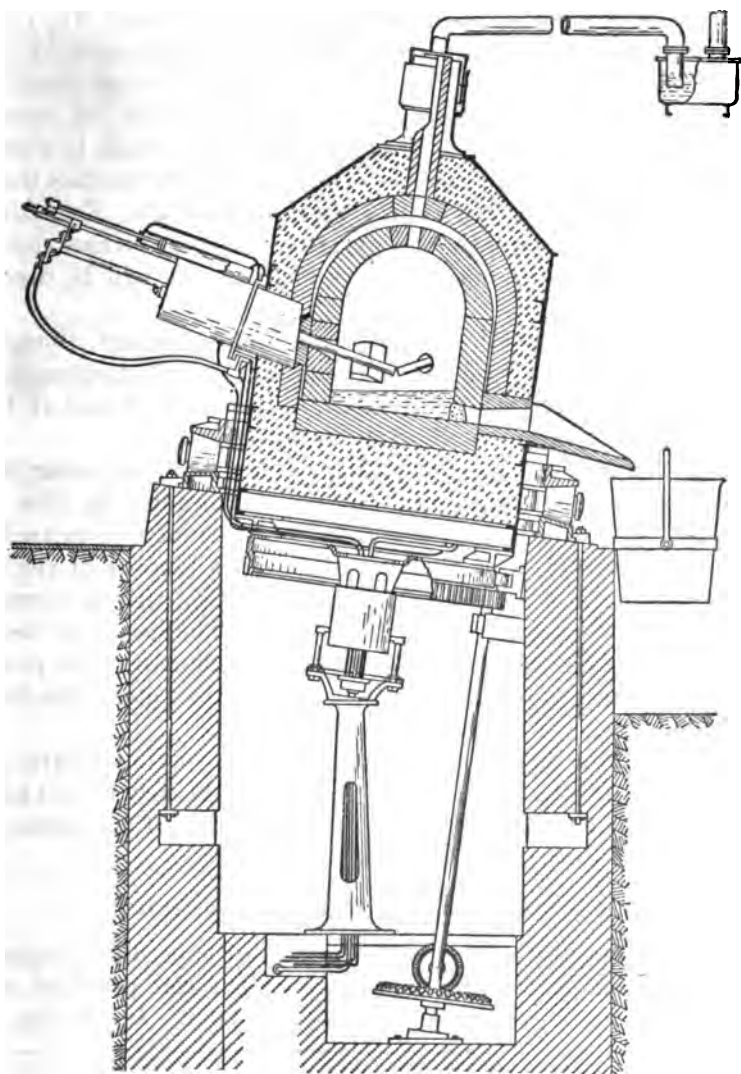


FIGURE 30.—Elevation of gyrating three-phase Stassano furnace.

furnace. It can also be tilted to pour, while the earlier form had to be tapped. The oscillating or gyrating design of the Stassano is very like the later design of the Thomson zigzag-resistor furnace.

¹⁷ Schmelz, E. M., A new electric steel-casting plant: Met. and Chem. Eng., vol 11, 1913, p. 709.

Keeney¹⁸ asserts that agitating the bath of steel had not proved of great value for steel and had been abandoned in the more recent Stassano furnaces.

The moving Stassano steel furnace, as far as careful study of the literature reveals, has never been tried or even suggested by its inventor for use on brass. Though the Stassano motion would tend to mix the metal somewhat, it is doubtful whether the relatively slight motion would mix the charge thoroughly enough to overcome in melting alloys high in zinc the overheating of the surface through the concentrated radiation from a high-powered arc. One furnace, the Volta, which is essentially a gyrating Stassano furnace, has very recently been brought out for brass melting. It will be described later.

Other moving electric furnaces have been suggested, though not especially, save in the case of the Thomson, for brass melting, among them those of Burton¹⁹, of Johnson²⁰, of Ischewsky²¹, and of Thomson.²² These furnaces are all of the resistor type.

Another rotating resistance furnace, designed among other things, for melting copper alloys, was described by Braun²³ in 1904. The resistance was embedded in the walls, and current was carried to it by sliding contacts, from the leads, through a number of electrodes also embedded in the walls. The refractory hearth was corrugated in order to secure agitation of the charge. The design of the furnace, as might be expected from a design of 1904, was not practical and as nothing has been heard of it beyond the patent, the furnace probably was never operated.

Rideal²⁴ states that F. Clerque designed a revolving electric radiation furnace for the manufacture of ferronickel that is said to have been in use at Essen, Germany. No description of the furnace nor reference to a description is given.

BENSEL RESISTOR ROLLING FURNACE.

Bensel²⁵ describes a laboratory furnace of spheroidal shape with the longer axis horizontal. A charging door was provided on the circumference and the furnace was tilted to pour. It was also rolled

¹⁸ Lyon, D. A., and Keeney, R. M., *Electric furnaces for making iron and steel*: Bull. 67, Bureau of Mines, 1916, p. 82.

¹⁹ Burton, G. D., U. S. Patent 679,452, July 30, 1901.

²⁰ Johnson, W. M., U. S. Patent 825,058, July 3, 1906.

²¹ Ischewsky, B. M., U. S. Patent 847,003, May 12, 1907.

²² Thomson, J., and Fitzgerald, F. A. J., U. S. Patent 950,878, Mar. 1, 1910; Thomson, J., U. S. Patent 950,880, Mar. 1, 1910.

²³ Braun, I., French Patent 339,942 of June 22, 1904.

²⁴ Rideal, E. K., *Electrometallurgy*, 1919, p. 150.

²⁵ Bensel, F., *Versuche zur Verminderung der Metallverluste beim Messingschmelzen*: *Metallurgie*, vol. 9, 1912, p. 533; *Chem. Abs.*, vol. 6, 1912, p. 3256; *Jour. Soc. Chem. Ind.*, vol. 31, 1912, p. 879; *Jour. Inst. Metals (British)*, vol. 8, 1912, p. 353; *Jour. Franklin Inst.*, vol. 175, 1913, p. 150.

back and forth while operating to mix the metal. A direct current was tried as the source of heat, but was said to be unsuitable as the positive electrode burned off too quickly. A resistor was therefore used, the carbon-rod resistor running through the long axis. The furnace held 20 kg., or 44 pounds, of metal and took an average of 44 volts at 530 amperes when hot. Bensel melted 44 pounds of a 70:30 brass made from new metals in 1 hour and 35 minutes, from a cold start, using 35.4 kw. h. He poured 19.3 kg., or 42½ pounds, of brass, a loss of 3.5 per cent, but he figured that much soaked into the lining and that the true loss was under 1 per cent.

By a peculiar process of calculation involving several assumptions he decided, on the basis of a single heat, that the furnace on an all-day run would melt at 7.3 kw. h. per 100 pounds, or 146 kw. h. per ton, a figure lower than the generally accepted value for the power theoretically needed at 100 per cent efficiency.

No further development of the Bensel furnace has been recorded.

CARRERÈ FURNACE.

Carrerè,²⁶ however, described an indirect-arc furnace having the form of a cylinder lying on its side, the electrodes entering through the ends. The body of the cylinder was rotated while the end walls were stationary, they and the electrodes not revolving with the furnace. This furnace was not suggested for use on brass.

WEEKS ZINC FURNACE.

The Weeks zinc furnace²⁷ is a horizontal cylinder heated by an indirect arc or arcs, but differs from the Carrerè furnace in that part of the end walls can move with the furnace body; only the electrodes and rings or trunnions about them that contain water coolers for the electrodes and an offtake for zinc vapor remain stationary; the furnace aside from these parts may revolve. A tap hole or pouring spout is shown in the patent, closed by a plug which is held in tightly to withstand the pressure of the molten zinc.

In both the Carrerè and Weeks furnaces the stationary ends and the revolving body involve many difficulties in maintaining a gas-tight seal over so large a joint and in sealing the charging door to prevent leakage of metal when the furnace is revolved. The charging door of the Weeks, on account of lack of space at the ends above the melt, was placed on the circumference of the drum.

No account of any actual experiments with the Carrerè furnace has been found in the literature. The Weeks zinc furnace, as has been previously stated, was tested at the plant of the General Electric Co. at Schenectady, N. Y., for zinc smelting; also one or two

²⁶ Carrerè, J. M., U. S. Patent 726,860, May 5, 1903.

²⁷ Weeks, C. A., U. S. Patent 949,511, Feb. 15, 1910; application filed Sept. 3, 1908.

heats of copper were made in it. The patent specifications for the Weeks furnace make mention of revolving the body of the furnace to interchange the roof and hearth, the advantage being in "uniformly wearing the lining and applying substantially all the heat to the contents." But in a published description of the furnace Weeks²⁸ makes no mention of revolving the furnace or of the effect of stirring on metal losses. He assumes a figure for metal losses on yellow brass, but it is not based on any experiments.

Hansen,²⁹ in describing the Weeks zinc furnace, says:

The drum was mounted on rollers which could be motor-driven. . . . The heat was supplied by radiation from an arc and the rotation of the furnace body was intended to equalize temperature of the furnace lining by periodically bringing all the parts of it in contact with the charge. Incidentally, thorough mixture of the charge would be secured by this rotation. The copper-melting experiments were of purely secondary consideration to Mr. Weeks. . . .

Hansen asserts that in melting zinc with this furnace the electrodes become soldered in place by condensed zinc and could not be adjusted to regulate the arc. Mr. Weeks has informed one of the writers that the furnace, though designed to rotate, was not rotated in the copper-melting experiments. No experiments were made with brass, and the effect of stirring on the superheating of the surface of the melt and on the loss of zinc was evidently not impressed on Hansen by the few experiments that were made; instead of continuing to experiment with the Weeks furnace Hansen built the three-phase Stassano-type furnace, which was tested on yellow brass borings at the Schenectady plant.

IMBERT ZINC FURNACE.

Another zinc furnace, the Imbert,³⁰ designed at about the same time as the Weeks, the patent application being made in Germany only two days before that of the Weeks in this country, greatly resembles the Weeks furnace; the chief variation is the leaving of a larger clearance space between the stationary part of the end wall and the revolving part, and the use of this space as offtake for zinc vapor. The Imbert furnace has not been suggested for use on brass.

HULDT FURNACE.

The Weeks and Imbert furnaces were designed for the reduction of zinc from its ores or for its purification by distillation. A more recent patent³¹ utilizes an indirect-arc furnace built as a horizontal

²⁸ Weeks, C. A., *Melting nonferrous metals in an electric furnace*: Met. and Chem. Eng., vol. 9, 1911, p. 363.

²⁹ Hansen, C. A., *Electric melting of copper and brass*: Trans. Am. Inst. Metals, vol. 6, 1912, pp. 110, 126.

³⁰ Imbert, A. H., German Patent 219,515 of Mar. 1, 1910; application filed Sept. 1, 1908.

³¹ Huldt, S., U. S. Patent 1,266,808, May 21, 1908, application filed Apr. 3, 1917. Norsk Elektrisk Metalindustri Aktieselskab; British Patent 106,558 of Mar. 21, 1918 (convention date, Sweden, Apr. 8, 1916).

cylinder, not for distillation of zinc, but for the melting of "blue powder," the oxide film on the surface of the particles being rubbed off by the motion of the particles sliding on each other in a rotating furnace. It is said that the drum may have a rocking motion if desired. The furnace construction is not explained in detail in this patent.

ROCKING OF FUEL-FIRED BRASS FURNACES.

Rotary fuel-fired furnaces, such as the cement kiln, and rotary roasting furnaces²² are well known. In a number of brass foundries it has been found that time could be saved in the operation of open-flame oil furnaces by rocking the furnace back and forth by hand during the last few minutes of the heat, so that the hot walls just above the metal are washed by the metal, and give up some of their stored heat to it. Mechanically rotating and rocking oil-fired brass furnaces are now on the market.²³

Rennerfelt²⁴ says that some "of the heat absorbed by the walls may be recovered by tilting the crucible, or the hearth, to bring the charge into contact with the heated walls, thereby absorbing some of the heat."

Rennerfelt also mentions the possible use of a projection in the furnace bottom to stir the metal when tilting the furnace. No provision is made for continuous tilting, nor, so far as known, has any Rennerfelt furnace been built with such a projection.

While moving electric furnaces of various types had been known, they had not been tried for brass melting save for a couple of heats in the Bensei resistor furnace, and though fuel-fired brass furnaces had been operated with slight stirring they had not been so operated mechanically. As far as can be ascertained, the Bureau of Mines was the first to make actual experiments on brass melting in an electric-arc furnace rocked or otherwise moved to stir the metal in order to prevent surface overheating of the melt; the first laboratory heat was made on August 25, 1915, in the electric furnace laboratory of Cornell University, which by a cooperative agreement is used by the Ithaca field office of the bureau.

The fundamental idea behind the bureau's study of the rocking indirect-arc furnace for brass was that several of the essentials of an "ideal" electric brass furnace—ruggedness, simplicity in the hot zone, and ability to be run at a high rate of power input—were present in the indirect-arc type, and that the drawbacks—superheating

²² Canby, R. C., *Trans. Am. Electrochem. Soc.*, vol. 27, 1915, p. 65; Anonymous, New foundry for making mill brasses: *Iron Age*, vol. 107, 1921; Anonymous, Makes asset of a building obstacle: *Foundry*, vol. 49, 1921, pp. 29, 197.

²³ Compare *Metal Ind.*, vol. 16, 1917, p. 38; Jones, A., U. S. Patent 1,384,459; Advertisement, *Metal Ind.*, vol. 18, 1920, p. 3; Advertisement, *Foundry*, vol. 48, 1920, p. 102.

²⁴ Rennerfelt, L., U. S. Patent 1,076,518, Oct. 21, 1913, p. 2, line 50.

of the surface of the charge, consequent inability to keep the furnace tight on alloys high in zinc, too much heat storage to give perfect control of temperature, thermal losses through the roof and consequent troubles with refractories in the roof—would either be avoided or decreased by a stirring action brought about by rocking.

Inasmuch as the ordinary stationary indirect-arc furnace gives far less trouble from surface superheating than the direct-arc type, it was evident that the overcoming of this trouble by stirring would be easier and simpler in the indirect than in the direct arc type. Indeed, it is very doubtful whether any commercially attainable degree of stirring could sufficiently counteract the overheating of the metal directly under the arc in a direct-arc furnace.

LABORATORY ROCKING FURNACE.

The small laboratory furnace, a fire-brick-lined cylinder lying horizontally, with 2-inch diameter graphite electrodes entering at the ends of the furnace, was so mounted that it could be rocked back and forth while melting, as in Plate XIV. The single-phase indirect-arc ran at 50 to 75 volts and took about 30 to 35 kw. If the furnace was operated on alloys high in zinc without rocking, it acted like any other stationary indirect-arc furnace, being hard to keep tight, superheating the surface of the metal and giving too high metal losses. By rocking the furnace back and forth, however, the furnace operated well, surface overheating was avoided, and the metal losses were low.

Some of the results on metal losses are tabulated in Table 44.

TABLE 44.—*Metal losses in Bureau of Mines laboratory rocking furnace.*

Alloy.				Material.	Pouring temperature.	Net loss.
Copper.	Zinc.	Tin.	Lead.			
65.6	34.4			Ingot.....	* C. 1,080	Per ct. 01.06
58.0	40.0	0.3	(b)	Chips.....	1,000	03.00
71.5	25.0	.5	3.0	do.....	1,125	1.60
71.5	25.0	.5	3.0	Ingot.....	1,150	.50
83.0	10.0	3.0	4.0	Chips.....	1,200	1.00
83.0	10.0	3.0	4.0	Scrap and gates...	1,225	.50
81.5	8.5	4.0	6.0	Ingot.....	1,200	.20

* The net loss on this alloy when the furnace was not rocked was 6.45 per cent, including some soakage into the fresh hearth.

^b Al, 0.4 per cent; Fe, 1.5 per cent; Mn, trace (manganese bronze).

* The net loss on the same chips in an oil-fired furnace, crucible practice, was 7.2 per cent, the gross loss 7 per cent in the rocking furnace.

POWER CONSUMPTION.

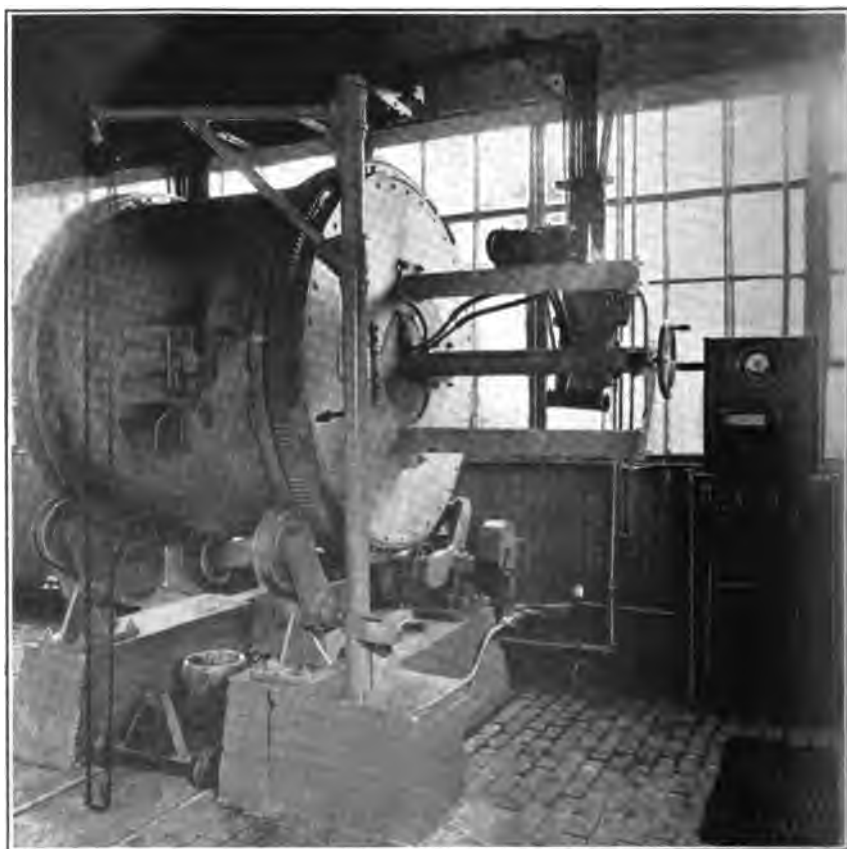
The furnace, working on 125 to 130 pound charges of red brass, poured at 1,200° C., melted five heats, 640 pounds, in five hours, including 1 hour and 10 minutes used in charging and pouring—start-



A. LABORATORY ROCKING FURNACE, SIDE VIEW.



B. LABORATORY ROCKING FURNACE, END VIEW.



4. ONE-TON DETROIT ROCKING FURNACE WITH OLD-STYLE SUPPORTS.

ing with the furnace cold—by using 136½ kw. h., or at the rate of 430 kw. h. per ton.

On 10-hour operation it should give 10 heats, 1,250 pounds, at the rate of 375 kw. h. per ton, and on 24-hour operation 24 heats, nearly 3 tons, at less than 325 kw. h. per ton. The electrode consumption was at the rate of about 8 pounds per ton.

BUREAU OF MINES EXPERIMENTAL 1,300-POUND ROCKING FURNACE.

The laboratory experiments on a furnace of 125 pounds capacity having indicated that such a furnace might be of value, the cooperation of the Detroit Edison Co. and the Michigan Smelting and Refining Co. was enlisted, and tests on a 1,300-pound furnace were made by the Bureau of Mines at the plant of the latter firm. The first heat was made on May 9, 1917.

The experimental furnaces and the results of the experimental work have been fully described elsewhere³⁵ and need be but briefly summarized here.

The furnace (see Pl. XIV) held 1,300 pounds and was run at 100 to 200 kw., averaging about 165. The arc ran at about 110 to 120 volts, and the furnace had a power factor of about 85. The electrodes were 4-inch diameter, graphite.

The inner lining of the furnace was of corundite brick, back of which was a special heat-insulating brick, with infusorial earth brick next the shell. Handwheels were provided for electrode adjustment, and the rocking motion was automatically controlled by a reversing motor, contactors, and suitable control device.

The furnace was kept stationary after a heat was started until some of the charge had begun to melt; then it was rocked through a small angle, or "safe rock," because by being rocked too far the solid part of the charge would fall on the electrodes and break them. The large furnace required much more care in this respect than the laboratory furnace, as the latter could be rocked through a large angle early in the heat, especially if the material melted was borings or other fine scrap not heavy enough to break the electrodes; in the large furnace even that kind of material would mat together into heavy chunks and would break the electrodes if the furnace were rocked too far too early in the heat. As the charge melted the angle of rocking was gradually increased until by the time that all was melted only the charging door and pouring spout were unwashed

³⁵ Gillett, H. W., and Rhoads, A. E., Melting brass in a rocking electric furnace: Bull. 171, Bureau of Mines, July, 1918, 181 pp.; A rocking electric brass furnace: Jour. Ind. Eng. Chem., vol. 10, 1918, p. 459; Met. Chem. Eng., vol. 18, 1918, p. 583; Met. Ind. (N. Y.), vol. 16, 1918, p. 265,355; Met. Ind. (London), vol. 18, 1918, pp. 102, 116, 132; Foundry, vol. 46, 1918, p. 814; Brass World, vol. 14, 1918, p. 217; Amer. Machinist, vol. 48, 1918, p. 1013.

by the metal. Figure 31 shows the stages in the rocking of a heat. The furnace was set off the horizontal to induce an end-to-end

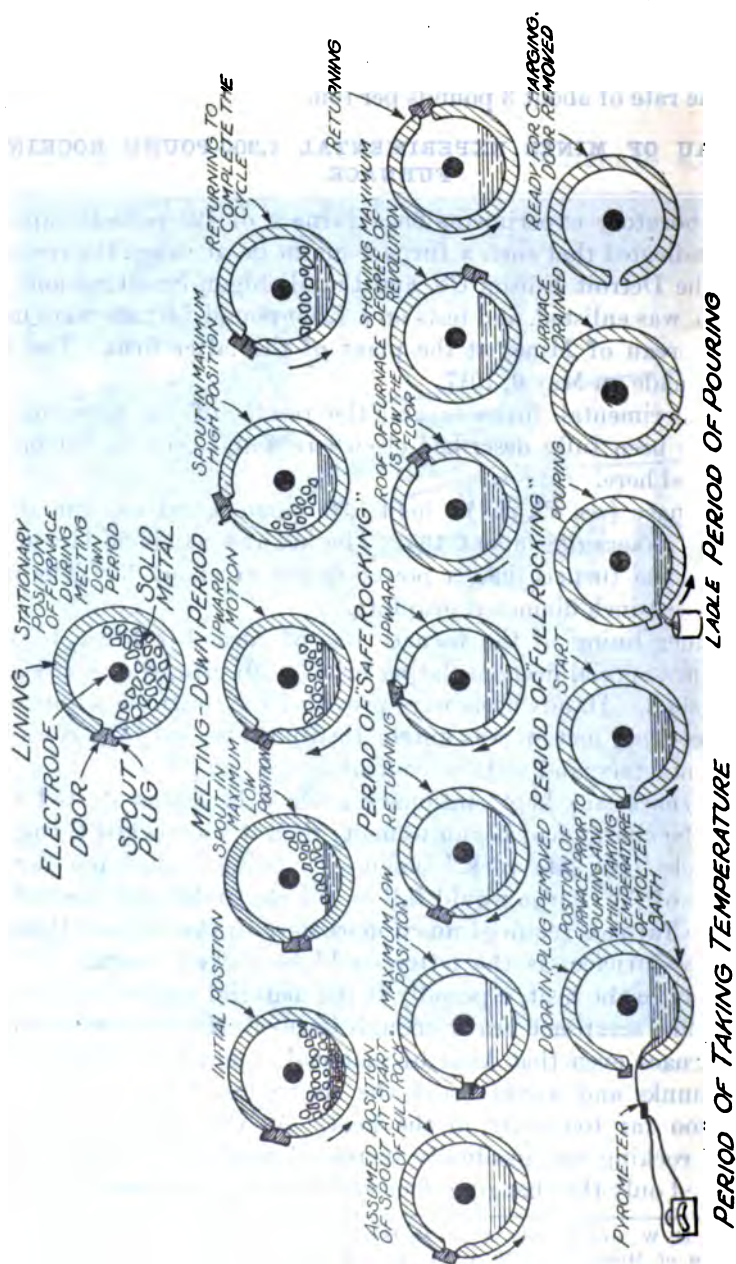


FIGURE 31.—Stages in the progress of a heat in a rocking furnace.

motion of the metal, as well as a rotary motion, and give more complete stirring.

As the work of the plant where the experimental furnace was set up was the making of ingot of standard composition from scrap charges, the true metal loss was not ascertainable, because of varying amounts of dirt, oil, and other nonmetallic material in the charge. The coke-fired crucibles running on exactly the same charge gave figures with which the electric furnace losses could be compared.

COMPARATIVE METAL LOSSES.

On 102 tons of metal melted in the rocking furnace in strict comparison with the same material charged to the coke fires, the alloys melted running from 90 to 66 per cent copper, 1 to 9 per cent tin, $1\frac{1}{2}$ to $26\frac{1}{2}$ per cent lead, 0 to 30 per cent zinc, the rocking furnace gave 3,626 pounds, or 1.8 per cent more ingot than the coke fires did from the same amount of the same material. Comparative figures on some specific alloys follow in Table 45.

TABLE 45.—Comparative metal losses in Bureau of Mines 1,300-pound rocking furnace and in coke fires.

Composition.				Weight charged.	Loss (metal, oil, and dirt), coke fires.	Loss (metal, oil, and dirt), electric.	Saved by electric.
Cu.	Sn.	Pb.	Zn.				
Per ct.	Per ct.	Per ct.	Per ct.	Pounds.	Per ct.	Per ct.	Per ct.
85	5	5	5	6,576	4.6	3.2	1.4
84	7	8	1	11,600	7.0	3.7	3.3
84	6	10	1	14,300	2.4	1.8	.6
79	9	10	2	11,790	3.6	2.1	1.4
78	2	10	10	11,840	7.1	3.1	4.0
76	8	13	3	11,805	4.0	2.4	1.6
73	4	20	3	14,392	3.7	2.9	.8
67 $\frac{1}{2}$	4	26 $\frac{1}{2}$	2	5,224	3.0	2.4	.6
67	1	2	30	7,200	8.0	5.1	2.9

In melting 75,805 pounds, which contained 266 pounds, or 0.35 per cent oil of leaded bearing bronze, 73,990 pounds good ingot, 106 pounds small ingot or spillings, and 795 pounds metallic in skimmings were obtained, giving on the oil-free basis 1.8 per cent gross loss, 0.6 per cent net loss.

In operating both the laboratory and the 1,300-pound furnace care was taken to keep the furnace door and spout closed and plugged as tightly as possible. The electric furnace gave alloys which analyzed remarkably close to the composition desired. On alloys high in zinc it was especially noticeable that the analyses of the produced showed a much smaller loss of zinc than in the coke fires. For example, a charge aimed to give 67 per cent copper, 30 per cent zinc, and 3 per cent tin and lead and divided between coke fires and the electric furnace gave, on melting in the coke-fired crucibles, an alloy of 68.4 per cent copper and 29.3 per cent zinc; in the electric furnace it gave one of 66.6 per cent copper and 30.4 per cent zinc.

The metal was found to be thoroughly mixed. Analyses of the first ingot from the first ladle and the last ingot from the last ladle of 1,200 pounds heats of an alloy in which 60 per cent copper, 37 per cent zinc, and 3 per cent lead was aimed at, giving the following results:

Heat No.	Per cent copper.	
	First ingot, first ladle.	Last ingot, last ladle.
322.....	59.76	59.54
323.....	59.78	59.66

An alloy containing 26 per cent lead was handled with satisfactory results as to freedom from segregation, the metal being more uniform than that from the coke fires.

The metallurgical results were excellent and the reliability and freedom from troubles with refractories appeared good, though the experiments were not continued long enough to get adequate data on the lining life under operating conditions.

RATE OF PRODUCTION AND THERMAL EFFICIENCY.

On 10-hour operation, on red brass poured at 1,150° C., the furnace gave five to six heats, 6,500 to 7,800 pounds, and on a 5-day run, in which 17.2 tons were poured, 5,750 kw. h. were used, or 335 kw. h. per ton.

On 24-hour operation, in a 4-day run, the furnace gave up to 14 heats of 1,300 pounds, or 9 tons per day; 34½ tons were melted in the run, with the use of 8,865 kw. h., or 257 kw. h. per ton. In both these tests the electrode consumption was slightly under 2 pounds per ton. In all these tests the furnace was somewhat handicapped by delays outside the furnace, such as waiting for the crane or for helpers for pouring. Without delays it would be possible to melt 18 heats of 1,350 pounds, or 10 tons, in 24 hours in this size furnace.

Some delay was experienced whenever the charge contained a large proportion of very oily borings, as the vapor is a nonconductor and it is difficult to hold the arc in such an atmosphere. It was therefore necessary to leave the spout unplugged and to wait till the oil had distilled out before starting the arc. The delay could be shortened by charging the borings first so that the oil would be coming off while the rest of the charge was being added, up to, say, 30 per cent oily borings being thus handled with little delay. With larger proportions it was better to leave the door open, charge the borings, and rock the furnace through a small angle to hasten the heating of the borings by the hot walls, and to distill the oil off faster.

In the earlier experiments some difficulty was found in the sticking of the electrodes when melting alloys high in zinc, owing to the condensation of zinc about them. The trouble was not like that found in the unrocked Stassano and Rennerfelt furnaces on similar alloys, and there was no trouble such as with the Rennerfelts on alloys high in lead, even when the alloys contained 26 per cent lead.

Experience in furnace operation showed how to obviate the trouble; either by making the holes through which the electrodes pass sufficiently smooth, as by the use of graphite rings or by using a suitable granular packing about them, and by allowing the electrodes to become a little hotter, reducing the amount of cooling water used, the difficulty was prevented entirely on 24-hour operation.

Similar precautions, coupled with careful cleaning of the electrodes from any deposit at the end of the day, made it possible to operate satisfactorily on alloys high in zinc on 10-hour operation.

It was found both feasible and convenient when making yellow brass from copper, zinc, and scrap to charge the zinc with the rest of the charge at the start instead of "speltering" toward the end of the heat.

The metal melted in the experimental work supervised by the Bureau of Mines all went into the regular output of the Michigan Smelting and Refining Co. After the bureau's tests were finished that firm put the experimental furnace into regular commercial operation.

COMMERCIAL OPERATION OF 1,300-POUND FURNACE.

In over 600 heats of red brass, leaded bearing bronze, cupronickel, etc., in which 725,999 pounds were charged, 697,256 pounds good ingot was obtained, giving a gross loss of 28,743 pounds, or 4 per cent. No deduction is made for spillings, small ingots, metal recoverable from slag, or for oil, dirt, and other nonmetallic on the charge. A similar gross loss figure for a year's operation on the coke fires was given as 7 per cent. The furnace tenders, helpers, and pourers on the crucible furnaces began to draw a bonus for saving metal when the gross metal losses in crucible melting were 6 per cent.

In melting the 363 tons of metal 127,231 kw. h. were used, metered on the secondary side of the transformer, equivalent to 138,500 kw. h. metered on the primary side, or about 380 kw. h. per ton. The furnace was run on one shift, 10 hours or less operation, and was subject to many delays due to outside causes. One furnace operator averaged 380 kw. h. per ton on about 200 tons and a series of less experienced operators 390 kw. h. per ton on about 150 tons. The electrode consumption ran from $3\frac{1}{2}$ to 5 pounds per ton. This furnace was operated until the improved 1-ton rocking furnaces were installed and then dismantled.

The rocking furnace was patented³⁶ and the patents assigned to the Secretary of the Interior as trustee.

Licensing by the Secretary of the Interior was adopted in order to obviate the danger of having some unscrupulous furnace manufacturer with a poorly built furnace advertising his furnace as "advocated by the United States Bureau of Mines" and as far as possible to insure that furnaces made under license are capable of giving the results that the bureau's experiments show should be given by the rocking type of furnace.

DETROIT ROCKING ELECTRIC FURNACE.

The Detroit Electric Furnace Co. was licensed to build and sell rocking furnaces, their furnace embodying improvements on the experimental furnace suggested by the bureau, as well as others indicated by further experience with this type of furnace. The first heat in the first furnace installed by this firm was made on August 27, 1918. On August 1, 1921, 55 furnaces were in actual operation on nonferrous alloys and five more were being installed.

Most of these furnaces are rated at 1-ton capacity, 300 kv. a. Detroit furnaces are shown in Plates XV (p. 213), XVI, XVII, XVIII, and XIX. They are described by St. John.³⁷

The furnaces are lined with corundite brick on the inside, back of this with a layer of fire brick specially chosen for its heat-insulating properties, and on the outside, next the shell, with Sil-o-cel or infusorial-earth brick.

Some furnaces have also been constructed with a two-layer lining, the other of calcined Sil-o-cel brick, the inner of corundite, the whole lining being but 9 inches thick, which increases the capacity of the furnace. The shell is slightly tilted in the large ring gears to induce end-to-end motion of the charge and to increase the stirring action.³⁸ The plug door is entirely removable from the body of the furnace by a small jib crane located near the furnace.

The door is of generous size, and by turning the furnace till the door is uppermost the charge can be put in by a mechanically dumped bucket or similar device. The electrode holders have a long path of travel, so that the electrodes can be drawn completely out of the furnace chamber to avoid breakage while charging. The other

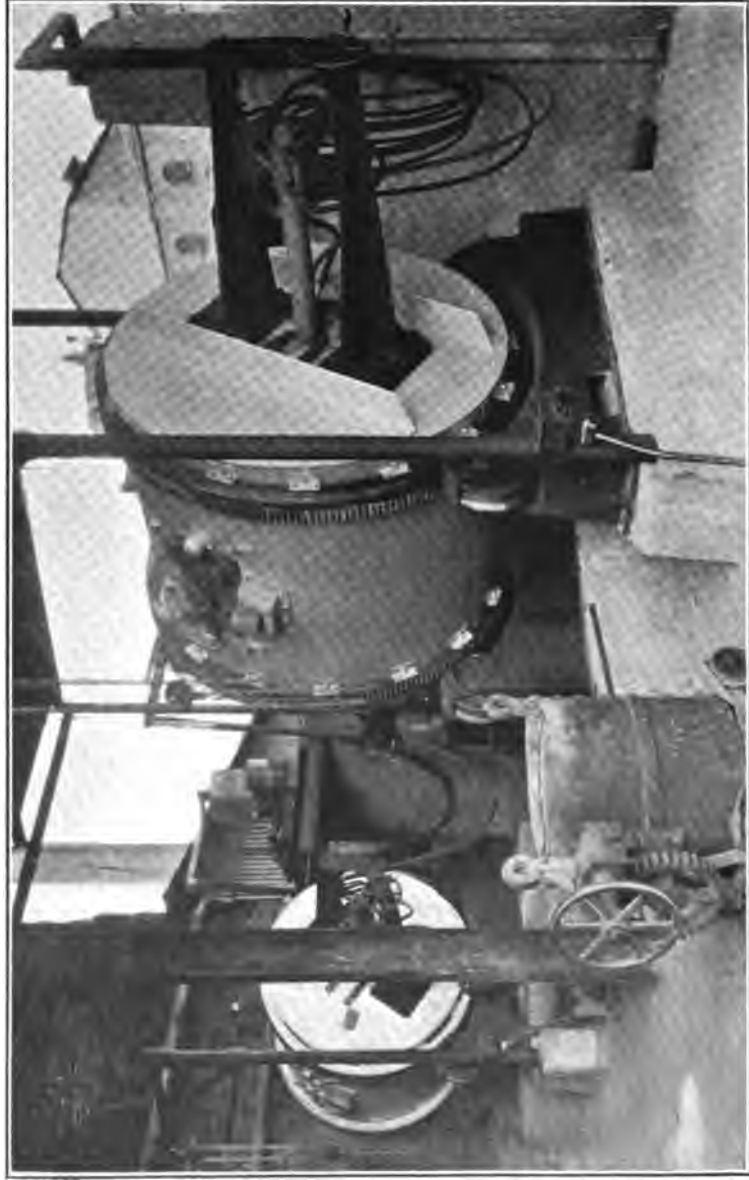
³⁶ Gillett, H. W., and Lohr, J. M., U. S. Patent 1,201,224, Oct. 10, 1916, application filed Nov. 5, 1915; Gillett, H. W., U. S. Patent 1,201,225, Oct. 10, 1916, application filed Mar. 11, 1916.

³⁷ St. John, H. M., The Detroit rocking furnaces for melting brass and bronze: *Metal Ind.*, vol. 17, 1919, p. 320; *Metal Ind.*, vol. 18, 1920, p. 211; see also St. John, H. M., Melting nonferrous metals and their alloys in the electric furnace: *Chem. and Met. Eng.*, vol. 22, 1920, p. 148; Yates, R. F., Melting brass electrically: *Sci. Amer.*, vol. 122, 1920, p. 221; Some practical data on operating electric brass furnaces: *Elec. Rev.*, vol. 76, 1920, p. 215.

³⁸ See Gillett, H. W., U. S. Patent 1,201,225, Oct. 10, 1916; compare also Crosby, E. L., U. S. Patents 1,336,820, Apr. 13, 1920, and 1,367,021, Feb. 1, 1921.



BATTERY OF ONE-TON DETROIT ROCKING FURNACES AT THE PLANT OF THE MICHIGAN SMELTING AND REFINING CO



A ONE-TON AND A HALF-TON DETROIT ROCKING FURNACE AT THE PLANT OF THE OREGON BRASS CO., SHOWING
NEW STYLE SUPPORT.

details are made clear by the photographs. A recent development has been the adaptation of this furnace for pouring direct into molds instead of by a ladle. Plates XVIII and XIX show direct-pouring furnaces in process of erection, some cables, meters, etc., not yet being in place.

In this form the furnace rests on the rollers in the base shown while melting, this form of base having now superseded that shown in Plate XV on the ladle-poured furnace as well. At pouring the furnace is rocked forward till the bar in front engages the sockets in the front of the base, and a lifting screw at the rear is swung into position so that it engages the rear hinge. The motor-actuated screw then rises, lifting the furnace bodily off the rollers and driving gears, so that the furnace is then supported and tilted like all direct-pouring or nose-tilting furnaces. The supporting bar in front is located below the lowest position of the spout during rocking and hence does not interfere with rocking. Data on some of the installations of Detroit rocking furnaces are given below.

FURNACES AT C. B. BOHN FOUNDRY CO.

On a metal loss test on 60:40 brass, poured at 975° to 1,000° C. (1,790° to 1,830° F.), in which heavy scrap and ingot were melted, 14 heats, 28,207 pounds charged, gave 27,743 pounds castings, 48 pounds spillings, and 125 pounds metal recoverable from skimmings, a net loss of 291 pounds, or 1.03 per cent.

Although the furnace was only operated five to seven hours per day during this test, the power consumption averaged 325 kw. h. per ton. The average power input was 210 kw.

During a period when two furnaces were operating, one on two 10-hour shifts, the other partly on one and partly on two shifts, 412,183 pounds of red brass and phosphor bronze were melted with 88,360 kw. h., or 433 kw. h. per ton. The furnaces were run slowly, often waiting for molds, and the time taken in charging, pouring, and waiting between heats was excessive, being longer than the actual heat itself. The actual running time per 1-ton heat averaged about one and one-half hours, while the time spent in charging, pouring, and waiting averaged about two hours per heat, although one-half hour is ample if the demand for metal is sufficient to allow running the furnaces at full speed. Such operation is, of course, a great handicap to any furnace. On 20 tons melted in one furnace during this period the average power consumption was 375 kw. h. per ton. Notwithstanding that the furnaces were not run at their highest efficiency, the melting cost was lower than in the coke fires.

The makers claim that the first furnace installed at this plant made savings in the first 18 months of its operation equivalent to five

times its cost of installation; in this period the furnace paid for itself and for four other furnaces now installed at this plant.

FURNACE AT MICHIGAN SMELTING AND REFINING CO.

On a run in which 356,391 pounds red brass were charged, 344,843 pounds of good ingot were obtained, a gross loss, including oil and dirt, of 12,548 pounds, or 3.5 per cent. Of this charge 173,551 pounds were borings, on which the average loss, when melted alone in coke-fired crucibles, was 7 per cent, or 12,100 pounds.

The 172 tons poured used 52,700 kw. h., or about 310 kw. h. per ton. The furnace was operated for two 10-hour shifts per day; that is, a 20-hour operation.

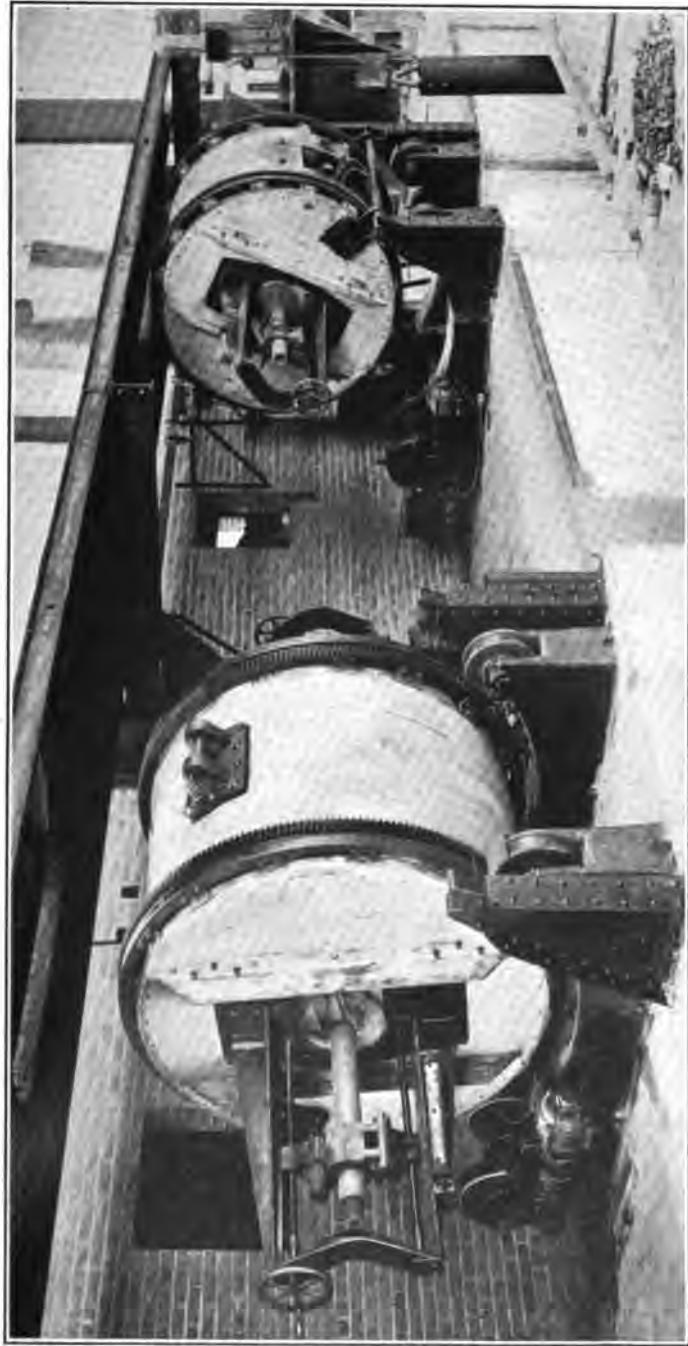
On a run in two furnaces on red brass, two-shift, 20-hour operation, in which 497,565 pounds were charged, 479,272 pounds of good ingot and 3,243 pounds metal recovered from skimmings were obtained, a loss of 15,050 pounds, or 3 per cent, which includes oil, dirt, etc., on the material charged. The 240 tons poured used 81,900 kw. h., or 340 kw. h. per ton. On 6,000 tons melted in rocking furnaces in 1919 at this plant the average power consumption was 364 kw. h. per ton.

The cost department of this firm is said to take the melting cost in the rocking furnace on 20-hour operation at one-half that in the coke fires.

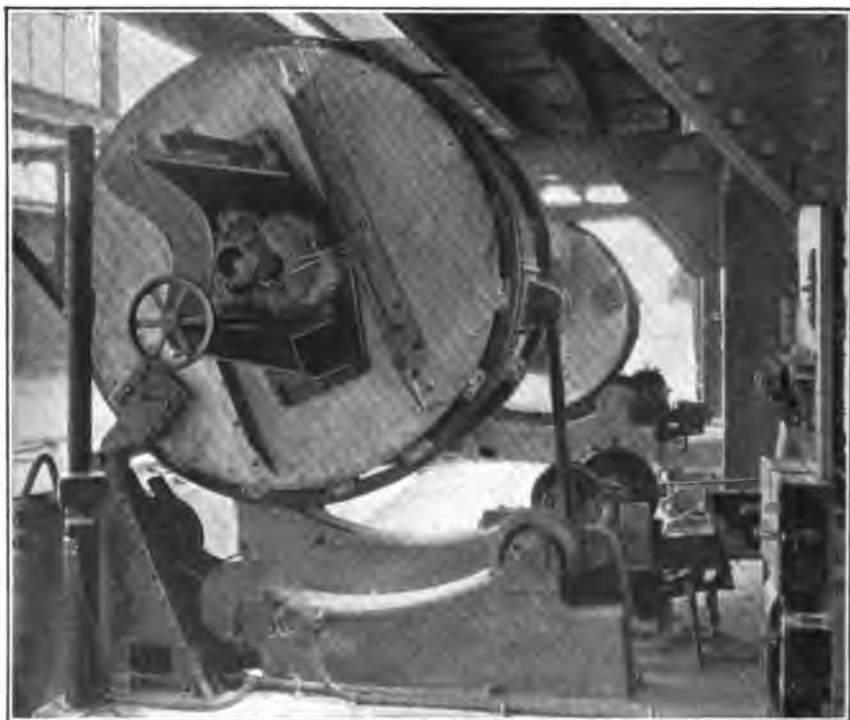
In order to find out whether it were practical to melt a charge of all or practically all oily yellow brass borings, with no admixture of heavy scrap or new metals, a test was run in one of the rocking furnaces of the Michigan Smelting and Refining Co. The borings used contained 3.7 per cent nonmetallic, mostly oil; that is, they were 96.3 per cent metallic. The composition of the brass was approximately 62 per cent copper, 35 per cent zinc, $2\frac{1}{2}$ per cent lead, $\frac{1}{4}$ per cent tin and iron.

Only 1,500 pounds of borings could be charged at the start into the furnace, which holds 2,000 to 2,400 pounds of ordinary charges, on account of the bulkiness of the borings. As the 1,500 pounds of borings carried over 50 pounds of oil and other nonmetallic material, say about 4 gallons of oil, a good deal of oil vapor appeared.

On the first three heats no attempt was made to burn off the oil after charging and before starting the arc, and the spout was not plugged tight at any time during the heat, being left open to allow the oil vapor to escape. The "safe rock" was started after the first 35 to 45 kw. h. of each heat. The furnace was warm at the start from one previous heat, but was not at full running temperature throughout the furnace walls, and therefore the power consumption was higher in the first series of heats.



A ONE-TON DETROIT ROCKING FURNACE, DIRECT-POURING TYPE IN A ROLLING MILL. LEFT, FURNACE IN ROCKING POSITION; RIGHT, FURNACE IN POURING POSITION. PHOTOGRAPHED DURING INSTALLATION; CABLES NOT ATTACHED.



DIRECT-POURING DETROIT ROCKING FURNACE IN POURING POSITION. PHOTOGRAPHED DURING INSTALLATION; CABLES NOT ATTACHED.

TABLE 46.—*Test on all-yellow borings in rocking furnace, spout open throughout run.*

[Elapsed time for three heats: 5 hours and 20 minutes—1 hour and 47 minutes per heat.]

Heat No.	Charged.			Poured.	Average recovery from skimmings.	Gross metallic loss.	Net metallic loss.	In product.	Kw. h. used.
	Metallic in borings.	Ingot.	Total metallic.						
	<i>Pounds.</i>	<i>Pounds.</i>	<i>Pounds.</i>	<i>Pounds.</i>	<i>Pounds.</i>	<i>Pounds.</i>	<i>Pounds.</i>	<i>Per cent.</i>	
1.....	a 1,444½		1,444½	1,253	75½	191½	116	66.0	260
2.....	1,444½		1,444½	1,262	75½	182½	107	66.5	248
3.....	1,444½	49½	1,494	1,302	75½	192	116½	66.9	242
Total....			4,383	3,817	226½	b 566	c 339½	66.5	d 750

a 1,500 pounds oily borings.

b Equal to 12.9 per cent.

c Equal to 7.7 per cent.

d Average, 395 kw. h. per ton poured.

In the next series the furnace was rocked back and forth during charging and after charging to burn off the oil, the spout was left open until the oil vapor no longer came off in great volume and then was plugged. If the luting blew out it was replaced until the furnace remained tight. Smoky oil vapor continued to come out of cracks in the spout luting and around the electrodes for some time after luting up, but no zinc flames appeared till the metal was almost ready to pour, when very small zinc flames appeared in places around the edges of the charging door.

TABLE 47.—*Test on all-yellow borings in rocking furnace, spout plugged.*

Heat No.	Charged.			Poured.	Average recovery from skimmings.	Gross metallic loss.	Net metallic loss.	In product.	Kw. h. used.
	Metallic in borings.	Ingot.	Total metallic.						
	<i>Pounds.</i>	<i>Pounds.</i>	<i>Pounds.</i>	<i>Pounds.</i>	<i>Pounds.</i>	<i>Pounds.</i>	<i>Pounds.</i>	<i>Per cent.</i>	
4.....	1,444½	42	1,486½	1,361	75½	125½	50	64.0	210
5.....	1,444½	47	1,491½	1,415	75½	76½	1	63.6	200
6.....	1,444½	54	1,498½	1,384	75½	114½	39	63.3	195
7.....	1,444½	34	1,478½	1,372	75½	106½	31	63.4	195
8.....	1,444½	38	1,482½	1,366	75½	116½	41	63.4	200
9.....	1,444½	43	1,487½	1,404	75½	83½	8	63.4	190
10.....	1,444½		1,444½	1,447	75½	2½	a 78	63.8	208
11.....	1,444½		1,444½	1,399	75½	45½	b 30	63.7	210
12.....	1,444½		1,444½	1,374	75½	70½	b 5	n. d.	c 290
13.....	1,444½		1,444½	1,344	75½	100½	25	62.8	190
14.....	1,444½		1,444½	1,319	75½	125½	50	64.6	200
15.....	1,444½		1,444½	1,355	75½	89½	14	64.0	200
16.....	1,444½		1,444½	1,373	75½	71½	b 4	n. d.	d 250
Total....			19,026½			e 1,123½	f 142	63.8	g 2,730

a Furnace not completely drained on all previous heats.

b Gain.

c Furnace left idle for 7 hours after most of oil had been smoked off, but before charge had begun to melt.

d Four hours between end of heat 15 and start of heat 16. Total elapsed time for 13 heats, deducting idle time in heat 12 and between heats 15 and 16, 33 hours, or 2 hours and 33 minutes per heat.

e Equal to 5.9 per cent.

f Equal to 0.75 per cent.

g Average, 200 kw. h. per ton poured, continuous or 24-hour operation.

As not all the 50 pounds or thereabouts of oil and other nonmetallic materials present in each heat volatilized entirely, but much de-

composed and gave a blanket of soot over the metal in the furnace, much metal was entrained in this soot. The soot and metal shot skimmed off from each heat was not kept separate. In the 16 heats 1,664 pounds of this material was skimmed off, from which 321 pounds of large pieces were hand-picked. The other 1,343 pounds contained 66.6 per cent, or 884 pounds, of recoverable metal, a total of 1,205 pounds or $75\frac{1}{4}$ pounds per heat. The recovered metal thus amounted to about 5 per cent of the charge.

The net loss, obtained by weighing the empty ladles before pouring, skimming, and weighing the ladle plus metal, of $7\frac{1}{4}$ per cent in the first series and only three-fourths of 1 per cent does not check well with that calculated on the analysis, which figures out to about 9 per cent in the first series and $2\frac{3}{4}$ per cent in the second. All the figures, however, tend to show that the net loss is considerable when the spout is left open throughout the heat, and not very high, for this class of material, when the spout is plugged tightly as soon as the oil is sufficiently distilled off to allow it, but before the zinc begins to be volatilized.

The furnace runs much more slowly, and gives a much lower production when operated by not running the arc till the oil is smoked off than it would if the oil were not present and the arc could be run from the start.

If the amount of borings is small enough that the heat of the furnace walls can distill off all the oil so quickly that little of it decomposes inside the furnace to form soot, production is not much cut down, as the borings can be charged first and the oil will come off while the rest of the charge is being put in. Mechanical charging is desirable in such a case. The outrush of burning-oil vapor makes hand charging difficult, though by pointing the spout upward and throwing the charge in with a shovel, the workmen can keep out of the flame. If the oil is distilled out of instead of decomposed within the furnace the entrainment of shot metal by the blanket of soot is prevented, and the percentage of charge that has to go through a recovery process is much reduced.

It is possible that much better results would be obtained on charges of all yellow, oily borings if 1,000 pounds only were charged at a time in a 1-ton furnace. Red borings are not so bulky, do not as a rule hold so much oil as yellow ones, and, because of their greater compactness, are heated more quickly to the point where the oil will distill out, so that charges of all red borings, even though quite oily, are not troublesome. If 500 pounds were charged and melted, and then 500 pounds more charged and melted, followed by another or perhaps two more such charges, the escape of oil might be hastened. As it is generally poor policy, however, to charge a large amount of cold metal into a small mass of hot metal on account of the danger of

freezing the whole charge, and as it takes time to open and close the furnace, the use of 1,000-pound or even smaller charges might be the best way to handle charges of all oily yellow borings in the 1-ton furnace.

Melting this oily material alone is about the most disagreeable work the brass melter ever has to do. The usual method is to use a pit coke-fired crucible furnace, and to poke the charge continually in order to keep the soot from preventing the coalescence of the globules formed as the borings melt. The labor cost for melting is high, as a furnace tender can attend only to few pots.

Such poking or rabbling is difficult on any large electric furnace taking a large charge on account of the oil flame. All oily yellow borings have been satisfactorily melted in the small Ajax-Wyatt induction furnace, by adding about 25 pounds every 10 minutes, and poking them in, but this method also makes the labor cost high.

According to the technical superintendent²⁹ of the Michigan Smelting and Refining Co., neither the rocking furnaces nor the Bailly furnaces, of which four of each are installed at that plant, can handle as well as the coke fires charges of 100 per cent oily yellow borings, the former because of lowered production from difficulties caused by the oil and the latter because of low production and the necessity of having a man stand in front of the furnace and rabble the charge continually. Such charges therefore are still run in the coke fires. Of the total output of brass and bronze in that plant about 60 per cent, late in 1919, was being melted in the four rocking furnaces, about 30 per cent in the coke fires, and about 10 per cent in the four Bailly furnaces.

FURNACES AT DETROIT COPPER AND BRASS ROLLING MILLS.

In a test on yellow brass 61½ per cent copper, 36 per cent zinc, 2½ per cent lead, at the Detroit Copper and Brass Rolling Mills, in 9-hour operation, 14 heats of 1 ton each required 4,085 kw. h., or about 290 kw. h. per ton. On 16-hour, 2-shift operation, 18 heats required 4,500 kw. h., or 250 kw. h. per ton. On the test runs the metal loss, on yellow brass, made up of fairly heavy scrap and new metals (zinc charged at the start), ran from 0.5 to 0.6 per cent.

According to the metallurgist of this firm, in regular operation 10 heats are usually obtained in 16-hour operation, two 8-hour shifts. The power consumption averages 250 kw. h. per ton or less, and the metal loss runs under 1 per cent. The total electrode consumption is about 3½ pounds per ton. The metal is said to be mixed with absolute uniformity, no greater difference in composition between the first and last metal poured being found than between duplicate analyses of the same sample. That is, within the limits of analytical er-

²⁹ Private communication, October, 1919.

ror the mixing is perfect. The metal is poured direct from the furnace to the molds. The first furnace, installed in an inconvenient location and under rather unfavorable conditions, when operated entirely by a negro crew, gave approximately the same labor costs as on the coke furnaces. With a battery of mechanically charged furnaces it is expected that the labor cost will be materially reduced.

On another recent 5½-day test at this plant 34 tons were melted, 6 heats of 1 ton each being made per 9-hour day, the average power consumption being about 275 kw. h. per ton, and the net metal loss 0.90 per cent on a charge of half new metal, half scrap.

The original furnace at this plant has been altered to the direct pouring type, and when the furnaces that are now being installed are in operation this firm will have eight 1-ton direct-pouring Detroit furnaces.⁴⁰

RESULTS AT CHASE METAL WORKS.

At the Chase Metal Works, operating two furnaces on 60:40 brass, poured at 2,000° F. (1,100° C.) or above, one furnace was run one 8-hour shift only, and for 170 1-ton heats averaged 6,800 pounds output per 8 hours, the average power consumption being 280 kw. h. per ton.

The other furnace, run two 8-hour shifts, averaged for 100 1-ton heats 16,600 pounds output; that is, it melted 8 to 9 tons in 16 hours. The actual melting time per heat was only 56 minutes, so that by speeding up the pouring and charging the output could be increased materially. The average power consumption was 225 kw. h. per ton.

For 127 heats in both furnaces, the average metal loss was 1 per cent.

In the melting of 500 tons of 60:40 brass in four furnaces, this plant later cut the actual melting time to about 52 minutes per heat. The average power consumption was about 240 kw. h. per ton and the net metal loss 1.02 per cent, the charge being one-half new copper and zinc, the other half heavy scrap and briquetted borings. Still later figures on 60:40, 24-hour operation, showed on 55 heats of all clean metal an average of 225 kw. h. per ton and an output of 16½ tons per day, with 33 consecutive heats running at 210 kw. h. per ton and 18½ tons output, while on 79 heats which contained about 25 per cent of oily borings the average was 265 kw. h. per ton and the output 12½ tons. Gross losses ran from 1.85 per cent on clean metal to 4.27 per cent on dirty, oily charges. Net losses were not determined.

⁴⁰ See Prentiss, F. L., Direct routing in a modern brass plant: *Iron Age*, vol. 108, 1921, p. 63.

FURNACE AT CLEVELAND BRASS AND COPPER ROLLING MILLS.

On charges of 1,000 pounds heavy scrap, 900 pounds light scrap, the alloy melted being 60 per cent copper, $37\frac{1}{2}$ per cent zinc, $2\frac{1}{2}$ per cent lead, a $5\frac{1}{2}$ -day run, working 8 hours per day, at the Cleveland Brass and Copper Rolling Mills, showed a production of 6 heats or 5.7 tons per day, a power consumption of about 220 kw. h. per ton, and a gross metal loss of 1 per cent. The electrode consumption was 2.4 pounds per ton.

METAL LOSS COMPARED WITH THAT IN OIL FURNACE.

An interesting comparison was made by the Ford Motor Co. between the composition of the metal produced by the rocking furnace and an open-flame oil furnace, the same charge being fed to both furnaces. The alloy desired was a bearing bronze of about 87 per cent copper, $8\frac{1}{2}$ per cent tin, $3\frac{1}{2}$ per cent zinc, $1\frac{1}{2}$ per cent lead with a small amount of phosphorus. The analysis for zinc and lead showed the following results:

TABLE 48.—Comparative analysis of products from similar charges melted in open-flame oil furnaces and in rocking electric furnaces.

Open-flame oil furnace.		Electric rocking furnace.	
Zinc.	Lead.	Zinc.	Lead.
Per cent.	Per cent.	Per cent.	Per cent.
3.08	1.73	3.71	1.30
3.08	1.85	3.84	1.70
1.32	1.85	3.82	1.77
3.14	0.99	3.59	1.73
1.55	1.83	3.59	1.73
2.62	1.28	4.08	1.85
2.38	1.65	4.01	1.56
3.26	1.31	4.09	1.56
1.21	1.47	4.71	1.73
2.56	1.42	4.18	1.48
3.12	1.59	4.71	1.93
1.16	1.47	3.84	1.60
2.94	1.91	4.38	1.96
3.37	1.39	4.35	1.74
2.11	1.39	6.63	1.56
		4.05	1.47
		4.25	1.43
a 2.53	a 1.54	b 4.05	b 1.65

a Average—open-flame oil furnace—zinc, 2.53 per cent; lead, 1.54 per cent; total, 4.07 per cent.

b Average—electric rocking furnace—zinc, 4.05 per cent; lead 1.65 per cent; total, 5.70 per cent.

An occasional variation may be due to variations in the individual charges, though all charges to both furnaces were calculated to be the same. The lowest zinc content in the electrically melted metal was higher than the highest from the open-flame oil furnace. As the electrically melted alloy contained in 100 pounds 94.3 pounds of copper and tin and 5.7 per cent lead and zinc, the metal obtained in the oil furnace from the same amount of charge would

contain 94.3 pounds copper and tin and 3.95 pounds lead and zinc $\left\{ \frac{3.95}{598.2} = 4.08 \text{ per cent} \right.$, the metal melted by oil gave 94.3 and 3.95, or 98.25 pounds, from the same weight of charge that gave 100 pounds to the electric; that is, the metal loss, calculated from the average analysis, is 1.75 per cent more in the open-flame oil furnace than in the electric.

In another plant which also operated another make of open-flame oil furnace and a rocking furnace on this same alloy, with a charge that contained 25 per cent borings and grindings, the gross metal loss in the oil furnace was found on test to be 2.69 per cent and that in the rocking furnace 0.94 per cent; the rocking furnace thus saved again 1.75 per cent over the open-flame oil furnace.

The saving found in one plant, calculated by analysis, and that found in another plant, determined by weighing, is thus seen to be exactly the same on the same alloy.

After the test cited above showed that the Ford Motor Co.'s electric furnace was giving a product higher in zinc than was aimed at owing to its lower loss, the charge to the electric furnace was altered to compensate for this difference, and the following analyses on 13 consecutive heats shows the uniformity of the product from heat to heat:

TABLE 49.—*Analysis of 13 consecutive heats from the rocking furnace.*

Copper.	Tin.	Lead.	Zinc.	Phosphorus.
86.90	8.35	1.63	3.04	0.015
87.18	8.43	1.09	3.28	.020
86.68	8.37	1.37	3.56	.020
87.24	8.36	1.26	3.10	.020
86.84	8.20	1.21	3.73	.014
86.94	8.10	1.04	3.91	.014
87.80	7.64	1.36	3.20	.020
87.16	8.27	1.30	3.26	.014
86.96	8.27	1.33	3.43	.010
86.82	7.98	1.21	3.96	.025
87.20	8.27	1.25	3.27	.014
87.12	8.27	1.23	3.36	.016
86.80	8.59	1.21	3.39	.013
≈ 87.06	≈ 8.24	≈ 1.27	≈ 3.42	$\approx .0165$
MAXIMUM DEVIATIONS.				
-0.26 + .74	-0.60 + .35	-0.23 + .34	-0.38 + .54	

$\approx$ Average.

FURNACE AT PLANT OF ALUMINUM MANUFACTURERS, INC.

On 40 heats, 2,125 pounds per heat, of phosphor bronze, poured at 2,250 to 2,300° F. (1,230 to 1,260° C.), a much higher temperature than the average plant requires, the average power consumption in a rocking furnace at one of the plants of Aluminum Manufacturers,

Inc., was 330 kw. h. per ton, the furnace being operated 9 hours per day. The charge was poured in 8 ladles, each holding 265 pounds. It took an average of 25 minutes to pour, and 25 minutes to charge a heat. On account of the long time spent in charging and pouring, only 4 heats, $4\frac{1}{2}$ tons, were made in 9 hours. This firm now has two rocking furnaces in operation.

FURNACE AT ROME WIRE CO.

It is claimed by the users that the 1-ton furnace at the Rome Wire Co., melting cabbaged copper wire and scrap copper into wire bar, operating 10 hours per day, produced 100 per cent conductivity, Mathiesen scale, metal at 280 kw. h. per ton with an output of 6 tons per day. The furnace requires expert operation to give as good a performance as this on copper. This firm has discontinued that part of their manufacturing work for which they were using the furnace, so that this furnace is no longer in use.

FURNACE AT PLANT OF MICHIGAN LUBRICATOR CO.

A time analysis on 18 heats of brass containing 73 per cent copper, 18 per cent zinc, 4 per cent tin, 5 per cent lead, melted in a rocking furnace at the plant of the Michigan Lubricator Co., showed that the average time used in a heat for the different operations was as follows:

	Minutes.	Remarks.
Opening furnace door-----	2	Metal is poured with charging door in.
Cleaning out dross and slag-----	$2\frac{1}{2}$	
Charging-----	$11\frac{1}{2}$	
Rocking—no arc on-----	$8\frac{1}{2}$	This to drive out oil as charges contained oily borings.
Clearing charge away from electrodes-----	$2\frac{1}{2}$	Rocking to drive out oil may bring the charge so close to the electrodes that there is danger of overheating before rocking is started, so any metal lying too close to the arc is pushed away.
Closing and luting door-----	$2\frac{1}{2}$	Average power input to arc about 240 kw. while it is running.
Running arc—furnace not rocked-----	$27\frac{1}{2}$	
Running arc—furnace rocked-----	$62\frac{1}{2}$	
Pouring-----	28	Included time spent at end of day taking out and cleaning electrodes to prevent sticking, as well as taking a fresh grip on electrodes with holders, oiling bearings, and all other furnace maintenance.
Adjusting electrodes and all furnace maintenance-----	6	
Total, 2 hours $33\frac{1}{2}$ minutes.		

It thus takes 10 hours for 4 heats, 2,040 pounds per heat, poured at about $1,100^{\circ}$ C. When the charge does not contain oily borings, it does not have to be left with the arc off for oil to smoke out, does

not have to be cleared away from the electrodes, the arc holds better than it does in the atmosphere of oil vapor so that a higher kilowatt-hour input can be held; there is no blanket of soot on top of the metal to slow up the absorption of heat, therefore 5 heats can be made in $9\frac{1}{2}$ hours. A sixth heat could be made by speeding up the pouring time. On a 4-day run, 20 heats, the power consumption was 315 kw. h. per ton; on another 4-day run, 16 heats, with oily borings present, so that only 4 heats per day were made, the power consumption averaged 350 per ton. The last heats of the day, under the condition that would hold in 24-hour operation, came out at 300 to 315 kw. h. per ton when borings were used, and 260 when they were not used. On another run on the 18 per cent zinc alloy, melting 37 tons in 7 working days, 5 heats in 9 hours, the power consumption was about 315 kw. h. per ton, the electrode consumption 3.2 pounds per ton, and the gross metal loss 2.5 per cent on a charge of 45 per cent gates, scrap, lead, and copper, and 55 per cent borings. The average nonmetallic in the charge was 1.6 per cent, making the net metallic loss 0.9 per cent.

FURNACE AT SHERWOOD BRASS FOUNDRY.

On a 50-ton run of red brass, 85 per cent copper, 5 per cent tin, 5 per cent zinc, 5 per cent lead, in one of the 1-ton rocking furnaces at the Sherwood Brass Foundry, the charge being mostly ingots and gates, the average output was $4\frac{2}{3}$ tons in 9 hours, 20 minutes, or 1,000 pounds per hour, elapsed time, including all delays.

The power consumption was about 345 kw. h. per ton. In melting 51 tons, 170 pounds electrodes were actually consumed and 34 pounds broken and discarded, total 204 pounds, or 4 pounds per ton.

The other 1-ton rocking furnace at this plant, on 17 tons of leaded bronze (82 per cent copper, 10 per cent tin, 8 per cent lead, trace phosphorus), from a charge containing 20 per cent borings, showed an output of 825 pounds per hour, elapsed time, including a good deal of waiting for molds, and used about 330 kw. h. per ton. The total electrode consumption was 4.1 pounds per ton. On 24 tons, partly of this alloy and partly of red brass, the output was 890 pounds per hour, the power consumption 330 kw. h. per ton, and the electrode consumption 5.1 pounds per ton. A third furnace has since been installed.

FURNACE AT PLANT OF DENNY-RINE CO.

A 1-ton rocking furnace operated in 1919 at the plant of the Denny-Rine Co., Chicago, making ingot, on 24-hour operation, on an alloy of 83 per cent copper, 3 per cent tin, 4 per cent lead, 10 per cent zinc, made up of bulky scrap and 30 per cent oily yellow borings, the charge being so bulky that only 1,600 pounds could be charged

into the 1-ton furnace, gave 20 heats, or 16 tons, in 24 hours, at about 270 kw. h. per ton, with a gross metal loss of 3.6 per cent on a charge containing 2.9 per cent of oil, dirt, and other nonmetallic, or 0.7 per cent net metal loss.

On 24-hour operation on 2,200-pound charges of all red brass borings, 21 to 22 tons were produced in 24 hours, at about 240 kw. h. per ton. The electrode consumption was $2\frac{1}{4}$ pounds per ton on 24-hour operation and $3\frac{1}{2}$ pounds per ton on single-shift operation.

During the period February 1 to September 1, 1919, it is said that the power consumption per ton of metal shipped from this plant was 290 kw. h. per ton. This period included both 24-hour and 1-hour shift operation. The lining life was 600 heats on intermittent operation. This company sold its plant to the Hills-McCanna Co., which now operates the 500-pound furnace formerly used by the Denny-Rine Co. The furnace used by the Denny-Rine Co. is now the property of the Parrish-Pool Co., of Cleveland, which also operates two other 1-ton Detroit furnaces.

A test on leaded bearing bronze made in the Denny-Rine Co.'s furnace shows how lead is absorbed by a hearth until the hearth becomes saturated with it, and also shows the uniformity of the alloy produced by the rocking furnace.

TABLE 50.—*Analysts of first and last metal poured from rocking furnace melting alloys high in lead.*

Heat No.		Cu.	Sn.	Pb.	Appar- ent metal loss—no correc- tion for oil or dirt.
		Per cent.	Per cent.	Per cent.	Per cent.
1	First ingot from first ladle.....	72.73	4.97	21.02	8.0
	Last ingot from last ladle.....	72.12	4.88	20.97	
2	First ingot from first ladle.....	68.89	4.86	23.19	5.86
	Last ingot from last ladle.....	69.09	4.47	22.96	
3	First ingot from first ladle.....	66.51	4.24	25.49	3.5
	Last ingot from last ladle.....	67.15	4.30	25.28	
4	First ingot from first ladle.....	67.07	4.30	25.02	2.6
	Last ingot from last ladle.....	66.01	4.47	25.87	

FURNACE AT GENERAL ALUMINUM AND BRASS MANUFACTURING CO.

On a test of red brass 85 per cent copper, 5 per cent each of lead, tin, and zinc (38 per cent new metal and ingots, 38 per cent foundry scrap, 24 per cent borings), 34 tons were melted in 60 working hours. Six 1-ton heats were made in 9 hours and 7 in 10 hours. The weight of charge was 68,404 pounds and 68,101 were poured, a gross loss of 303 pounds or 0.44 per cent. The daily gross loss varied from 0.3 per cent to 0.7 per cent. The average electrode consumption was 3.5 pounds per ton. About 9,880 kw. h. were used, an average of

about 290 kw. h. per ton. The average pouring temperature was 1,140° C. (2,085° F.). The power consumption at the end of a day's run was about 250 kw. h. per 1-ton heat. A second furnace is now in use by this plant.

FURNACE AT BRIDGEPORT BRASS CO.

A Detroit furnace used at the Bridgeport Brass Co., operating on pure copper and copper-tin alloys, had some trouble from short life of the refractory lining, but a recent communication⁴¹ from this firm says: "We have recently been getting very satisfactory results as we have largely remedied the lining troubles that we previously had."

FURNACE AT WHEELER CONDENSER AND ENGINEERING CO.

This furnace was at first handicapped by the line voltage falling lower than was expected, with the result that the furnace could take only about 200 kw. Production was retarded and the power consumption raised. On 150 tons of 60:40 brass, 10-hour operation, with a good many days when the furnace was not run owing to troubles with the power supply, and when as a result unusual preheating was involved, the power consumption ran something under 400 kw. h. per ton, and the gross metal loss was 0.91 per cent. The charges were practically all light scrap and each contained 100 pounds of recovered concentrates which carried some nonmetallic.

On 25 tons, melted later, of 60:40 alloy, 10-hour operation, a Monday morning preheat being included, the power consumption was about 250 kw. h. per ton. In a 94-hour continuous run, melting 37 tons of copper, the gross loss was 0.34 per cent, and the power consumption was about 375 kw. h. per ton. All these runs were handicapped by the low power input.

FURNACE AT WHITE AND BROTHER.

On the first 97 tons, both red and yellow, 10-hour operation, the power consumption was around 350 kw. h. per ton. The latter part of the run gave around 300 kw. h. per ton. If given in terms of the furnace kw.-h. meter all the power consumption figures on the Detroit rocking furnaces cited in this report have been corrected to include the losses in transformers and secondary leads in order to include all the power used.

The power factor of these rocking furnace installations including transformers is usually better than 80, though McMicken⁴² cites one installation with an over-all power factor of but 72.

Prices of the Detroit rocking furnace, quoted February 24, 1920, east of the Rockies, are f. o. b. Detroit, as follows: 500-pound size, \$6,500; 1,000-pound size, \$8,500; 1-ton size, \$11,500. Automatic

⁴¹ Webster, W. R., personal communication, Aug. 24, 1921.

⁴² McMicken, A. C., Elec. World, vol. 75, 1920, p. 1251.

electrode control costs about \$750 additional and a nose-tilting device for the 1-ton furnace costs \$1,250 additional.

These figures include transformer, reactance, oil-switch, and its remote control operating device, switchboard complete with watt-meter and watthour-meter, contactors, rocking motor, worm-gear reduction drive, rocking-control device, etc. Hyatt roller bearings are supplied on all principal bearings.

LIFE OF LINING IN THE ROCKING FURNACE.

The general average of the life of the lining in the 1-ton rocking furnace is given as about 350 heats though 600 to 800 heats are not uncommon. One plant, pouring at about 1,250° C. (2,275° F.), has obtained only about 125 heats, in one case getting only 25. Oddly enough the brick used in the lining that ran but 25 heats was from the same shipment as a lining that ran 737 heats in another 1-ton furnace at another plant.

The cost of the inner course of corundite brick, which is all that need normally be replaced when relining is done, is given (Mar. 1, 1920) as about \$100, and the labor for relining about \$50. Graphite electrodes cost about 25 cents per pound.

PERFORMANCE OF SMALLER SIZES OF ROCKING FURNACE.

Only scanty data are available on the performance of the smaller sizes. The 1,000-pound furnace at the Oregon Brass Works, operated to average 5 heats, or 2½ tons in 8 hours, gave an average power consumption of 350 kw. h. per ton on 60 heats of red brass, gun metal, and leaded bronze. This size has not been operated on 24-hour service, but the figures for the last heats of the day indicate that it would run at about 250 kw. h. per ton.

The 500-pound furnace at the Hills McCanna Co., operated to give four heats, or 1 ton in 8 hours, used about 385 kw. h. per ton on red brass. On 14 heats, the furnace being operated only for 2 or 3 heats on most of the days, the power used was 425 kw. h. per ton. From the last heats of the day, it is indicated that this size would take about 300 kw. h. per ton on 24-hour operation. According to later data,⁴⁸ when operated to give 6 heats (1½ tons) in 9 hours, the furnace takes around 315 kw. h. per ton, measured on the secondary side of the transformer or, say, about 340, measured on the primary. By cutting down the delays between heats, 7 heats, 1½ tons, can be made in 9 hours, and the primary power consumption would be about 325 kw. h. per ton. On 24-hour operation, it appears from the last heats of the day, the power consumption should fall to about 275 kw. h. per ton.

⁴⁸ Diller, H. E., Adapt ingot foundry to castings, Foundry, vol. 48, 1920, p. 506.

In May and June, 1920, the furnace being operated to give from 2 to 6 heats a day, averaging about 4, the average output was 346 pounds per hour, at 374 kw. h. per ton, measured on the secondary, that is, about 400 on the primary. The Chapman Valve Manufacturing Co., Indian Orchard, Mass., is installing a furnace like the regular 500-pound size, but with a thinner lining, so as to hold 750 pounds of metal, and provided with a 125-kw. transformer.

ROCKING FURNACE USED IN PREPARATION OF ALUMINUM STEEL AND GRAY IRON.

The Michigan Steel Castings Co. installed a 1-ton rocking furnace for use as a mixer in making up an experimental series of alloys consisting of a low-carbon steel containing 12 to 20 per cent aluminum and 0 to 1 per cent titanium. It was hoped that such alloys might be useful where a material resistant to oxidation at high temperatures is required.

The furnace was operated intermittently; for example, the cold furnace would be raised to a red heat by the use of about 240 kw. h., 200 pounds aluminum charged, melted, and held by the use of 50 kw. h. till the steel was ready to add, then 700 to 750 pounds of molten steel from a Heroult furnace was added, 40 pounds ferrotitanium charged, and the furnace rocked 15 to 20 minutes, using 75 to 100 kw. h. before pouring the resultant alloy.

The furnace was said to give thoroughly mixed metal of uniform analysis, and to allow controlling the pouring temperature, about 2,900° F. (1,600° C.) satisfactorily. The ordinary corundite lining used was not attacked in the 25 experimental heats run; instead there was slowly built up an accretion of slag or dross, which, on being chipped off, left the lining underneath in good condition.

In the casting and use of these alloys some difficulties were found which required further experimental work on the properties of the alloys themselves. No commercial use therefore has been made of the furnace for this purpose.

The Homestead Valve Mfg. Co., Homestead, Pa., has installed a 1-ton Detroit furnace with automatic electrode control to melt gray iron and semisteel. Another 1-ton furnace for the same purpose is being installed by the Russell Wheel and Foundry Co., Detroit, Mich.

According to tests on melting gray iron, it appears that the furnace will produce about 5 tons in a 10-hour day, at around 500 kw. h. per ton. Tensile tests of metal of around 2.75 per cent C, 1.50 to 1.70 per cent Si, 0.50 per cent Mn, 0.06 per cent S, 0.45 per cent P gave about 42,000 pounds per square inch. As high as 70 per cent borings and turnings were successfully used.

DETROIT FURNACE MELTING ALUMINUM.

A 150-kw. Detroit furnace, with a thinner lining than the regular 150-kw. brass furnace, so as to give a capacity of 1,000 to 1,200 pounds of aluminum has been installed for aluminum melting by the Storm Peak Potash Co., Denver, Colo., but no data are yet available on its performance.

UNIVERSAL ENGINEERING CORP. ROCKING FURNACE.

The Universal Engineering Corp., Baltimore, Md., applied early in 1919 for a license to manufacture rocking furnaces for sale under the Bureau of Mines patents and built a furnace at the plant of Jos. Brenner & Co., Hagerstown, Md., the drawings for which show that the design embodies many of the improvements worked out by the Detroit Electric Furnace Co. over the bureau's experimental furnace, apparently differing only in minor details of design from the Detroit furnace.

The furnace was operated in August, 1919, and was reported by the electrical engineer of the Universal Engineering Corp. as working very well, but as having some minor faults in mechanical construction. Late in October the electrical engineer reported it had been decided practically to rebuild the furnace and to correct all the faults of construction that had yet been discovered before submitting the furnace for the inspection and test by the Bureau of Mines that must be made before a license to sell the rocking furnace can be issued. No quantitative data on the performance of the furnace has been submitted to the bureau, but it was stated that in the first trials yellow brass scrap was melted, and then skimmings were tried, the recovery of metal being so high that the furnace was kept at this work for some time, though some trouble was met by the slag building up onto the lining. The furnace was then used for recovery of lead residues. The furnace is said to have operated for a period of 8 months on a corundite lining in such service. How nearly continuous the operation was during that period is not known. Repeated requests to the Universal Engineering Corp. for data on the performance of the furnace failed to secure any data whatever, and since the work with the furnace appeared to have ceased, notification was sent to that firm by the Director of the Bureau of Mines in July, 1920, that in view of the firm's failure to supply data and to continue development of the furnace, cooperation of the bureau was at an end and a license would not be issued.

REVOLVING BRASS FURNACE.

In 1917 one revolving oil-fired open-flame brass furnace was put on the market ⁴⁴ and another ⁴⁵ was advertised late in 1919.

⁴⁴Anonymous, A noncrucible revolving furnace, *Metal Ind.*, vol. 16, 1918, p. 38.

⁴⁵Anonymous, A continuous revolving noncrucible metal melting furnace, *Metal Ind.*, vol. 17, p. 498, 1919.

Following publication of the experiments of the Bureau of Mines with the rocking type of furnace, and the successful commercial use of the Detroit furnace, the idea of moving an electric brass furnace while running was taken up by various designers and builders of electric brass furnaces and applied to various furnaces, on the operation of which very little information is yet available.

The gyrating or revolving Thomson zigzag-resistor furnace design and the Harvey revolving-furnace design have already been described. Another furnace, the Booth, has been designed, in which the furnace is revolved in one direction instead of rocked back and forth. The Bureau of Mines considered the complete revolution of the furnace instead of rocking it, and the use of a taphole, as well as the use of more than one set of electrodes for polyphase current, as will be seen from the patents,⁴⁶ but did not construct a rotating or revolving form of furnace for the following reasons: First, continuous rotation of the furnace involves the use of some sort of a sliding contact in the electrical circuit which is not extraordinarily difficult to handle on a small furnace taking, say, only 100 kv. a. though even in the smaller furnaces trouble would be expected in the maintenance of such contacts amid the dust and dirt of a foundry. But in a furnace of sufficient capacity to make it generally useful, as a 1-ton, 300-kv. a. furnace, the use of a moving contact to carry a current of around 2,500 amperes hardly appears to be desirable.

There must also be a revolving joint in the water line to the electro coolers, which must be carefully maintained to keep it from leaking. The pains taken by Stassano⁴⁷ to construct his later forms of the moving Stassano furnace to allow the use of cables and water connections having a permanent and solid connection in place of the brushes, slip-rings, etc., used in the early rotating form, shows the conclusions reached on this point in attempts to employ a sliding contact on a large electric furnace under foundry conditions.

Second, a furnace which admits only of continuous rotation in one direction can not be started in motion when it contains a partly solid charge containing heavy ingots or chunks of metal, as these may be carried up by the motion of the furnace so far that they do not fall until they are over the electrodes, and when they do fall, they hit the electrodes and break them. On a charge of all borings, the furnace could be rotated with comparative impunity, as the borings might be too light to break the electrodes, but on an ingot charge, it is necessary either to use a charge far under the proper capacity of the furnace or to wait till the material is almost all melted before starting to

⁴⁶ Gillett, H. W., and Lehr, J. M., U. S. Patent 1,201,224, Oct. 10, 1916.

⁴⁷ Schmelz, E. M., A new electric steel casting plant: *Met. and Chem. Eng.*, vol. 11, 1913, p. 709; Stassano, E., U. S. Patent 1,024,657, Apr. 30, 1912; 1,059,499, Apr. 22, '13; 1,105,859, Aug. 4, 1914.



A. A ONE-TON BOOTH ROTATING FURNACE, SHOWING AUTOMATIC ELECTRIC CONTROL



B. A 500-POUND BOOTH ROTATING FURNACE.



A. FIRST COMMERCIAL AJAX-WYATT FURNACE.



B. HERING BRASS FURNACE TESTED AT THE PLANT OF THE
AJAX METAL CO.

rotate. Inspection of figure 32 will show how rotation of the furnace past a certain point in the first three or four views would cause the ingots to tumble over on the electrodes.

On alloys high in zinc it is essential that the furnace start to move as soon as an appreciable quantity of molten metal has accumulated so that any of it which has become pocketed under the arc may be poured out of the pocket. If it is retained, it becomes superheated, just as in any stationary indirect-arc furnace, and the pressure of zinc vapor becomes so great that the furnace can not be kept tight, and loss of zinc ensues. Just as it was found that the 600-pound Snyder furnace could be kept tight on an alloy high in lead, though the 1-ton furnace could not, so the difficulty of keeping an indirect-arc furnace tight on alloys high in zinc increases as the size increases. A 1-ton furnace melting 60:40 brass can be operated only with great difficulty, if at all, unless the stirring of the metal by rocking is begun long before enough of the charge has melted to allow wide rocking of the furnace without breaking the electrodes. Hence the rocking furnace has been provided with means for varying the rocking angle from the "safe rock" to the "full rock" as the melting progresses.

These reasons, and various other minor ones having to do with convenience in operation, were sufficient to outweigh the two advantages of complete rotation: (1) The washing of all the furnace wall with metal instead of about 90 per cent of it; (2) the greater stability of a solid lining instead of one pierced by the opening for the charging door.

BOOTH REVOLVING FURNACE.

The Booth ⁴⁸ furnace is a rotating indirect-arc furnace, which in a 250-pound size was first tried out at the plant of Leitel Bros., Chicago, the first heat being made on May 17, 1919. A 500-pound size of this furnace is shown in Plate XX, *B*, and a 1-ton size in Plate XX, *A*; it is a horizontal cylinder mounted on rollers and with the electrodes entering on the horizontal axis, adjustable by electrode holders, the design closely resembling that of the rocking furnace save that the metal is tapped from one end and charged from the other by opening a door which carries one electrode and its holder. This method of tapping and charging has been criticised as inconvenient by some users. Current is carried to the electrodes by flexible leads connected to two bronze shoes on the shell and insulated from it which take current from the main cables by sliding contacts.

⁴⁸ Booth, C. H., The Booth electric rotating brass furnace, *Iron Age*, vol. 103, 1919, p. 1699; *Metal Ind.*, vol. 17, 1919, p. 317, vol. 18, 1920, p. 456; *Chem. Met. Eng.*, vol. 21, 1919, p. 637; *Foundry*, vol. 18, 1920, p. 994; Booth, W. K., U. S. Patent 1,332,795, Mar. 2, 1920, and 1,380,767, June 7, 1921; Harper, C. W., U. S. Patent 1,336,073, Jan. 18, 1921.

The makers of the furnace say that when turnings or similar materials are melted, rotation can be started at once; but when ingots are melted rotation must be delayed till the metal is partly melted in order to avoid electrode breakage. Emphasis is laid on the fact that a small furnace of this type could be lifted off bodily and carried to the molds for pouring by a crane; it is not known whether this has been actually tried.

The output of a 250-pound furnace is given as 1 ton in $7\frac{1}{2}$ to 8 hours, and the power consumption is stated to be as low as 240 kw. h. per ton with the furnace hot; 300 kw. h. per ton is claimed on 8-hour operation, and a lining life of 600 to 1,000 heats is claimed.

Metal losses are low; dirty concentrates from floor sweepings that give 30 to 40 per cent loss in crucibles showed $17\frac{1}{2}$ per cent gross loss. Yellow brass turnings have been melted with a gross loss of $1\frac{1}{2}$ per cent; yellow ingot with 1 per cent, and red brass and bronze, below 1 per cent.

On individual heats of clean, yellow, and red brass, in which great care was taken not to overheat the metal, figures of 0.4 per cent and 0.22 per cent, respectively, were obtained. A test on the 250-pound furnace on red brass showed a production of 2,000 pounds in 9 hours, at 400 kw. h. per ton with an electrode consumption of 3 pounds per ton and a metal loss of 0.75 per cent. Further data on the operation of the 250-pound Booth furnace at Leitel Bros. has been given.⁴⁹ In the month of June, 1919, the furnace run on one-shift, 9-hour operation, melted 35,215 pounds, averaged 480 kw. h. per ton, and used 4 pounds graphite electrodes per ton. The power factor of the furnace installation itself, on the secondary side of the transformer, was 70 to 71 per cent; that on the primary side, including the transformer, was 63 per cent. This power factor is so low that it involves a power factor penalty in most power contracts. It was made so low in order to make the arc maintain itself with the least attention, so that the operator would not be forced to stay at the furnace all the time for adjustment of the arc. As the output of a 250-pound furnace is so low, the labor cost with one man per furnace would become excessive. Another way out of the dilemma would be to attach a device for automatic electrode control.

During two weeks' operation in June the following metal-loss data were obtained:

	Charged.	Obtained.	Loss.	
	Pounds.	Pounds.	Pounds.	Per cent.
Red brass.....	14,787	14,067	120	0.81
Yellow brass.....	1,961	1,933	28	1.40
Yellow brass borings.....	408	350	58	14.21

⁴⁹ Electric brass furnace reduces foundry costs, Elec. World, vol. 74, 1919, p. 1107. Foundry test of rotating furnace, Chem. and Met. Eng., vol. 22, 1920, p. 136.

The borings were said to contain 8.28 per cent oil and 5.38 per cent nonmetallic, or 13.66 per cent, leaving a net metal loss of 0.55 per cent.

The power consumption on this two weeks' run was 441 kw. h. per ton. Booth⁵⁰ states that the 250-pound rotating furnace linings have lasted 550 to 600 heats and claims for the 500-pound size, in 9-hour operation, an output of 3,800 pounds of red brass at 310 kw. h. per ton. In a run of one month, during which 40 tons were melted, the power consumption was 370 kw. h. per ton, and the electrode consumption was 3.7 pounds per ton.

The first lining of the 250-pound furnace is said to have lasted almost 500 heats. The makers recommend that the power factor be made 70 per cent in order to cut down the amount of hand regulation required. In this case it is stated two men can charge and handle three 250-pound furnaces.

At the Fulton Harwood Brass Works, South Bend, Ind., in the period June 6 to July, 1920, 40 tons of red brass were melted at 371 kw. h. per ton in the 500-pound Booth furnace; 3.7 pounds electrodes were used per ton. The metal loss was estimated at 0.75 per cent.

In other plants operating the 500-pound furnace on yellow brass, 10-hour operation, the power consumption is stated to run close to 300 kw. h. per ton and the output around 2½ tons.

The usual lining life is given as 600 heats.

Booth⁵¹ has recently given details of the operation of 250 and 500 pound furnaces, but no data have been published on the operation of the 1-ton furnace.

Bragg⁵² stated that as far as he was able to see the 1-ton Booth furnace at the Michigan Smelting and Refining Co. "looked satisfactory, but he was not ready to speak with certainty at that time."

A Booth furnace is installed at the Bridgeport Brass Co., but this firm states:⁵³ "The Booth furnace was not a success, as the type of lining furnished did not stand up at all well."

The furnace is made for 110-volt single-phase current. The furnace is said to have been designed in capacities of 150, 250, 500, 1,000, 2,000, and 3,000 pounds. The transformer capacity for all sizes is not given, but that used for the 250-pound furnace is 75 kv. a.; for the 500-pound, 125; for the 1,000, 180; and for the 2,000, 300. The price of the 250-pound furnace, without transformer, primary switch, etc., was said to be about \$3,000 in September, 1919. The additional equipment necessary for a complete installation would cost several hundred dollars more.

⁵⁰ Booth, C. H., *Later type furnace*, Foundry, vol. 48, 1920, p. 466.

⁵¹ Booth, C. H., *Booth rotating electric furnace*, Metal Ind., vol. 18, 1920, p. 456.

⁵² Bragg, C. T., *Discussion*, Metal Ind., vol. 18, 1920, p. 447.

⁵³ Webster, W. B., personal communication, Aug. 24, 1921.

The furnace seems to be applicable in small sizes, but its degree of availability in large sizes has not yet been thoroughly shown.

AMERICAN REVOLVING FURNACE.

A later entrant into the field was the American indirect-arc furnace, designed by the American Metallurgical Corp. This furnace was first advertised⁵³ late in 1919. A description given by Ryan⁵⁴ shows that the furnace, though claimed to be an adaptation of the Weeks zinc furnace, deviates greatly from the original design of Weeks, following very closely indeed except in one particular the design of the Detroit rocking furnace.

Instead of the stationary electrodes of the Weeks furnace, the electrodes and supports are attached to the furnace shell and move with it. In the Weeks patent⁵⁵ a handwheel is shown, and the photograph illustrating Hansen's⁵⁶ paper on the Weeks furnace shows a motor, evidently reversed by hand, if at all, by which the Weeks furnace could be turned over now and then to interchange roof and hearth in order to obtain uniform wear of the lining as contemplated by Weeks for work on zinc. Ryan, however, provides a system "of automatic switches by which the drum can be rotated to any percentage of its circumference at the will of the operator, a special control, making it possible to set operation at any given point"; it is analogous to the control device previously used in the Detroit rocking furnace for automatic control of rocking and for varying from "safe rock" at the start to "full rock" at the end.

The furnace is also designed to be rotated completely if desired, although the charging door and pouring spout are on the circumference, and while a pouring spout can be plugged so as to be fairly safe against pressure of molten metal, it is difficult to see how a large charging door can be made safe against such pressure. One would also expect that metal would run around the door joint, solidify, and prevent the opening of the door.

The American furnace was designed for brass melting and aimed to prevent metal loss through stirring by rocking; the Weeks furnace was designed for zinc, never tried on brass, and when tried on copper was kept stationary during melting.

The American furnace design has the charging door and pouring spout on the circumference of the shell, like the Detroit rocking furnace, instead of on the ends, as in the Booth. It follows Booth in

⁵³ Brass World, vol. 15, 1919, October, Front cover.

⁵⁴ Ryan, F. J., Weeks' electric rotating furnace as applied to the brass industry, Metal Ind., vol. 17, 1919, p. 519. See also Acheson Graphite Co. Brass Melting (pamphlet) August, 1920, p. 29.

⁵⁵ Weeks, C. A., U. S. Patent 949,511, Feb. 15, 1910.

⁵⁶ Hansen, C. A., Electric melting of copper and brass: Trans Am. Inst. Metals, vol. 6, 1912, p. 110.

carrying the current by a sliding brush contact from motor brushes attached to the leads to a copper collector ring, carried on each end of the furnace instead of on rings attached to the circumference of the shell as in the Booth, thus avoiding the swinging cables of the Detroit furnace, but involving the use and maintenance of a sliding contact for a heavy current, and of a moving joint in the water connections. As in the Detroit and Booth furnaces, graphite electrodes were to be used.

An American furnace was ordered by the York Hardware and Brass Co., York, Pa., but was never delivered, and the Weeks patent, on which the design of the American furnace was stated to be based, which had been acquired by the American Metallurgical Corp., was next acquired by the Electric Furnace Construction Co., of Philadelphia, which stated in June, 1921, that it planned to put an electric brass furnace on the market.

The American furnace appears not to have been constructed or tested, and it is not known whether the Electric Furnace Construction Co.'s projected furnace is to follow the design of the American furnace or not.

MOORE FURNACE.

The "Moore Rapid 'Lectromelt Furnace" has been advertised by the Pittsburgh Electric Furnace Corp., but no detailed description of the exact construction has yet been given out, and no commercial installations have been made, though one experimental furnace is said to have been constructed. From the patent^{56a} it appears that the furnace is of the indirect arc type, the electrodes being mounted in a stationary roof, the body of the furnace being a cylinder set on end with its axis off the vertical. The body is revolved or oscillated about its axis. The revolving of the tilted furnace is to stir the charge. Like the Volta furnace, this does not wash the roof or much of the side walls with molten metal.

The advertisements⁵⁷ state, "Heat is transmitted to the charge by distributed radiation and reflection only. Arcs are struck between the electrodes rather than against the charge. Segregation and volatilization are minimized by tumbling the charge. They use polyphase or single-phase power."

The furnace is designed in 250, 500, 1,000, and 2,000 pound sizes. It is claimed to be especially adapted for melting brasses high in zinc.

^{56a} Moore, W. E., U. S. Patent 1,878,972, May 24, 1921.

⁵⁷ Metal Ind., vol. 18, February, 1920, advt. p. 10; June, 1920, p. 91; Elec. World, vol. 75, Apr. 24, 1920, p. 98.

REARDON FURNACE.

Still another rocking indirect-arc design has been described by Reardon⁵⁸ in which the oil burners of a Schwartz open-flame oil furnace are to be replaced by electrodes between which an arc is drawn, and the furnace is to be rocked mechanically on its trunnions. Reardon says that the design for the adaptation of the Schwartz furnace to electric heating was begun in December, 1913, but no figures on the performance of an actual furnace were available in August, 1921. On account of the location of electrodes, charging door, and pouring spout, only about half the lining can be washed by the metal.

The furnace is designed to take 300 kw. on a 1-ton size, at 220 volts. With as long an arc as 220 volts will give, and with the electrodes placed close together in the location of the burners, the treatment of the unwashed roof above the arc would be severe, and refractory troubles might be expected.

As the cost of the furnace shell is but a minor part of the total cost of an electric, it would appear wiser to the writers to use an electric furnace designed as such than to attempt to make over an oil furnace of this form into an electric furnace.

VOLTA FURNACE.

Although the original gyrating Stassano furnace was not applied to nonferrous melting,⁵⁹ the design has recently been resurrected, the furnace being provided with a mechanically simpler gyrating device, so made as to give the furnace a greater tilt and hence greater stirring than the original Stassano. The furnace is a three-phase, indirect-arc furnace with the electrodes 120° apart and slightly tilted downward from the horizontal. Provision is made for changing the angle of the electrodes, much as is done on the side electrodes of some Rennerfelt furnaces.

The stirring action is obtained by mounting (see fig. 32) between the base and the bottom of the furnace shell, a rotatable ring actuated by motor-driven gearing and resting on rollers. Several peaks of different heights are carried by this ring, each peak carrying a roller which runs underneath a track on the bottom of the furnace shell.

The shell has a projection carrying a roller, which runs up and down in a vertical, fixed track, thus preventing sidewise motion of the shell. As the ring revolves it alternately raises and lowers each point on the bottom of the furnace shell, giving a gyrating motion

⁵⁸ Reardon, W. J., Electric melting in an oil furnace, *Metal. Ind.* vol. 18, 1920, p. 207.

⁵⁹ Anonymous, Three-phase electric brass furnace of novel design: *Canadian Chem. Jour.*, vol. 4, 1920, p. 315.

to the whole furnace. Pouring is done also by use of the motor-driven ring.

The motion of the furnace is stopped when the spout is in its highest position, the tap hole is opened and cleaned, being then above the metal level, the ring is started in motion and the spout lowered, thus pouring the charge.

The roof is made in the form of a parabola, so as to reflect the heat from the arc directly down on the bath. It is claimed that since the three-phase arc covers a wider area than a single-phase arc of equal kilowatt, and the source of heat is therefore not so localized, that the gyrating motion gives sufficient stirring to avoid local overheating of the surface of the melt.

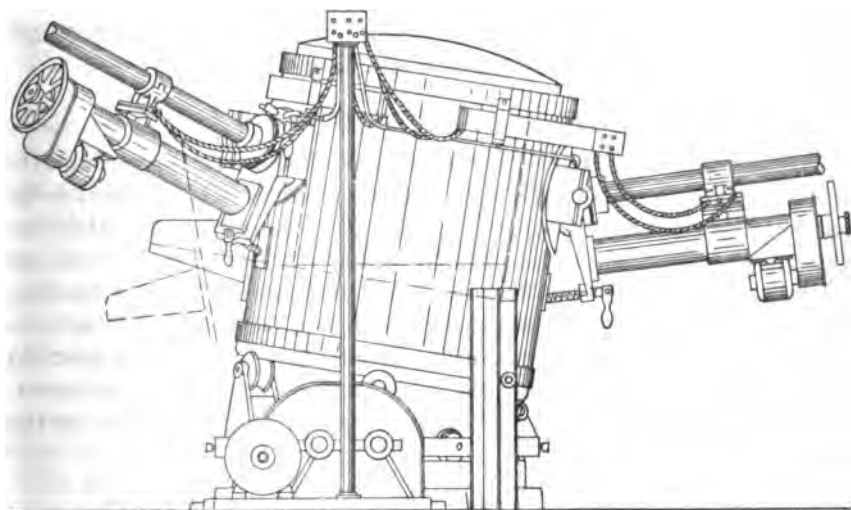


FIGURE 32.—Gyrrating Stassano-type furnace sold by Volta Manufacturing Co.

The charge is always below the electrodes so that the electrodes can not be broken by the charge falling on them. There is of course no gain in thermal efficiency due to cooling the roof to the temperature of the bath, such as is obtained in the rocking and rotating furnaces.

The furnace of a 1-ton size was briefly tested at the plant of the Titanium Bronze Co., Niagara Falls, N. Y., and it is claimed that "as low as" 225 kw. h. per ton of yellow brass and 300 kw. h. for aluminum bronze was obtained. These figures do not include any preheating and are calculated for continuous 24-hour operation. The metal loss is said to have been under 1 per cent; but it is not stated whether this was on yellow brass or aluminum bronze.

Data are lacking by which to fix definitely the efficiency of the metal losses on this furnace. If the stirring is adequate to avoid surface overheating, it would be expected to show metal losses of the

same order as those of the other moving indirect-arc furnaces. Its power consumption per ton would be higher than that of the furnaces in which the lining is thoroughly washed by the metal.

It would be useful in localities where a single-phase arc load is not acceptable to the central station company, but where a three-phase arc load can be handled.

The furnace is designed in 500 pounds, 125 kw.; 1,000 pounds, 175 kw., and 1-ton, 300 kw. sizes, and is sold by the Volta Manufacturing Co., Ltd., Welland, Ontario, Canada. The electrodes can be adjusted by hand or from the switchboard when motor control is used, or the furnace could be provided with automatic electrode control.

A demonstration furnace is installed at the Fitzgerald Laboratories, Niagara Falls, N. Y. One furnace each of the 500, 1,000, and 2,000 pound sizes are said to have been ordered, but delivery of all was deferred on account of the business depression.

ROCKING METALLIC RESISTOR FURNACE FOR ZINC.

Collins^{50c} of the General Electric Co. has designed a three-compartment furnace in which solid metal, as zinc, which is to be freed from dross, is charged into one part of the central compartment. Holes at the bottom of this compartment admit molten metal from the two end chambers, each of which is provided with metallic resistor heating elements. The furnace is mechanically rocked back and forth so as to wash the solid metal with superheated molten metal, thus melting the solid metal, but trapping the dross in the charging compartment. Clean molten metal may be ladled or tapped from another part of the central compartment.

SUMMARY ON MOVING INDIRECT-ARC FURNACES.

It is interesting to note that in addition to the makers of the furnaces built under the Bureau of Mines patents, seven other brass furnace designers—Thomson in his gyrating resistor furnace, Moore in the Pittsburgh Electric Furnace Co.'s rotating furnace, Booth in a rotating indirect-arc furnace, Ryan in the American rocking indirect-arc furnace, Reardon in his adaptation of the Schwartz oil furnace into a rocking indirect-arc furnace, and the makers of the Volta furnace, as well as Hervey in his furnace design—have taken up the principle of preventing local overheating of the surface of the metal through stirring by means of motion of the furnace while running a principle first applied to a commercial electric furnace by Stassano and worked out for electric brass melting by the Bureau of Mines. Collins also makes use of a rocking device in his zinc furnace. The value of the principle appears to be fully established. It

^{50c} Collins, E. F., U. S. Patent 1,378,526, May 17, 1921.

takes the indirect-arc type of furnace beyond the field to which the stationary indirect-arc furnaces are limited, that of alloys low in zinc, and thus produces a furnace suited metallurgically for use by plants that have to melt a large variety of alloys in the same furnace, showing not only a low power consumption per ton but low metal losses on a wide range of alloys.

Due to absorption by the moving charge before pouring of heat stored in the furnace walls, the temperature of the furnace walls and hence the radiation losses are reduced, and on account of the smaller heat storage in such a furnace than in furnaces of the reflected heat type, the moving indirect-arc type can be used on single-shift operation, without night heating, showing on such operation materially higher production and lower power consumption than do the types having a high heat storage.

The moving indirect-arc type is therefore suited especially for use by jobbing foundries, smelting and refining plants, and by plants operating only one shift per day. It is somewhat less simple to operate than stationary direct or indirect arc furnaces, than the reflected heat types, or than the vertical ring induction furnace. In flexibility, ability to operate on any alloy or any number of shifts per day, it surpasses all these other types. It holds its own on metal losses with any of the others in the field they can cover; in ability to operate at a low power consumption per ton, even in small sizes, it is surpassed only by the vertical ring induction type and possibly by the high-frequency current type. Its lining life and its electrical characteristics though not as satisfactory as those of some of the other furnaces are yet very good.

The main advances to be looked for in moving indirect-arc furnaces are in the addition of devices for rapid mechanical charging, to which those having the charging door on the circumference of the shell are well adapted, and for automatic control of the arc. Advances in the knowledge and use of better refractories for electric furnace use will improve the reliability of this as of all other types.

The principle of a moving indirect-arc electric furnace for brass has been accepted rapidly and widely, as may be seen by the following summary: The first test of such a furnace on brass was in August, 1915; the first semicommercial test in May, 1917; and the first commercial furnace started operation in August, 1918. On August 1, 1921, 100 rocking or rotating furnaces taking over 21,000 kw. were in operation at or being installed in nonferrous alloy plants in the United States; and half a dozen furnace makers or designers have incorporated the principle of moving an electric brass furnace while operating it into designs brought out since the experiments with the rocking furnace were published.

FURNACES IN WHICH THE METAL ITSELF IS THE RESISTOR.

In order to attain the highest thermal efficiency in an electric furnace the heat should be generated within the charge itself, not at a distance from the charge. As small a part as possible of the heat should be forced to the walls or roof before it is reflected back to the charge, nor should it be forced to pass through anything, like a crucible, which would baffle the flow of heat.

This principle of generation of heat within and in contact with the charge is the basic principle of efficient heating. It is seldom practical in fuel-fired furnaces, though it obtains in cupolas and blast furnaces, the most efficient of the type. It is more often possible to construct an electric furnace on this principle, which is carried out in all the most successful electric furnaces. Such electric furnaces as deviate from it do so only because electrical, metallurgical, or structural reasons make it impractical.

Electric furnaces for making graphite, carborundum, and carbon bisulphide generate the heat in or within the charge itself. Calcium carbide furnaces and ferro-alloy smelting furnaces usually operate either with the arc buried in the charge or with the charge as resistor.

Steel-melting furnaces which use carbon electrodes can not be so operated, but the direct-arc types of furnace, in which at least 95 per cent of the world's output of electric steel is melted, do the next best thing by generating the heat directly in contact with, though above, the charge.

Gin⁶⁰ tried to make a furnace in which a long, narrow channel between two electrodes was filled with the material to be melted, and heat was to be generated by the resistance of the charge itself. The furnace could not be operated even on steel, and as the resistance of nonferrous alloys is so much lower than that of steel, it would be even⁶¹ more impractical on brass than on steel. If an induction furnace is constructed by doing away with the electrodes and by making the charge the secondary of a transformer, steel can be melted by the generation of heat in the charge itself.

Inasmuch as the induction furnace allows the generation of heat in the steel itself, it might seem that induction furnaces would be the most logical type for use on steel. However, there is another factor entering in to affect its usefulness—the usual basic principle of an induction furnace shown in figure 33 and exemplified in the Colby-Kjellin,⁶² Frick, Roechling-Rodenhauser,⁶³ and other induction fur-

⁶⁰ See Stansfield, A., *The electric furnace* 1914, p. 247.

⁶¹ For brief description of the principles on which an induction furnace operates, see Clamer, G. H., *Melting brass in the induction furnace*, Jour. Am. Inst. Metals, vol. 11, 1917, p. 382.

⁶² Kjellin, F. A., U. S. Patent 682,088, Sept. 3, 1901.

⁶³ See Rodenhauser, W., and Schoenawa, J., *Trans. by vom Baur, Electric furnaces in the iron and steel industry*, 1917, p. 172.

naces demands that the steel be contained in one or more horizontal ring-shaped troughs, which form the secondary circuit. The ordinary induction furnace thus has a huge volume and huge wall space for a comparatively small metal capacity, and the electrical requirements make the type violate the second principle of thermal efficiency, that the furnace shall be as small as possible for a given capacity in order to avoid thermal leakage through excessive wall area. For this reason the thermal efficiency is not as good in a steel furnace of the horizontal ring induction type as in a direct-arc furnace. Its main theoretical advantage having been thus overbalanced, only its avoidance of the use of electrodes remains to compare with the disadvantages of inconvenience, low-power factor, and lower slag temperatures, which tend toward slower refining.

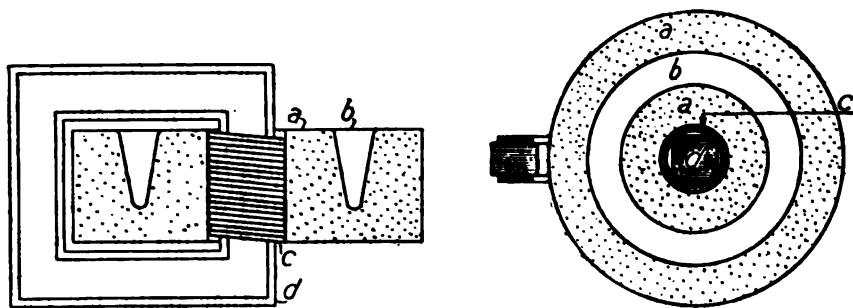


FIGURE 33.—Diagram of horizontal ring induction furnace: *a*, stamped lining; *b*, crucible; *c*, primary coil; *d*, core.

The induction furnace for steel melting in all countries with the possible exception of Germany has fallen back steadily till it almost can be classed as an obsolete type. The General Electric Co., however, has again attacked the problem of the induction furnace for steel and recently brought out a new modification, which is described by Unger and Scharschu.⁶⁴

HORIZONTAL RING INDUCTION FURNACES FOR BRASS.

Besides the disadvantages which have made the ordinary induction furnace of little value for steel, there is another which makes it inapplicable to copper, brass, or bronze. This is the "pinch effect," which has been described by Bary⁶⁵ and by Hering.⁶⁶

Briefly stated, the "pinch" phenomenon is due to the fact that currents flowing in the same direction through parallel conductors attract each other. This is the converse of the principle of the von

⁶⁴ Unger, M., and Scharschu, C. A., New type of induction electric furnace: *Iron Age*, vol. 108, 1921, p. 344.

⁶⁵ Bary, P., *La pulvérisation électrique des métaux: L'industrie électrique* No. 224, Apr. 25, 1901, p. 178.

⁶⁶ Hering, C., A practical limitation of resistance furnaces, the "pinch" phenomenon: *Trans. Am. Electrochem. Soc.*, vol. 11, 1907, p. 329; The working limit in electrical furnaces due to the "pinch" phenomenon: *Trans. Am. Electrochem. Soc.*, vol. 15, 1908, p. 255.

Schlegell "repelling arc," where the fact that currents flowing in opposite directions through parallel conductors repel each other is made use of to force the electrodes (which are pivoted so as to be free to separate) apart and start the arc. The current flowing in a single conductor also tends to attract the parts of a cross section of that conductor one to another. That is, a conductor through which a current is flowing tends to shrink and become more dense. Solid conductors can not yield to this force, but a liquid one, like mercury or molten brass, can do so.

If the conductor has exactly the same cross section throughout, the contracting forces are balanced and nothing happens, but if the cross section is smaller at one point, as can not fail to be the case in the secondary of an induction furnace through the dropping in, for example, of a piece of brick from the roof, the forces no longer balance at this point.

Now this force is greater the greater the current density, and it varies with the square of the current. At a point of slightly smaller cross section in the liquid resistor the current, then, if great enough, can exert so much force that it actually constricts the liquid as if it were contained in a sausage skin and a rubber band were put about it. As the liquid is constricted the current density gets higher and makes it still further constricted till it actually pinches the liquid off and breaks the circuit. The metal will then run together again, but the pinch force again makes it separate. At high current densities, as a result, the metal can not be kept together long enough for sufficient power to be supplied to keep it molten.

Materials of comparatively high electrical resistance at temperatures above their melting points, such as steel, platinum, and nickel and its alloys, can be melted in the induction furnace, because the voltage it takes to overcome their resistance is high enough that sufficient power kilowatts can be put into them to melt them and not exceed the current that can be used without making the pinch force exceed that of gravity.

But with materials of relatively low electrical resistivity at their melting point, such as copper, aluminum, brass, and bronze, the voltage required to overcome their resistance is very low, hence huge currents are required to generate enough heat to melt them, since power (watts) = volts $\times$ amperes $\times$ power factor. As the power factor in ordinary induction furnaces is low the current (amperes) must be high when the voltage or the electrical resistance is low. As the pinch force varies with the square of the current it turns out that copper, aluminum, brass, and bronze are all affected by the pinch force in a horizontal ring induction furnace, and that consequently it is impossible, or at least impracticable, to melt them in such a furnace.

This conclusion does not rest on theory alone. Fitzgerald⁶⁷ cites a case where aluminum could be just melted but not superheated at all because of the rupturing of the ring by the pinch force. By working with an extremely shallow ring of steel so that the force of gravity was not enough to overcome the pinch force, the phenomenon was duplicated with steel.

SPENCER FURNACE.

The same phenomenon was noted in an attempt to melt brass in a small laboratory induction furnace at the Massachusetts Institute of Technology, and in a test made at the Pierce Arrow Motor Car Co. several years ago in a furnace built by Mr. E. B. Spencer it was found that on nonferrous alloys of high conductivity the pinch effect was so great as to make the regular type of induction furnace impractical. The American Brass Co. had a similar experience in a trial of an induction furnace on brass.

HARDEN FURNACE.

Harden⁶⁸ suggests that for melting brass and aluminum the horizontal ring induction furnace should be built with a conducting ring-shaped crucible, as of fire-clay graphite mixture, to hold the metal, with the idea that by generating a large proportion of the power in the crucible the current density at which the pinch force starts to give trouble may not be exceeded.

GEHRKENS FURNACE.

Gehrrens⁶⁹ also suggests avoiding pinch-effect troubles on a horizontal ring induction furnace for brass or aluminum by having a conducting secondary other than the charge itself as a permanent part of the furnace.

SOLOMON FURNACE.

Solomon⁷⁰ patents a horizontal ring induction furnace in which the metal is contained in an annular crucible that can be removed to pour. The crucible would be excessively fragile, and the furnace was not designed especially for brass melting.

GREENE FURNACE.

Greene⁷¹ patents a horizontal ring induction furnace designed for melting copper, brass, bronze, and aluminum, in which the primary circuit is brought into the refractory lining close to the secondary, which is made up of the material to be heated. The primary is made a resistor, and the heat generated in it is transmitted through

⁶⁷ Fitzgerald, F. A. J., Discussion, Trans. Am. Electrochem. Soc., vol. 15, 1909, p. 278.

⁶⁸ Harden, J., U. S. Patent 977,303, Nov. 29, 1910.

⁶⁹ Gehrrens, E. F., assignor to General Electric Co., U. S. Patent 1,011,769, Dec. 12, 1911.

⁷⁰ Solomon, H. G., U. S. Patent 1,081,164, Dec. 9, 1913.

⁷¹ Greene, A. E., U. S. Patent 1,078,619, Nov. 18, 1913.

the refractory lining to the hearth or secondary. Both primary and secondary thus are to be sources of heat. This principle practically amounts to an attempt to run a brass furnace with a metallic resistor, the primary, as one of the main sources of heat. No metallic resistor of suitable physical properties is available at a price that would allow it to be used in a commercial brass furnace. The furnace does not appear to have been constructed or tested.

KJELLIN FURNACE.

Rowlands⁷² cites a test on brass of a Kjellin furnace provided with a conducting crucible in which the pinch effect was overcome. He states also that in a three-phase furnace without a conducting crucible the pinch effect was less intense than in a single-phase. Brass is said to have been melted in the three-phase furnace at 165 kw. h. per net ton, a very low figure.

Arnold⁷³ stated in 1910 that tests were to be made at Sheffield University on a comparison of melting brass, bronze, and German silver by coke, and in a Kjellin induction furnace; but, as no report of such tests has been made other than that by Rowlands, above cited, it may be assumed that the operation of the furnace on those alloys was found impractical.

ROECHLING-RODENHAUSER TYPE.

The Baltimore Copper Smelting and Rolling Co. tried several years ago to use an induction furnace of the Roechling-Rodenhauser type for making brass, the idea being to run in molten copper from a fuel-fired furnace, add zinc, and put in enough power to hold the alloy molten for pouring. The main purpose was to make a heated mixer rather than a melting furnace. It was thought that so little heat would have to be added that enough power could be put in without the current density exceeding that which would cause rupture. Various troubles with the furnace caused the abandonment of the idea before the experiments had gone far enough to show whether the pinch force would make the furnace inoperable.

Vom Baur⁷⁴ says that he has been told that brass has been melted at the Roechling Eisen und Stahlwerke, in a three-phase Roechling-Rodenhauser furnace, using pole plates,⁷⁵ without interference from pinch troubles.

On the other hand, Dr. K. G. Frank, formerly American agent for Siemens and Halske, in response to an inquiry, informed one of

⁷² Rowlands, T., Induction furnace progress, Trans. Am. Electrochem. Soc., vol. 17, 1910, p. 103.

⁷³ Arnold, J. O., Department of nonferrous metallurgy at Sheffield University, Eng. Supp. London Times, Oct. 26, 1910. Jour. Soc. Chem. Ind., vol. 29, 1910, p. 1254.

⁷⁴ Vom Baur, C. H., Personal communication.

⁷⁵ See Rodenhauser, W., and Schoenawa, J., Trans. by vom Baur, C. H., Electric furnaces in the iron and steel industry, 1917, p. 215.

the writers in 1915 that the induction furnace then being made by Siemens and Halske as a combination of the Roechling-Rodenhauser, Frick, and Kjellin induction furnaces, was not deemed suitable for brass because of troubles from the pinch force. This statement appears to cover the present situation of the horizontal ring induction furnace.

HOWARD FURNACE.

Howard⁷⁶ suggests a furnace designed for melting copper, brass, and aluminum, in which a shallow bath of metal forms part of the secondary of an induction furnace, the rest of the secondary being a heavy U of conducting material, as of copper, placed below the bath, each leg of the U having a tip of graphite or of water-cooled steel which makes contact with the bath. The loss of heat in such a solid portion of the secondary would be terrific, and it is very doubtful if a furnace so constructed could be made to operate at all.

HERING FURNACE.

Having discovered that the pinch force spoiled all possibility of the ordinary induction furnace being made to operate on brass, Hering set to work to tame this force and to utilize it in the construction of a brass furnace. By opposing the pinch force with a sufficient hydraulic head of molten metal, it is obvious that the metal can be kept from rupturing. Hering found that not only was this the case, but that if the bottom of the tube containing the molten resistor was closed by a solid material, the pinch force caused the tube to act as a pump, drawing cool molten metal down the outside of the tube, heating it by its own resistance, and ejecting it up the center of the tube. This squirting action produces a little fountain of molten metal that is continually taking in cool metal and throwing out hotter metal. All the heating is done in the tube, but it is possible to heat up a large volume of metal because of the circulation.

The Hering furnace has been fully described in the literature.⁷⁷

⁷⁶ Howard, L. E., U. S. Patent 1,076,887, Oct. 28, 1913.

⁷⁷ Hering, C., U. S. Patents 988,986, Apr. 9, 1911; 999,720, Aug. 1, 1914; 1,105,656, Aug. 4, 1914; A new type of electric furnace: *Trans. Am. Electrochem. Soc.*, vol. 19, 1911, p. 255; Electric furnaces for molten materials: *Met. and Chem. Eng.*, vol. 9, 1911, p. 371; Ideals and limitations in the melting of nonferrous metals: *Jour. Inst. Metals (British)*, vol. 17, 1907, p. 243; Advantages of a small high-speed electric furnace: *Met. and Chem. Eng.*, vol. 11, 1913, p. 183; Possible reduction in the power consumption in electric steel-refining furnaces: *Met. and Chem. Eng.*, vol. 9, 1911, p. 590; Clamer, G. H., The Hering electric furnace for commercial brass melting: *Trans. Am. Inst. Metals*, vol. 7, 1913, p. 350; Clamer, G. H., and Hering, C., Electric brass melting: *Trans. Am. Inst. Metals*, vol. 8, 1914, p. 270; The electric furnace for brass melting: *Trans. Am. Inst. Metals*, vol. 6, 1912, p. 95; Lyon, D. A., and Keeney, R. M., Electric furnaces for making iron and steel: *Bull. 67, Bureau of Mines*, 1916, p. 87; see also *Elektrochemische Werke*, German Patent 309,973 of Dec. 8, 1915.

After development on a laboratory scale, the furnace was built on a commercial scale, with a total capacity of about 1,500 pounds, of which 500 to 750 pounds was retained from one heat to another. Furnaces were tried out by the National Cash Register Co., the Scoville Manufacturing Co., and the Ajax Metal Co. The first of these large furnaces, that at the National Cash Register Co., was tried out on April 30, 1914, one of the writers being present. There appeared various transformer and other troubles incident to the design of a large furnace of a new type previously operated only in

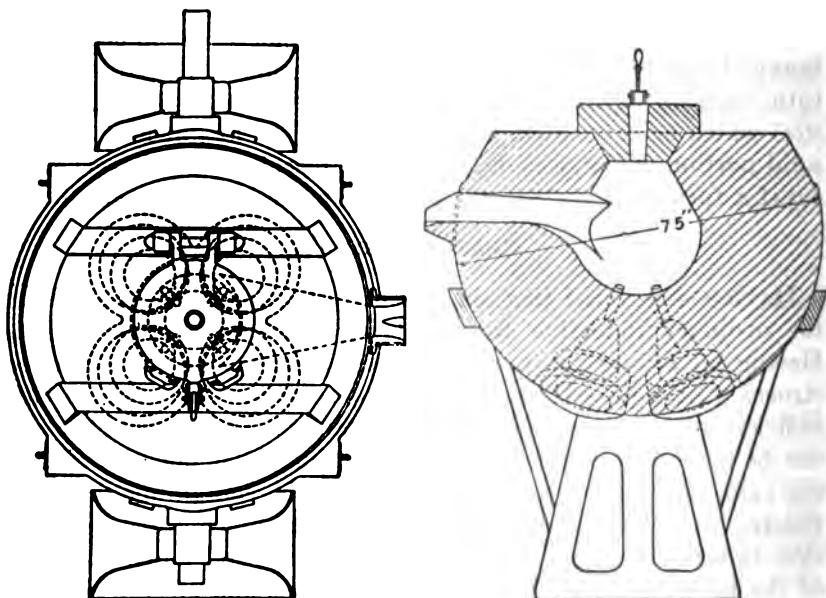


FIGURE 34.—Plan and elevation of Hering resistance furnace.

a much smaller size, so that the furnace could not be made to melt metal at that time. Some further work was done with it, but it did not do any commercial melting.

The furnace at the plant of the Scoville Manufacturing Co. melted a good deal of metal, but was finally discarded. The results of the tests at that plant are not available, save that the metal loss on yellow brass was well under 2 per cent, and a loss of one-half of 1 per cent was considered attainable.

The furnace at the Ajax-Metal Co., shown in figure 34 and Plate XXI, *B*, (p. 235) was operated commercially for some time. One of the writers was present in December, 1914, while this furnace was in operation. The furnace was built in the shell of a Schwartz open-flame furnace, which was approximately spherical and about 6 feet in diameter. The lining was about 18 inches thick and consisted of fire brick with an inner rammed lining of high temperature asbestos cement. Inasmuch as the resistor tubes can not operate unless they

are filled with molten metal, the furnace must be started by pouring in enough molten metal to fill the tubes and provide also sufficient hydraulic head to oppose successfully the pinch force. In order to preheat the empty furnace when starting up, and to prevent freezing to the starting charge of molten metal, the oil burners of the Schwartz furnace were left in place.

Inasmuch as the available power supply was two-phase, four-wire, four-resistor tubes were provided. These tubes were placed vertically in the bottom of the furnace and were formed by putting tubes made of the regular crucible mixture of graphite bonded with fire clay in place and ramming up the bottom around them. The electrodes themselves were heavy copper blocks set at the bottoms of the resistor tubes and were arranged to be water-cooled by a copious water supply. The electrodes were approximately 5 inches thick, and were in contact with the molten charge in the tubes above and with the cooling water below. The flow of cooling water had to be kept up in sufficient volume to prevent the copper blocks from melting into the charge, so that the molten charge could not run down into the water and produce a disastrous explosion.

It is obvious that a fin of metal entering a crack in the bottom of the hearth and extending from one electrode to another in the same phase would bridge across between the electrodes and short-circuit the current out of its proper path. The graphite fire-clay tubes were used to prevent this contingency, as they were less liable to crack than the lining itself. The whole bottom could not well be rammed up of crucible mixture, as it would itself conduct the current and would be likely to short-circuit it.

The cylindrical resistor tubes shown in figure 34, which were originally used, had been replaced at the time of the test by funnel-shaped resistors $1\frac{1}{2}$ inches diameter for the first $2\frac{1}{2}$ inches from the electrode and then flaring up from $1\frac{1}{2}$ inches diameter to $2\frac{1}{2}$ inches diameter at the hearth, which was 12 inches from the electrodes. These tubes allowed a freer circulation of metal than plain cylindrical tubes.

As only an extremely low voltage is required or can be used, the current carried is very heavy. The voltage drop per resistor tube was around 3 volts, the amperes per phase being around 11,000. It is on account of the high current and low voltage required that carbon or graphite electrodes could not be used; had they been used, their resistance would have been so great in comparison to the resistance of the resistor that the heat would have been generated mainly in them instead of in the tubes. The water-cooled copper electrodes were therefore inevitable.

These heavy currents made it imperative for the leads from the low voltage transformer to be very short in order to avoid heat losses in them and to avoid a low power factor. The transformer was

therefore built integral with the furnace, being attached to the bottom of the furnace shell, and tilting with it when the furnace was tilted to pour.

The transformers were designed for a total power input of 120 kw., but could not be operated at more than 100 kw. without dangerous heating, even though they were regularly air cooled by blowing air on them by electric fans. The joints or connections in the secondary circuit between transformer terminals and the heavy leads going to the electrodes had to be packed with waste, and the waste wet with water every 15 minutes to prevent overheating of the joints. Some of the transformer difficulties might have been reduced by further experiments, but the design of a transformer for such low voltages and high currents, where lightness is desirable and where a high-power factor is needed, is a difficult problem. The transformer is essentially one of the weak points in a furnace of this design.

Before the test the furnace had been operated for two full days, 24 hours a day, and was therefore fully heated up. The material melted in the first part of the test was manganese bronze ingot, containing about 40 per cent zinc, the ingots weighing about 30 pounds each.

The furnace held about 1,500 pounds of metal, but only about 625 pounds, or one ladleful, was poured at a time, the rest being retained in the furnace. Zinc and aluminum to bring the ingot to the desired composition were added in the ladle.

In the test 15 heats of around 625 pounds and 2 heats of around 300 pounds, 9,978 pounds total, including ladle additions, were melted in 21 hours 20 minutes, continuous running, the output thus being about 450 pounds per hour. The power used was 1,797 kw. h., at the rate of 396 kw. h. per ton, the metal being poured at an average of 1,010° C. (1,850° F.). The average power input was 93 kw. The power factor averaged 80 per cent.

On the total run of manganese bronze ingots 25,257 pounds of clean metal were charged and 24,479 pounds of metal obtained. The skimings contained 428 pounds of recoverable metal. The gross loss was thus 778 pounds, or 3.08 per cent, and the net loss 350 pounds, or 1.42 per cent. The furnace had been given a new rammed-up asbestos lining before this run, and there was doubtless some soakage into the hearth.

The circulation of the metal in the tubes was very rapid; when the metal level was low, after pouring, the surface of the molten metal over each tube was decidedly raised through the ejection of a stream of metal. By sticking the end of a pyrometer with a small tip on the protecting tube⁷⁸ into the resistor tubes, the temperature of the

⁷⁸ See Manning, Van, H., Year book of the Bureau of Mines for 1916: Bull. 141, Bureau of Mines, 1917, p. 106.

stream of molten metal coming out of the center of the tube was found to be only about 20° C. (35° F.) higher than that of the body of the melt.

After the run on manganese bronze ingot, the furnace was operated on "irony brass" during the visit of one of the writers. A bath of yellow brass was melted and then a charge was fed in consisting of yellow brass that could not be readily separated from the iron it contained. The charge included old alarm clocks with some brass wheels on steel axles, steel shotgun shells with brass heads, and flexible tubing with a steel inside and brass outside, which would normally be fed to a cupola and only the copper recovered.

The charge usually floated on the surface of the melt, and as soon as the brass had melted the iron and dirt was rabbled or skimmed out. Aside from dirt the charge was probably 70 per cent iron, which was, of course, heated up as well as the brass, and only about 30 per cent brass. This material had to be charged and the iron taken out continually; therefore the furnace could not be tightly closed, as it had been on the manganese bronze run, and the cover was open most of the time. Some zinc was thus lost by volatilization.

It took about 2½ hours to melt 550 pounds of brass off of, say, 1,000 pounds of iron. The power consumption was about 800 kw. h. per ton of brass poured or 275 kw. h. per ton of junk charged. The temperature was kept down to about 1,010° C. (1,850° F.) to minimize the solution of iron. The metal obtained contained nearly all the zinc of the original brass, and though it contained some dissolved iron, it was worth much more than the copper obtained by cupola melting would have been worth.

The power consumption in the manganese bronze run on 24-hour operation of about 400 kw. h. per ton, on an alloy heated only to 1,850° F., was not as good as might be expected from a furnace generating heat within the metal itself, so that it was desirable to find out where the lost heat was escaping. There was no excessive loss of heat through the furnace walls, as the hand could be held on the shell after a week's steady operation. The furnace had been in steady operation for several days when the test was made, therefore no heat could be applied to heat storage in the walls. The only avenues for large loss were from the transformer and the electrodes.

It was necessary to keep a couple of electric fans blowing on the transformers and to keep wet waste on the connections between transformer taps and leads; a notable loss, which could not be measured, was evidently going on there. The greatest loss was in the cooling of the copper electrodes to keep them from melting.

By measuring the rate of flow of the electrode-cooling water and by taking the temperature of the incoming and outflowing water, it was found that, with the furnace operating at a little under 100

kw., $46\frac{1}{2}$ kw.—109.5 pounds water per minute, at 24° F. temperature rise—and $48\frac{1}{2}$ kw.—93.6 pounds water per minute, at $29\frac{1}{2}^{\circ}$ F. temperature rise—or an average of about $47\frac{1}{2}$ kw., was lost in the electrode-cooling water. As several kilowatts were evidently lost in the transformer, at least half the power supplied was used in heating air and water required to keep transformer and electrodes in operation, leaving the other half to heat the brass and the furnace.

The furnace melted 450 pounds per hour. At the theoretical figure taken by Clamer and Hering⁷⁹ for melting 2:1 brass, 6.9 kw. h. per 100 pounds, the power usefully applied to melting brass was 31 kw. The heat balance then was evidently about as follows:

TABLE 51.—*Heat balance of Hering furnace.*

Application of heat.	Kw.
Usefully employed in melting-----	31
Lost in cooling electrodes-----	$47\frac{1}{2}$
Lost in cooling transformer and leads, approximately-----	$5\frac{1}{2}$
Lost through radiation from furnace shell, approximately----	16
Total-----	100

The Ajax Metal Co. made many attempts to improve the electrodes. Copper electrodes of various shapes were tried to no avail. Efforts were then made to utilize an electrode of higher melting point to require less water-cooling. Attempts were made to use steel electrodes with tips insoluble in brass, the tips being welded on or cast into the steel. Ferrochrome as a tip dissolved somewhat in the bath. Ferrotungsten did not, but gave trouble from cracking. The nearest approach to a satisfactory electrode was a tubular steel electrode with a graphite tip, but it did not last long, as the brass soon worked back of the graphite. Even with this change, 25 per cent of the power supplied was lost in the cooling water and the best figures on power consumption were 360 to 335 kw. h. per ton on continuous operation. Continual trouble was met through cracking of the furnace bottom and the seeping-in of metal, forming fins that sooner or later short-circuited the electrodes.

After long and patient experimental work the Hering furnace was abandoned by the National Cash Register Co., the Scoville Manufacturing Co., and the Ajax Metal Co., and has not since been tried out further. No other electric furnace suggested for brass melting that is not now in commercial use had such a thorough trial as the Hering.

Dr. Hering is understood to contemplate the test of a modification of his furnace, designed especially for steel, in which the furnace is

⁷⁹ Clamer, G. H., and Hering, C., The electric furnace for brass melting: Trans. Am. Inst. Metals, vol. 6, 1912, p. 95.

to have two hearths connected by a refractory wall in which is provided a hole connecting the hearths, the molten metal in this hole acting as the resistor. The low-voltage current is to be led into each hearth by many relatively small wires or rods—of iron for a steel furnace, of copper for a brass furnace. Whether thus subdividing the electrodes and embedding them in the refractory end walls of the furnace will allow operation without excessive heat loss can only be determined by trial.

Quesneau⁹⁰ has suggested a furnace in which the pinch effect was allowed to make and break the circuit continually in a resistor tube, the heating being done either wholly by the resistance heating in the tube or by having such a tube in the same circuit with an arc in order to stir the charge. These furnaces unquestionably never ran, for all experience shows that a furnace in which the pinch force is causing continual disruption of the current refuses to take enough power to do work and becomes inoperable; moreover, it is entirely out of the question to combine an arc and a pinch-effect resistor in the same circuit, as the current and voltage requirements of the two are widely different.

If some form of the pinch effect on a separate circuit could be utilized as an electromagnetic pump, not primarily as a source of heat, to stir the metal in a direct or indirect arc furnace, it might serve to decrease surface overheating; but no satisfactory form has been suggested, and other means of stirring appear cheaper and more convenient.

The main trouble with the Hering furnace, then, lies in the use of solid electrodes which have to be water-cooled to keep them solid and, in this water-cooling, use too much power. The next step obviously is to eliminate the electrodes by joining two resistor tubes together and by making them form the secondary of an induction furnace with the metal in the hearth, but retaining the use of the hydraulic head of the molten charge to oppose the pinch force and prevent rupture.

This suggested change brings the design to that of a vertical ring induction furnace that may be visualized by thinking of a signet ring with the signet held uppermost. The signet represents the body or hearth of the furnace, which contains the bulk of the charge, while the circlet of the ring forms the resistor loop or secondary of the induction furnace. The loop may, of course, be at any angle or of any shape as long as it is below the body of the bath, that hydraulic head may oppose the pinch force.

⁹⁰ Quesneau, A. L. J., U. S. Pat. 948,848, Feb. 8, 1910. A new electric steel furnace, *Trans. Am. Electrochem. Soc.*, vol. 17, 1910, p. 131.

VERTICAL RING INDUCTION FURNACE

Various forms of vertical ring induction furnaces have been suggested, and a few of them tried out, for melting steel, or for use in the hearth of a furnace for ore reduction, but melting brass in them was not suggested when they were brought out. Schneider⁸¹ first described a furnace of this type, the patent description being fairly simple and closely allied to later successful forms of this type. The Schneider furnace was built and tested at the Creusot works in France in a rather more complicated form. Guillet⁸² states that the circulation in the inclined resistor channels of the Schneider furnace was very rapid and ascribes it to the difference in density between hot and cool metal. The writers have experimented with copper and brass contained in tubular containers like those of the Hering furnace, highly heated at the bottom and comparatively cool at the top. The heat was developed outside the containers and not in the metal itself in order to eliminate pinch effect and to leave only the thermal circulation. The motion could hardly be noted and was not in the least comparable to the violent motion in a resistor in which the pinch force is acting. It is plain that circulation due to pinch force was obtained in the original Schneider furnace, though it was not recognized as such at the time. The idea that circulation was caused by differences in the density was taken up by Gin,⁸³ who designed a furnace, with the cooperation of Schneider, in which two hearths were connected by tubes below the metal level, but it does not appear to have had much, if any, commercial success. There have been some later developments of this type for steel melting, since a Saladin-Schneider is listed among induction furnaces, presumably operating according to a recent article by Escard.⁸⁴

The rights to the Schneider patents for the United States have been purchased by the Ajax Metal Co.

German patents to Wallin⁸⁵ and to Vereinigte chemisch-metallurgische und metallographische Laboratorien⁸⁶ describe modifications of the vertical-ring idea, but do not appear to have been brought to any commercial application.

Snyder⁸⁷ patents a form of vertical ring induction furnace in which part of the ring may be a solid or liquid conductor, the fur-

⁸¹ Schneider, C. P., U. S. Pat. 761,920, June 7, 1904; 763,830, June 21, 1904; Clamer, G. H., Melting brass in the induction furnace: Jour. Am. Inst. Metals, vol. 11, 1917, p. 885.

⁸² Guillet, L., Les alliages metalliques, 1906, p. 70.

⁸³ Gin, G., U. S. Pat. 875,801, 1908. Electrometallurgy of the rare metals, Trans. Am. Electrochem. Soc., vol. 12, 1907. The self-circulating Gin furnace for the electric manufacture of steel, Trans. Am. Electrochem. Soc., vol. 15, 1909, p. 205.

⁸⁴ Escard, J., Induction furnaces for making steel, Rev. gen. l'electricite, Dec. 18, 1919, abstracted in Iron Age, 105, 1920, p. 606.

⁸⁵ Wallin, N., German Patent 183,622, Mar. 4, 1907, application Oct. 9, 1904.

⁸⁶ Vereinigte chemisch-metallurgische und metallographische Laboratorien, G. m. b. H., in Berlin: German Patent 224,877, Aug. 3, 1910, application Nov. 27, 1907.

⁸⁷ Snyder, F. T., U. S. Patents 825,859, July 10, 1906; 859,184, 859,187, July 2, 1907.

nace being designed for smelting zinc. Copper is mentioned as the material of the solid conductor, but it would be dissolved by molten zinc in contact with it.

Crafts⁸⁸ described a somewhat similar design. Neither the Snyder nor Crafts induction furnaces were designed for brass melting nor do they appear to have gotten to the commercial stage.

FOLEY FURNACE.

A single-phase vertical-ring induction furnace, intended for brass melting, was built by C. B. Foley⁸⁹ at the plant of the National Cash Register Co. in 1913-14, and the first attempt at running it was made on January 20, 1914, in the presence of one of the writers.

The furnace consisted of a hearth 5 inches wide, 14 inches long, 10 inches deep. Connecting with each end of the hearth was a segment, about 260°, of a circle of about 18 inches diameter. This segment had a circular cross section of 2 inches diameter. To provide a conducting secondary to start the furnace a copper ring, 1 inch in cross-sectional diameter, was placed inside a wooden ring, the outside diameter of whose cross section was 2 inches. This wooden ring was rammed up inside the furnace to form the secondary tube. The lining of bonded magnesite was rammed up inside the furnace shell above this wooden form and about another one in contact with the first to form the hearth. After the magnesite had dried the hearth form was removed, and the current turned on to heat the copper ring and to char the wood of the resistor form; then an air blast was turned onto the wood and the wood burned and blown out, leaving the 1-inch diameter copper ring loosely hanging in the 2-inch diameter secondary loop. One leg of the rectangular, laminated core on which the primary coils were wound passed through a hole in the center of the furnace, thus being within the resistor loop.

The first thing to find out about this type of furnace was whether there would be the same sort of circulation in the Foley resistor loop as in the Hering resistor tubes. Hitchcock⁹⁰ had stated that in order to get a circulation due to the pinch force there must be a solid end in such a resistor, and Hering⁹¹ had said that in a single resistor tube with two outlets there would be a squirting of the metal from both ends of the tube, but a stagnant place in the tube "where the metal would not be circulating, and hence would immediately overheat and vaporize."

In order to test this statement and to get some information out of the furnace before running the risk of spoiling it by trying to use

⁸⁸ Crafts, W. N., U. S. Patents, 1,069,923, 1,069,924, Oct. 12, 1913; 1,070,017, Aug. 12, 1913.

⁸⁹ Foley, C. B., English Patent 114,853 of Apr. 13, 1916.

⁹⁰ Hitchcock, H. K., Discussion, Trans. Am. Electrochem. Soc., vol. 19, 1911, p. 267.

⁹¹ Hering, C., Discussion, Trans. Am. Electrochem. Soc., vol. 19, 1911, p. 269.

it on brass, it was decided to run on lead and to see whether the circulation desired was obtained. So the copper ring was brought to redness and the lead was fed in. The furnace took 8 to 8½ kw. with a primary voltage of 450 to 460 and an amperage of 41 to 44, the power factor being 23 per cent. One of the four primary coils was then cut out, the primary voltage being then 458, amperage 76, the kw. 17.4, and the power factor 50 per cent.

As soon as enough lead had been melted by the hot copper ring to fill the vacant space in the resistor tube and to establish the connection of molten lead in the hearth between the ends of the resistor tube, a violent circulation was set up, a little fountain of lead being formed over each opening into the resistor tube, just like the action of the Hering resistor tubes in small Hering furnaces that had been seen working on lead. The pumping of the lead out of the resistor was violent enough to force out or float out the nails that had been used in fastening the wooden form used for the resistor tube in the rammed-up lining.

In 13 minutes the furnace melted and raised to 525° C., 102 pounds of lead, the furnace operating at an average of 17.5 kw., using 3.8 kw. h. and having a power factor of 58.

The furnace had not been provided with a tilting mechanism, so some of the lead was ladled out, but the resistor tube was left filled with it and the current was cut off, the lead freezing in the tube.

On turning on the current and melting the solid lead it was found that a crack had been made in the magnesite lining, as lead began to drip out at one point. The leak was plugged up and the run continued. About 203 pounds solid lead left in the furnace was melted and raised to 425° C. in 40 minutes, using 12.5 kw. h. One hundred and three pounds more lead was then charged, melted and raised to 425° C. in 11½ minutes, using 3.5 kw. h.

The lead was again ladled out down to the copper ring and an attempt made to melt the ring. Before it melted, however, it became soft, and a piece broke out, thus breaking the secondary circuit. About 50 pounds of molten red brass was poured in to restore the circuit, and as the furnace was only taking 17 to 18 kw., another of the primary coils was cut out. The furnace then took 43 kw. at 440 volts, 163 amperes on the primary, or a power factor of 73. The copper ring melted and the furnace operated for a few moments on an alloy made up of lead, copper, and red brass; that is, it might be considered a heavily leaded red brass. The circulation with brass was the same as had been shown with lead, and nothing out of the way occurred that could be ascribed to a "dead spot" in the resistor.

The leak that had been successfully plugged when lead only was melted began again and the furnace could no longer be operated.

The run had indicated, however, that a pinch-force circulation could be obtained in a vertical-ring induction furnace. It had also shown that a charge should not be allowed to freeze in the resistor tube of such a furnace.

Work on the Foley furnace was not carried further at the National Cash Register Co., but was later resumed at the plant of the Bristol Brass Co. In March, 1917, one of the writers visited this plant and saw a 60-kw. furnace of about 1,000-pound capacity, capable of pouring some 700 pounds per heat. The furnace was not in operation, as it was undergoing alteration. Material was on hand for three larger furnaces of about 3,000 pounds capacity, but construction had not been started.

The experimental furnace had, at that time, a secondary in the form of a flat tube, about 6 inches wide by three-fourths inch thick instead of the circular one of the first furnace, the change being made to cut down magnesite leakage. The resistor loop is circular, not having the sharp angle of the loop in the Ajax-Wyatt furnace, which will be described later. The resistor is made by ramming up two large blocks from a mixture of ground old crucibles, bonded with fire clay and water glass. These blocks are allowed to dry slowly and a depression corresponding to half the resistor loop is cut in each. Cement is put on the faces of the two blocks and the two put together, thus completing the casing of the resistor loop. The two halves of the casing are held together in a heavy iron frame and tightly squeezed by screwing the frame together. The assembled casing is then similarly cemented and fastened tightly to the hearth by rods and turnbuckles, thus assembling the furnace. The hearth has a brick lining, with thick walls, about 15 inches being used.

The furnace at this time had a power factor of only 58, owing largely to the distance between secondary loop and core, this distance being about 8 inches—5 inches of refractory and 3 inches of air space. Water cooling of the refractory walls within the secondary and outside the core has since been introduced, and should allow cutting down this space and therefore lower somewhat the magnetic leakage and raise the power factor.

This furnace was built to take 2,300 volts direct on the furnace primary, which the writers consider unsafe. Mr. Foley states that he can use 220 or 440 volts just as well, as 2,300-volt furnace being so designed to save the cost of a transformer for stepping down from that voltage to whatever voltage is chosen for the furnace primary.

The furnace was designed to pour through a tap hole from beneath the surface of the metal, the flow being controlled by a graphite plug. It was mounted on a wheel-and-track system for pouring direct into molds.

No detailed data on actual operation of the Foley furnace are available, though some melting has been done. A power consump-

tion of 225 kw. h. per ton on 70:30 brass is claimed, this being taken with the furnace hot, as it would be in 24-hour operation.

A copper ring is put into the resistor loop when the furnace is assembled and in starting up this is heated by the current and zinc or fed in till the loop is filled, or molten zinc is poured in; then brass, and, finally, copper are fed in till the charge has the desired composition.

In February, 1918, Mr. Foley offered to install for a prospective customer a single-phase 125-kw. furnace, capable of melting 1,000 pounds of yellow brass per hour at not over 250 kw. h. per ton on continuous operation. In fact, it was stated that pure copper could be melted at that rate, and that it was possible to hold the metal molten over night when working one shift only, with a power input of 12.5 kw. The primary voltage would be 220 or 440, as desired, the secondary voltage 8 to 10. No statement was made as to power factor, save that it was "high." The life of the lining on continuous 24-hour operation was said to be 2,000 heats, or 1,000 tons. The same furnace is said to be applicable to brass, bronze, and aluminum. There are no data available on actual runs in this furnace to show whether the figures given the prospective customer have or have not been attained in actual operation. The Bristol Brass Co., at whose plant most of the experimental work on the Foley furnace has been carried out, stated²² that the furnace was not yet on a truly commercial basis. Nothing has been heard as to any later development.

Brophy²³ has patented a still different form of vertical ring induction furnace, but no actual application of it is known.

AJAX-WYATT FURNACE.

The only vertical ring induction furnace in wide commercial use in brass melting is the Ajax-Wyatt, developed by G. H. Clamer and J. R. Wyatt, of the Ajax Metal Co., after their experience with the Hering furnace had shown that a furnace with the metal circulation of the Hering was desirable, but that the electrode losses of the Hering should be eliminated. It has been the result of a long series of painstaking experiments. The first commercial Ajax-Wyatt furnace is shown in Plate XXI, A, p. 235. The 30-kw. furnace had almost its present form in October, 1915, and the 60-kw. in the summer of 1917. Though the furnaces were in constant use at the plant of the Ajax Metal Co., and a number were supplied to several other firms desiring to get an early start in the use of the furnaces, it was not until the fall of 1919,²⁴ by which time the design of the furnace had

²² Cartnell, Wm., Personal communication, Jan. 22, 1920.

²³ Brophy, O., U. S. Patent 1,296,752, Mar. 11, 1919.

²⁴ Metal Ind., vol. 17, November, 1919, p. v. See also 60 Ajax-Wyatts in operation. Review of Electric Furnace Industry, Elect. World, vol. 75, 1920, p. 161.

been standardized and full information collected on its behavior, that the furnace was advertised for general sale. The conservative attitude of the designers of this furnace is shown also by their frank acceptance of the limitations of this type and their refusal to supply it for use under conditions to which it is not fitted.

The Ajax-Wyatt furnace has been described by Wyatt,⁹⁶ Clamer,⁹⁶ Blakeslee,⁹⁷ and Kenyon.⁹⁸ It is built under the Wyatt patents under licenses as to the Schneider, Kjellin, and Hering patents.⁹⁹

To produce circulation of the metal, this furnace utilizes not only a slight amount of thermal or gravity circulation, due to heating the metal from the bottom, and some "pinch" force, due to the high current used, some 10,000 amperes at about 4 volts, in the secondary of the 30 to 40 kw. size and at about 6 in the 60-kw., but also a "motor" force.¹ This latter force is due to the attraction or repulsion between two conductors respectively carrying current in the same or in opposite directions. The "pinch" force in any conductor is due to the current in any single part of that conductor alone; the motor force is due alone to the current in another conductor or to another part or branch of the same conductor. When the conductors are molten the motor force causes a flow of the molten material.

By using a conductor with two branches at an acute angle with each other the circulation due to motor force can be made great and can be made to supplement or reinforce, instead of opposing, the circulation due to the "pinch" force.

The secondary of the Ajax-Wyatt vertical ring induction furnace is made in the general shape of an equilateral triangle instead of in that of a semicircle and is shown diagrammatically in figure 35. By this form circulation is obtained by a combination of three forces—gravity (heated metal rising), "pinch" force (as in the Hering furnace), and motor force. The circulation set up is very great, and the motion of the metal appears to facilitate the separa-

⁹⁶ Wyatt, J. R., U. S. Patents 1,201,671, Oct. 17, 1916; 1,235,628, 1,235,629, 1,235,630, Aug. 7, 1917; 1,312,069, Aug. 19, 1919; 1,313,571, Aug. 5, 1919.

⁹⁷ Clamer, G. H., Melting brass in the induction furnace, *Jour. Am. Inst. Metals*, vol. 11, 1917, p. 381. Ajax-Wyatt electric furnace, *Metal Ind.*, vol. 17, 1919, p. 362. The induction electric furnace, *Jour. Franklin Inst.*, vol. 190, 1920, p. 473. *Brass World*, vol. 16, 1920, p. 337, vol. 17, 1921, pp. 3, 36.

⁹⁸ Blakeslee, R. N., Jr., Operating brass-making induction furnaces, *Elect. World*, vol. 74, 1919, p. 642.

⁹⁹ Kenyon, O. A., Developments in brass melting, *Iron Age*, vol. 105, 1920, pp. 819, 880; *Brass World*, vol. 16, 1920, p. 87.

⁹⁹ Kjellin, F. A., U. S. Patent 682,088, Sept. 2, 1901; Schneider, C. P. E., U. S. Patents 761,920, June 7, 1904, and 763,330, June 21, 1904; Hering, C., U. S. Patent 988,936, Apr. 4, 1911.

¹ Northrup, E. F., Nature and explanation of the motor effect in the Ajax-Wyatt furnace: *Jour. Franklin Inst.*, vol. 190, 1920, p. 817.

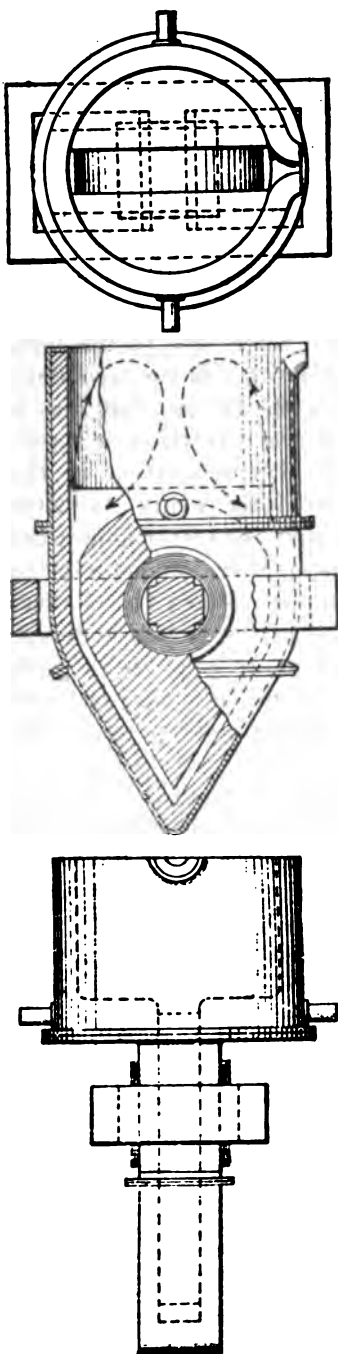


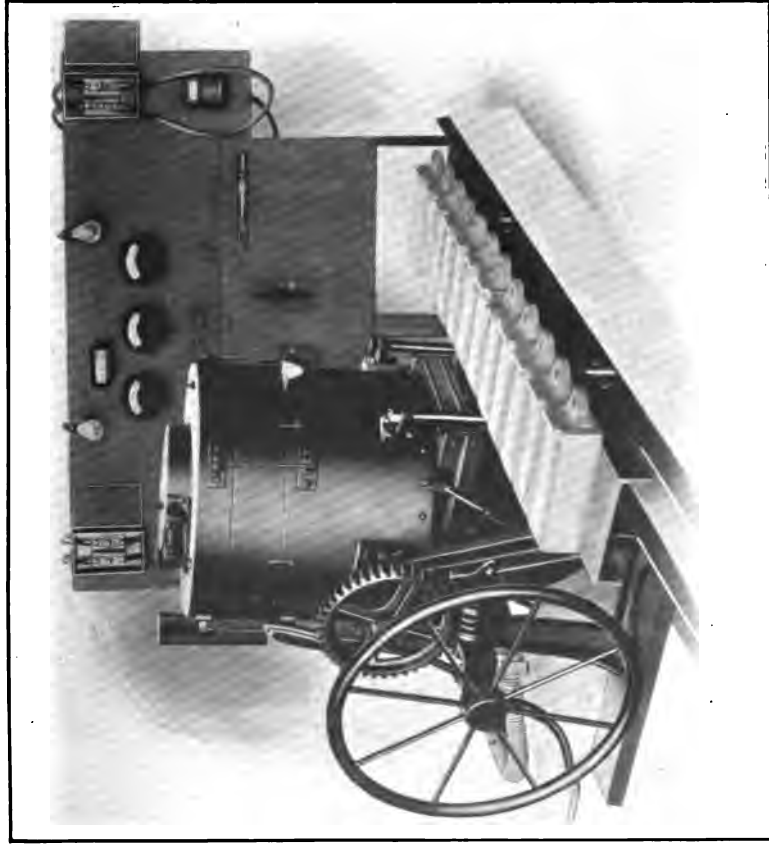
FIGURE 35.—Diagram of Ajax-Wyatt furnace.

tion of foreign particles such as dirt, dross, and oxide, and hence to produce clean metal.

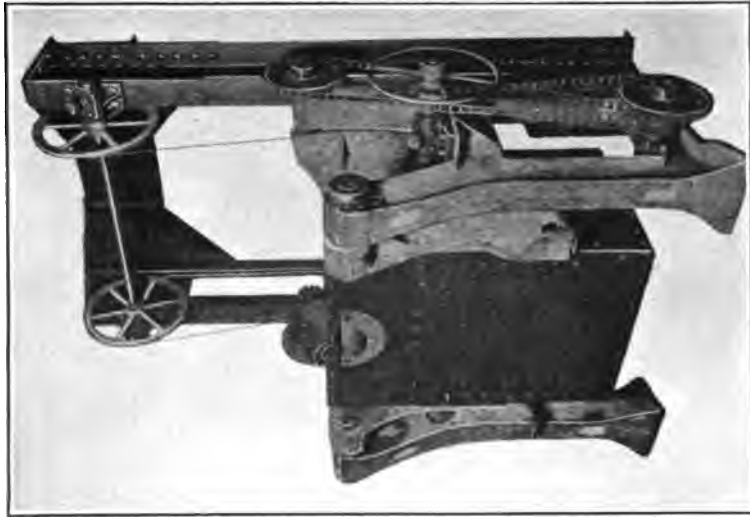
The furnace is designed to avoid the low-power factor common to horizontal-ring induction furnaces. The resistor loop or triangle has, in a 30-kw. size, a cross section of about $3\frac{1}{2}$ by $\frac{1}{2}$ inches, being made thin and flat to inclose as many lines of force as possible and to avoid magnetic leakage. The laminated iron transformer core of the furnace is built in $\square$ shape, as shown in figure 35, the primary coil, of flat copper insulated by asbestos tape, being wound about the central leg, and thus largely inclosed by the secondary loop. The core and primary are inclosed in a brass or other nonmagnetic casing, air from a small motor-driven blower being led into this casing to cool the core and the primary. There is no water cooling of any part of the furnace.

The 30-kw. size uses 220-volt, 60-cycle current at about 200 amperes on the primary, and forms a step-down transformer of a ratio of about 55 to 1, the secondary voltage being about 4, the amperage about 10,000, and the power factor, on 60-cycle current, above 80.

The 60-kw. furnace also has a power factor of about 80. It is calculated that 80 kw. is as high a power as a single loop can take on brass without having the power factor fall below 70. To construct furnaces of such large capacity that more power is required, it will be necessary to go to polyphase operation. A furnace with two or three secondary loops attached to the same hearth could thus be built to take at least 140 kw., at a 70 per cent power factor; such a



A. A 60-KW. TRUNNION-TYPE AJAX-WYATT FURNACE.



B. AJAX-NORTHROP HIGH-FREQUENCY INDUCTION TILTING FURNACE; USED FOR MELTING SILVER AT THE PLANT OF HANDY & HARMON BRIDGEPORT, CONN.



BATTERY OF TRUNNION-TYPE AJAX-WYATT FURNACES AT THE PLANT OF THE BRIDGEPORT BRASS CO.

furnace would melt at least 1,250 pounds of yellow brass per hour. The furnace is so far in commercial use only in 30, 60, 75, and 80 kw. sizes, single-phase, but experimental work is now under way on larger polyphase types.

The furnace does not require so thick a lining to attain a good thermal efficiency as most of the other types do, because at no time is any part of the hearth raised to a temperature above that at which the metal is poured, as the heat is generated wholly in the metal itself. Even the resistor tubes, where the heat is generated, are probably not heated over 25° C. (45° F.) above the pouring temperature of the charge. The problem of refractories as far as resistance to fusion at working temperatures is therefore not difficult, although other factors as will be shown later make this less simple than it appears from the point of view of temperature alone.

The furnace is compact, occupies relatively little floor space, and like all other well-designed electric brass furnaces, is cool to work around. There is no waste space at all in the furnace, so that the wall radiation is at a minimum. The furnace can be tightly closed to prevent loss of zinc, and even if the cover were not tightly closed is of such a form that the heavy zinc vapor will tend to lie in it and not be continually forced out, as there will be no draft through the furnace as long as the pouring spout is plugged tight.

There is, however, no automatically reducing atmosphere in the furnace such as is found in all tightly closed electric brass furnaces using carbon or graphite electrodes or resistors. The furnace atmosphere is normally oxidizing, and it is necessary to maintain a good cover of charcoal over the metal to counteract this. On account of the oxidizing atmosphere there is a tendency for a crust of zinc oxide to form on the inside of the hearth at the top, and in order to prevent this from reducing the hearth capacity, it is necessary to chip it out with an air hammer and high speed steel tools every two or three days.

The furnace is built in two forms, one for ladle pouring, in which the furnace tilts on trunnions, as in Plate XXII, A, the other for pouring direct into molds, in which it tilts on the lip. A battery of trunnion-mounted furnaces, Plate XXIII, and a battery of lip-tilting furnaces, Plate XXIV, are used at the plant of the Bridgeport Brass Co.

Inasmuch as the power supply to the furnace depends on the resistance of the molten metal in the resistor loop, and as the resistance of a 60:40 brass and of pure copper, for example, are widely different, it is obvious that if the furnace is to be used for such different alloys, some means must be provided for varying the primary voltage, as by an autotransformer stepping down the high tension current, say of 2,300 or 6,600 volts to the furnace voltage of around 220

volts, by which the furnace voltage can be altered. By such means alloys of 90 copper, 10 zinc, or 60 copper, 40 zinc can be handled satisfactorily in the same furnace. Alloys above 90 per cent in copper have, however, so low a resistance that a resistor of smaller cross section than that supplied for use on alloys high in zinc would be required. It is more difficult to get a good power factor on the high copper alloys, and the furnace is in general not so well fitted for such use.

An even more important limitation lies in the fact that any short-circuiting of the resistor loop would be fatal, and any break-out which might allow molten metal to run down onto the primary coils or core would not only put the furnace out of commission but might damage parts of the furnace beyond repair. It is thus essential that the refractory forming the resistor loop be free from any tendency to crack and to allow fins of metal to penetrate it.

To meet these conditions, the resistor loop is made by ramming up a high-temperature asbestos cement, containing fire clay and asbestos, about a smooth triangular brass form the size and shape of the resistor loop to be made.

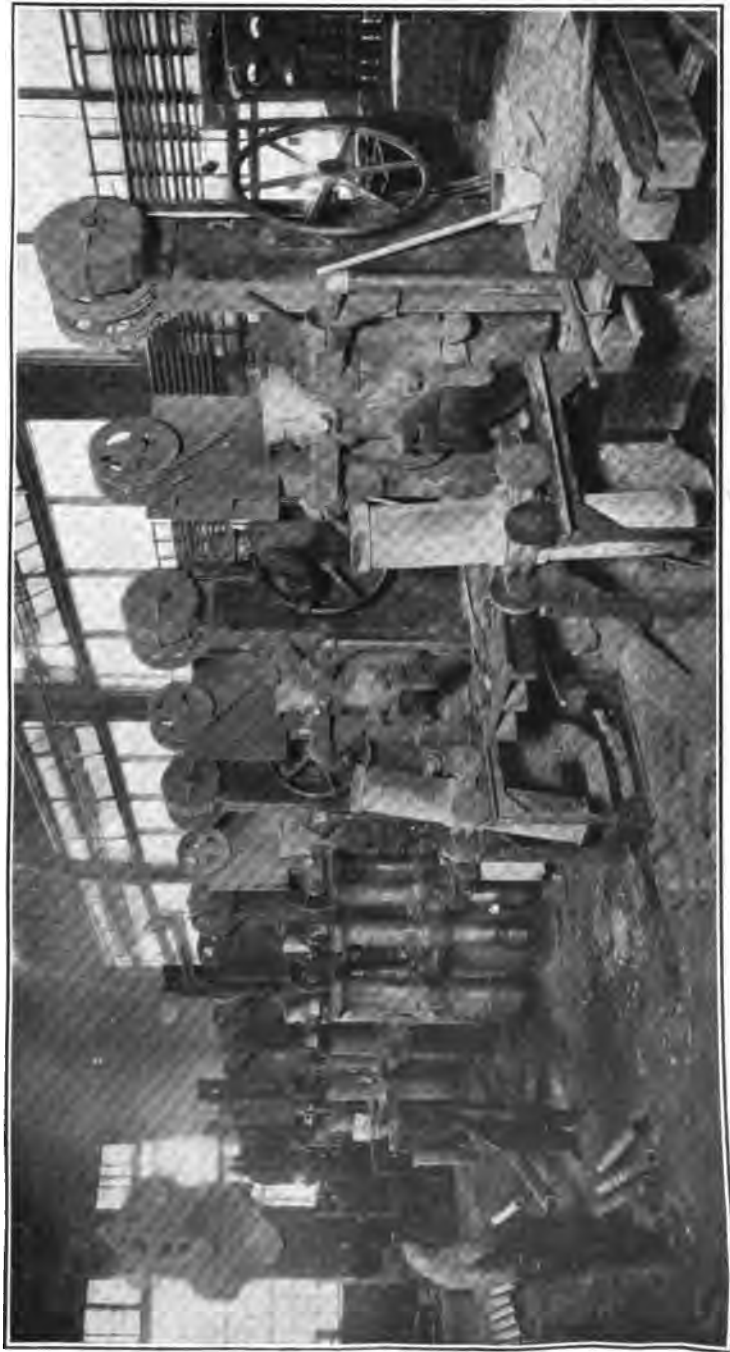
The furnace is dried out slowly in the air, then in an oven for several days, and finally gradually heated up to drive out all moisture and to set the cement firmly. The ramming and drying must be so well done that no cracks develop, and the operation requires skill and experience. Unless the work is properly done the life of the lining will be short.

The asbestos gives the lining sufficient flexibility to stand the expansion of the brass form without cracking. The brass form is left in, and when the furnace is assembled and ready to start a low voltage is impressed on the primary and the brass secondary gradually heated. Molten zinc is then fed in till the brass pattern has melted and the resistor tube is full of molten metal; brass and copper are then fed in till the charge has the desired composition.

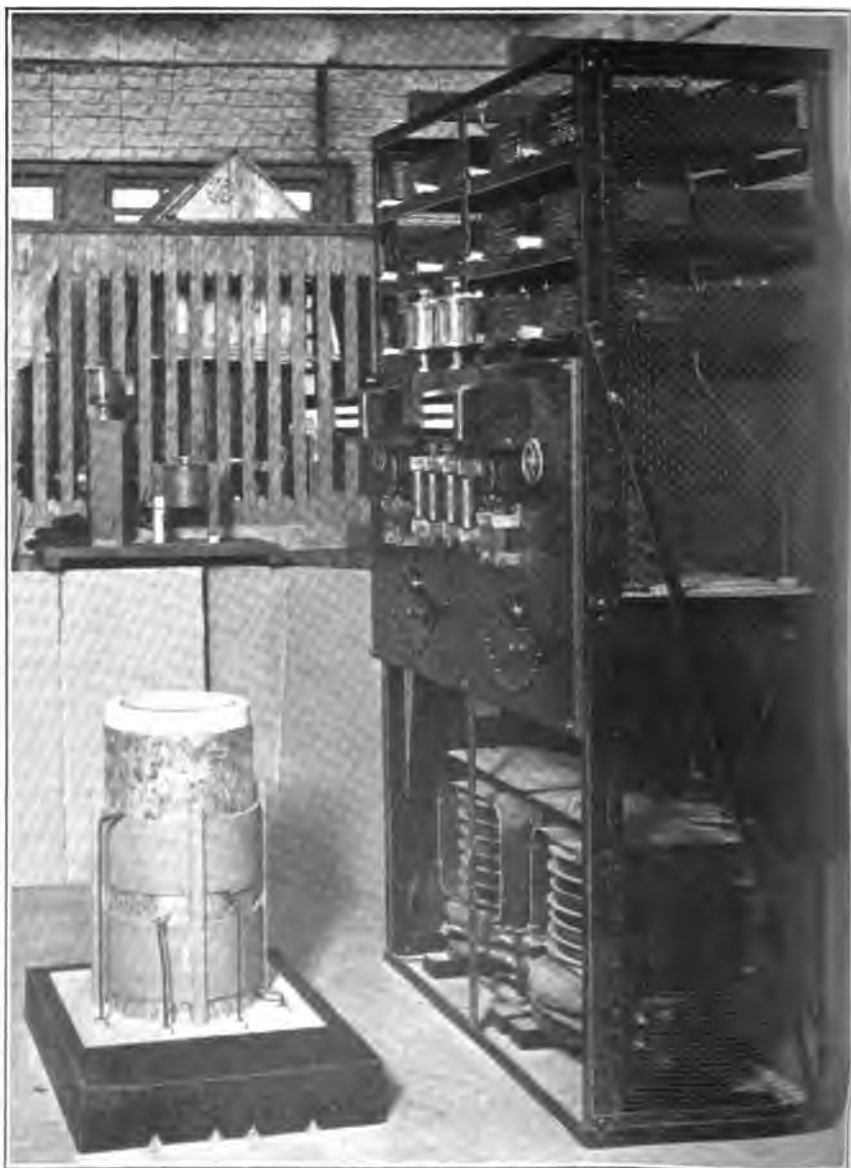
The asbestos cement lining withstands yellow brass successfully, the average life of a lining being 75 tons on the 30-kw. furnace, or 500 heats of 300 pounds, and as high as 375 tons or 2,500 heats has been obtained on individual linings. The cost of relining, including labor and materials, is given² as \$50 for the 40-kw. and \$75 for the 60-kw. size, at 1917 prices. Blakeslee³ gives it as "less than \$100" for the 30-kw. in 1919. The lining does not, however, so successfully withstand the action of alloys high in lead, which is notorious for its ability to soak through the pores of any refractory lining.

² Clamer, G. H., Melting brass in the induction furnace, Jour. Am. Inst. Metals, vol. 11, 1917, p. 381.

³ Blakeslee, R. N., Jr., Operating brass making induction furnaces, Elect. World, vol. 74, 1919, p. 642.



BATTERY OF LIP-POURING AJAX-WYATT FURNACES AT THE PLANT OF THE BRIDGEPORT BRASS CO.



NORTHROP HIGH-FREQUENCY FURNACE AND ACCESSORIES AT THE
PHILADELPHIA MINT.

The makers do not advocate the furnace for alloys containing over 3 per cent to $3\frac{1}{2}$ per cent of lead. One firm reports, however, that a 30-kw. furnace, melting mainly borings from an alloy of about 60 per cent copper, 30 per cent zinc, 5 per cent tin, 5 per cent lead, gave lining lives of 62, 64, and 43 tons. .

Since rolling mills seldom melt alloys containing over 3 per cent lead, extrusion metal being the only composition given by Wood,⁴ which contains more than that amount, the furnace is rather better fitted for the average work of a rolling mill than for that of a foundry making sand castings, as the average foundry must often handle mixtures rather high in lead. This limitation may ultimately be removed by further advances in knowledge of refractories.

POWER CONSUMPTION.

The Ajax-Wyatt is the most efficient electric brass furnace yet developed, in the economical use of power when operated 24 hours a day. Furnaces of other types are seldom very efficient in so small a size as 30 or 60 kw., but it is doubtful if any other type of furnace of whatever size can operate on yellow brass 24 hours a day with as low power consumption per ton of metal melted as the 60 or 80 kw. Ajax-Wyatt. This is thoroughly in accord with the theory, since the furnace has no waste space and generates all the heat in the metal itself. Its radiation losses are therefore small and unlike the Hering furnace it has no electrodes to draw away heat.

The 30-kw. furnace melts 70:30 yellow brass, poured at $1,065^{\circ}$ to $1,095^{\circ}$ C. ($1,950^{\circ}$ – $2,000^{\circ}$ F.) at a power consumption on 24-hour operation of something less than 250 kw. h. per ton as a grand average of figures obtained under average operating conditions in rolling mills. The figures vary from 200 to 300 kw. h. per ton, the lower figure being obtained on heavy, clean scrap and the higher on bulky, oily turnings, which have to be constantly poked down into the bath with the furnace cover open most of the time.

The 60-kw. furnace does better, averaging, perhaps, 200 kw. h. per ton. It has melted 60:40 and 2:1 brass at 174 to 180 kw. h. per ton. When the larger multiphase furnaces are perfected, it will probably be possible to average about 210 kw. h. per ton on yellow brass, 24-hour operation; and, under the most favorable conditions, to get down to 175 kw. h. per ton. As this represents an efficiency of about 85 per cent, based on the most probable figures for the power theoretically required, it is not likely that any other type of furnace, however improved, can melt at a lower-power consumption per ton than is possible in this type.

⁴ Wood, R. A., Brass rolling mill alloys, Jour. Am. Inst. Metals, vol. 11, 1917, p. 181.

On a test of one of the early low-powered Ajax-Wyatt furnaces, which one of the writers witnessed in January, 1916, the furnace taking only 22½ kw., the following results were obtained.

A total of 1,190 pounds of yellow borings—62 per cent copper, 36½ per cent zinc, 1½ per cent lead—poured at 1,010° C. (1,850° F.) average were melted in 7½ hours using 164 kw. h., at the rate of 280 kw. h. per ton.

On a week's run of these borings, which contained about 1 per cent oil and dirt, 94.2 per cent of the charge was recovered as ingot and large pieces of metal from the skimmings; 2.1 per cent was recovered by washing the skimmings, leaving a loss on the original charge of 3.7 per cent, or, deducting the nonmetallic, a net loss of 2.7 per cent. The best figures obtained on the same sort of charge in pit fires showed a loss on the original charge of 6 per cent, or 5 per cent net loss. This figure agrees with that recently given by the editor of *Metal Industry*, who declared⁵ that in melting turnings in a crucible, carefully pushing the turnings below the molten metal as fast as they are added, the volatilization loss is 5 per cent.

There was no apparent volatilization of zinc in this run, though there were formed considerable amounts of light, fluffy dross, which entangled much shot metal.

In a test made in 1917 on two of the first Ajax-Wyatt furnaces installed at the Bridgeport Brass Co., in 24-hour operation on extrusion metal of 60 per cent copper, 1.5 per cent lead, 38.5 per cent zinc, the charge consisting of two-thirds heavy scrap and one-third dirty, oily chips, sometimes carrying very large amounts of oil, up to 10 or 12 per cent, 157,399 pounds of metal was charged, 147,084 pounds ingots (93.5 per cent) were poured, and 3,952 (2.5 per cent) pounds metal were recovered from dross, making a total recovery of 96 per cent and an apparent loss of 4 per cent. Most of this loss was oil and non-metallic charged with the borings. In crucible practice on the same mixture 1½ pounds extra zinc per 100 pounds of charge must be added to make up for volatilized zinc, while if more than one-half pound were added to the electrically melted charges the zinc in the product was too high.

The furnace, therefore, saves at least 1 per cent of the metal charged. Not only was the net metal loss lower than in the pit fires, but the amount of metal to be put through a recovery process was decreased, a given charge producing more metal as ingot when melted in the Ajax-Wyatt than when melted in crucibles.

The 157,399 pounds, 78.7 tons, charged used 16,888 kw. h., or 215 kw. h. per ton. On the basis of the 147,084 pounds, 73.5 tons, of ingots produced, the power consumption was 230 kw. h. per ton. The average

⁵ Reply to inquiry on melting losses, *Metal Industry*, vol. 17, 1919, p. 518.

output was 250 pounds per hour. On further use one furnace melted 100 tons and the other 125 tons before requiring relining. Negro helpers operated the furnaces and poured and shrunk the metal, obtaining as good results as skilled casters on the pit fires.

The Bridgeport Brass Co. is now melting all its metal electrically, has torn down the stacks of its pit fires, and is making an advertising campaign⁶ based on the uniform high quality of electrically melted metal. In one of these advertisements it is stated that the introduction of the electric furnace has been the greatest advance made in the brass industry in 140 years, since direct alloying superseded the calamine process. This firm has even advertised⁷ its reserve stock of crucibles for sale.

Blakeslee⁸ gave in 1919 the following figures for regular operation of a battery of Ajax-Wyatt furnaces at the Bridgeport Brass Co.:

Average power consumption per ton, yellow brass, 24-hour operation, was 250 kw. h., average net metal loss, 0.5 per cent. On a test run, 7½ hours, a furnace gave the following averages:

Kilowatt input	37
Pounds output per hour	291
Pounds poured per kw. h.	9.63
	(or 208 kw. h. per ton)
Power factor	81.8

Blakeslee states that since all melting at this plant has been done electrically there has not been a single case of "brass shakes," and that there has been a reduction in the number of severe burns that formerly frequently occurred from the breaking of crucibles.

The furnace is said to give a well-mixed and uniform product. It is also stated that the ingot can be shrunk better when poured direct from the electric furnace than when poured from the crucibles. In the early test previously cited it was found that the better shrinking allowed making a much smaller crop, to cut off the piped top, than in crucible melting. This firm is now using some larger Ajax-Wyatt furnaces of 80-kw., 1,500-pound capacity, which give about 205 kw. h. per ton. The power factor of these was 60 to 65 during its early development, but it was hoped to improve this. The original lining of the first 80-kw. furnace was still in good shape after melting 995 tons. Attention has also been given by this firm⁹ to the design of a modification of the Ajax-Wyatt, in which heat may be supplied not only at the bottom of the hearth but at the top also by an arc, a re-

⁶ Rocking furnace, *Electrical World*, vol. 74, No. 3, July 19, 1919, front cover. Ajax furnace, *Electrical World*, vol. 74, No. 12, Sept. 20, 1919, p. 32. Removal of stack, *Electrical World*, No. 9, Aug. 30, 1919, p. 11; *Brass World*, March, 1920, p. 24.

⁷ Bridgeport Brass Co., *The Foundry*, p. 188, vol. 48, Apr. 1, 1920.

⁸ Blakeslee, E. N., jr., Operating brass making induction furnaces, *Elect. World*, vol. 74, 1919, p. 642.

⁹ Clark, W. R., U. S. Patents 1,328,712 and 1,328,713, Jan. 20, 1920; 1,370,632, Mar. 8, 1921.

sistor, or by fuel heating, as in the melting of oily borings and low grade, dirty scrap, a crust is apt to form, bridging over the hearth, and requiring excessive poking to submerge it into the bath, when the heating is from the bottom only. A design for preheating by means of fuel¹⁰ has also been suggested, but none of the modifications has been thoroughly tried out.

TESTS OF COMPLETE HEAT BALANCE.

Another interesting development in the work of the Bridgeport Brass Co. with the Ajax-Wyatt has been the making of a test to show the complete heat balance, along the lines of a boiler test. Unfortunately the results of this test have not been published, but it is stated that the energy actually absorbed by the metal agreed well with that determined by pouring the metal into a calorimeter. According to these figures it takes 137.5 calories per pound to heat brass of 61 per cent copper, 3 per cent lead, 36 per cent zinc, to 1,000° C. About 13.72 pounds would be melted per kw. h. at 100 per cent thermal efficiency, or 146 kw. h. per ton.

Richards¹¹ gives a figure for 65:35 brass at 1,000° C., equivalent to 136 kw. h. per ton, and for a 85:15 bronze at 1,050° C. he gives the same figure. Clamer and Hering¹² give 138 kw. h. per ton for 2:1 brass at 1,000° C. Hansen¹³ calculates that it takes 8,240 large calories, that is, 191 kw. h. per ton, to raise 100 pounds of copper to 1,300° C., 7,395 calories to raise 100 pounds of previously alloyed 80:20 brass to 1,100° C. 174 kw. h. per ton and 6,919 calories to raise 80 pounds copper plus 20 pounds zinc to 1,100° C., 161 kw. h. per ton, the last figure being lower than the one preceding it because of the heat of solution of zinc in copper. According to the probable specific heat of molten copper alloys it will probably take from 10 to 15 kw. h. per ton of brass or bronze extra for each extra 100° C. above the melting point to which the metal is raised. The experiments of the Bridgeport Brass Co. indicate that the figures given by Richards, on which electric furnace efficiencies have usually been calculated, are probably low. It seems likely that 150 kw. h. per ton for average yellow brass poured about 1,050° C. and 170 kw. h. per ton for average red brass poured at 1,200° C. are better figures for the theoretical power requirements.

In a 1917 test at one of the plants of the American Brass Co. it was found that on 100 heats of yellow brass the average analysis of the tops and bottoms of the ingots for copper showed a variation of only

¹⁰ Clark, W. E., U. S. Patent 1,328,714, Jan. 20, 1920.

¹¹ Richards, J. W., Electric power required to melt metals, Trans. Am. Brass Founders Assn., vol. 4, 1910, p. 95.

¹² Clamer, G. H., and Hering, C., The electric furnace for brass melting, Trans. Am. Inst. Metals, vol. 6, 1912, p. 101.

¹³ Hansen, C., Electric melting of copper and brass, Trans. Am. Inst. Metals, vol. 6, 1912, p. 112.

0.05 per cent. The slight variation that did exist was almost within the limits of analytical error, the top being higher in copper than the bottom as often as the reverse; that is, the mixing of the charge is perfect, owing to the stirring action of the moving metal. In crucible practice there is a considerable tendency for the composition of the top and bottom of an ingot to vary from incomplete mixing.

It is stated¹⁴ that at the American Brass Co. it has been found that 100 pounds of zinc charged into the 600-pound Ajax furnace is completely and thoroughly mixed in five minutes with no stirring whatever aside from that done by the furnace itself.

It is further stated that when the 60-kw. furnaces, of which this firm has 15 in operation in Waterbury, are operating steadily without delays or interferences they melt 2:1 brass at an average of 174 kw. h. per ton. Each 60-kw. furnace should then produce 6 to 7 tons of yellow brass in 24 hours. Some lining trouble has resulted from nonuniformity in the asbestos cement, and some linings have failed on the first heat. Some trouble has occurred from stoppage of the power supply just after charging, so that the metal froze and ruined the lining. Several furnaces are reported to be operating on linings that have run over 1,500 heats, and the high figure at the time of the report was 1,829—540 tons. With care and barring accidents it is expected to bring the average life up over 2,000 heats, or to over 600 tons' output per lining.

The American Brass Co. pours direct into the ingot molds, either from a nose-tilting furnace or by lifting the trunnion type by a crane. At the P. and F. Corbin plant of the American Hardware Co., New Britain, Conn., a 30-kw. Ajax-Wyatt furnace in 24-hour operation gave, melting an alloy of 60 per cent copper, 30 per cent zinc, 5 per cent tin, 5 per cent lead, the charge being mostly borings, on a run of about 170 tons, an average of 245 kw. h. per ton. It took 10 kw. to keep the furnace and the metal retained in the resistor loop molten hot overnight ready to start the next morning if the furnace was not operated on a continuous 24-hour basis. This heating can be done by using a special transformer for stepping down from the transmission voltage of 2,300 or of 6,600 to the 220 volts for which the furnace is usually designed, and by using a different tap on the transformer to a still lower voltage for the overnight heating.

It is of interest to see what the power consumption will be when the furnace is operated for various periods per day.

¹⁴ Bassett, W. H., Personal communication, Jan. 20, 1920.

TABLE 51.—*Power consumption in 30-kw. Ajax-Wyatt furnace under various schedules of operation.*

Period of operation.	Operating, kw. hours.	Idle kw. hours.	Total kw. hours.	Total out- put, tons.	Kw. h. per ton.
24-hour.....	(30×24)		—820	3.3	250
16-hour.....	(30×16)+ (10×8)		—560	2.2	255
10-hour.....	(30×10)+ (10×14)		—440	1.35	325
9-hour.....	(30×9)+ (10×15)		—420	1.2	350
8-hour.....	(30×8)+ (10×16)		—400	1.1	365

It should be remembered that Table 52 is calculated, not for the best possible 24-hour operation, but on the basis of actual plant operation for long periods, and that it is for the 30-kw. size, not the more efficient 60-kw. size.

Collins^{14a} compares the induction furnace on 8-hour operation with that of the larger General Electric furnace, which is far less efficient in the use of power. Though he is forced to concede that the induction furnace has the highest thermal efficiency of all types, he claims that the smaller size of the induction furnace must involve so much higher labor cost that the sum of the two factors, labor cost and power consumption, with power at 1 cent per kw. h., shows a balance in favor of the General Electric furnace on 8-hour operation. He assumes that the induction furnace metal is to be poured by means of hand ladles, and says that the induction furnace must be charged by hand, while the General Electric furnace can be mechanically charged. These assumptions need not be correct for all cases. In a rolling mill the metal from either furnace may either be poured direct into the molds or from a large ladle; and charging, it would appear to the writer, could be done mechanically into the induction furnace more readily than into the General Electric, because the charging door on the induction type is at the top of the furnace while that on the General Electric is at the side.

The Ajax-Wyatt furnace requires no more attention or control than does the General Electric, with all its elaborate automatic control devices, for the former requires no electrical control other than the pulling of a switch when it is desired to shut off the power.

After watching both types of furnace in operation it would appear to the writers that, under rolling-mill conditions, there is no justification for assuming a higher labor cost per ton with one furnace than with the other, for by staggering the operation, a couple of men could charge, operate, and pour several induction furnaces.

^{14a} Collins, E. F., *Melting of some nonferrous metals and their alloys in the electric furnace*, Chem. and Met. Eng., vol. 21, 1919, p. 673.

If the furnaces are used more than 8 hours per day, or if power costs more than 1 cent per kw. h., the advantage assumed by Collins decreases. The advantage of 25 cents per ton in cost of power plus labor, which Collins assumes the General Electric at 400 kw. h. per ton has over the Ajax-Wyatt at 300 kw. h. per ton, on the 8-hour basis with power at 1 cent, is entirely wiped out if power costs $1\frac{1}{4}$ cents per kw. h. instead of 1 cent.

There is room for a good deal of difference of opinion as to the most economical size of furnace for rolling-mill use,¹⁵ as well as to whether or not metal should be poured direct from furnace to mold. The decisions involve a good many factors. If the plant must pour a large number of small ingots, involving a pause to shrink each ingot, there is less advantage in a large furnace than when a plant can dispose of the metal rapidly into a few large ingots. If direct pouring is employed, then the small furnace allows easier control of the stream of metal.

It must not be forgotten that in big furnaces in which the metal is not mixed, as it is by the Ajax-Wyatt's self-circulating resistor, or by rocking or rotating other furnaces, there is danger that a large charge will not be thoroughly mixed.

In a report made by the metallurgist of a large rolling mill, the vertical ring induction furnace and a reflected-heat furnace are compared. Based on his experiments at a copper refinery where after copper was melted and refined in a 10-ton reverberatory, zinc was introduced in order to form brass and to save remelting the copper, but in which though "vigorously and faithfully" stirred, there was a good deal of variation in copper content of the resulting brass—he does not believe that the reflected-heat furnace in a 1,500-pound size or larger would be satisfactory in making cartridge brass, on account of the necessity of stirring the large charge through the door with a long stirrer.

ADVANTAGES OF AJAX-WYATT FURNACE.

From the point of view of its electrical characteristics, the Ajax-Wyatt furnace is eminently satisfactory. There is practically no fluctuation whatever in the load, so that ordinary distributing transformers are satisfactory for stepping down from line to furnace voltage, as there are no surges. The furnace runs along without any electrical attention. When the metal in the resistor tube reaches the boiling point, and bubbles of zinc vapor begin to be released, the resistance rises and the ammeter needle starts to vibrate or "kick." This

¹⁵ Compare Hering, C., Advantages of small high speed electric furnaces, *Met. and Chem. Eng.*, vol. 11, 1913, p. 188.

means either that the whole charge is hot enough to pour, being analogous to the "jump" of an iron bar thrust into the crucible to serve as a "works pyrometer" on yellow brass, or else that the charge has bridged over in the hearth and needs poking down.

The advantages of the Ajax-Wyatt may then be summed up as follows: It is a compact furnace, requiring little floor space. It reduces the metal losses on alloys high in zinc. It is thermally the most efficient brass furnace known; that is, on 24-hour operation it requires the lowest kw. h. rate per ton. It mixes the metal perfectly. It uses no electrodes or resistor carbon and requires no water cooling. Its electrical characteristics make it a desirable electrical load. Its reliability is high and its lining cost per ton is low.

Its disadvantages are (1) that it must be operated 24 hours a day or else kept hot overnight by using enough power to keep the metal retained in the resistor-loop fluid, as a freeze-up inevitably ruins the resistor tube and makes relining necessary; (2) if it is to be shut down, all the metal must be drained out, and in starting up it must be thoroughly preheated, as by an oil or gas flame, and it must then be started by pouring in a charge that has been melted in another furnace. The retention from one heat to another of enough charge to fill the resistor and to make a connection in the hearth between both legs of the tube makes it difficult to change from one alloy to another, unless the charge is such that the retained metal can be worked into the composition of the next charge.

The furnace will not handle alloys very high in copper, nor can alloys of high lead content be handled with the present lining. Even 5 per cent lead will probably reduce the life of the lining considerably.

The furnace is obviously fitted then for use in a brass rolling mill, in which yellow brass is melted, and in which 24-hour, or at least 16-hour operation is possible and very common, and in which the perfect mixture given by the furnace is very desirable. There are rolling mills in which conditions are such that furnaces of larger capacity than the present commercial sizes of the Ajax-Wyatt are desirable. In general, however, the furnace is worthy of careful consideration by any rolling mill, and so far it holds first place in actual achievement in that class of work.

It is obviously not fitted for use in foundries that do not melt large tonnages of yellow brass, but that must often change the composition of the charges to any one furnace from heat to heat. It is not fitted for use on highly leaded mixtures or on alloys around 90 per cent copper. It begins to operate at a decided handicap when it can not be used at least 16 hours a day, therefore it can not be considered at all by the average jobbing shop.

Its field is on the high zinc end of the scale, being best adapted for just those alloys which the direct-arc and stationary indirect-arc furnaces can not handle.

Nearly any alloy made in the nonferrous industry can be satisfactorily melted in either an Ajax-Wyatt furnace or a stationary indirect-arc furnace, but neither furnace can operate satisfactorily on even occasional heats in the other's field. Hence there is still room for the more versatile furnaces like the reflected heat and the rocking types.

The prices of Ajax-Wyatt furnaces, complete, save for the transformer for stepping down from line voltage to 220 volts, were given on March 29, 1920, as follows, for the standard sizes: 40 kw., \$5,500; 50 kw., \$6,500; 80 kw., \$7,500.

AJAX-NORTHROP FURNACE.

One more furnace remains to be considered, which is potentially even more versatile than any other as it can be built in either a tilting or tapping, or in a crucible lift-out type, can melt any alloy and work almost equally well either 24 hours a day or only an hour or two. It is considered here instead of under crucible furnaces because the heat is generated in the metal itself and because, like the Hering and the Wyatt, it is a type which the Ajax Metal Co. has been instrumental in developing. This is a new type of furnace invented by Prof. E. F. Northrup, of Princeton University, in July, 1916, and developed with the aid of the Ajax Metal Co. The Northrup furnace, instead of using ordinary alternating current, uses a high-frequency current such as is used in the wireless telegraph. Ordinary alternating current has 60 cycles or alternations per second, and the frequency of the current used in the Northrup furnace is preferably of the order of 10,000 or more cycles per second. Conducting materials in proper position to receive the potential energy released by a high-frequency current may absorb this energy, and by their resistance to the flow of current generate heat within themselves.

The Northrup furnace is an induction furnace, the heat being generated in the charge without the use of any electrodes to carry the current, but, unlike the usual induction furnace, the secondary does not have to be in the shape of a ring or loop. The heating is analogous to that due to eddy or Foucault currents in a transformer core.

Guggenheim¹⁶ has suggested the use of eddy currents in an electric furnace, but no commercial development of his suggestion is known.

¹⁶ Guggenheim, S., U. S. Patent 1,157,891, Oct. 26, 1915.

Northrup¹⁷ has briefly described his furnace and has given an exhaustive, 89-page account¹⁸ of the theory and the mathematics involved.

The furnace itself as applied to brass or silver melting consists of a conducting container, such as graphite for the metal, which may be a crucible that can be lifted out, a stationary crucible, or other container for pouring by tilting, or a container with a tap-hole at the bottom. This container is surrounded by a heat-insulating refractory, and in small furnaces is made very thin in order to secure "close coupling" of the inductor or primary coil with the secondary, the conducting crucible lining and the charge of brass itself. Large furnaces may be able to use a thicker refractory, since the radius of the crucible to be heated should theoretically be three-fourths the radius of the inductor coil.

Wrapped closely about the outside of the refractory is the "inductor," or primary, a spiral conductor that can be made of a nickel-chromium alloy, or may be of copper tubing and cooled by a stream of water. To this inductor spiral are connected the leads, one at each end for single-phase or one at each end and one in the middle for two-phase, from the source of high frequency current.

This current source may be an ordinary alternating current stepped up to a high voltage, 5,000 to 8,000 volts, connected to a bank of condensers and a special form of gap, also invented by Northrup, forming a Tesla oscillatory current circuit. The gap consists of graphite over mercury, in an atmosphere of alcohol vapor, obtained by a flow of alcohol from an oil cup. The Tesla circuit method has been tried only up to 60 kw., and is thought to be applicable only up to about 100 kw., because of the cost of the condensers. A special grounded guard is placed about the primary inductor to prevent shock, and a "focus inductor" may be used, consisting of a device placed inside the primary inductor and consisting of two parts joined together, one part near the primary, made of copper, and the other near the crucible and in contact with it, made in a

¹⁷ Northrup, E. F., Production of high temperature and its measurement, *Met. and Chem. Eng.*, vol. 17, 1917, p. 685; Some results obtained in electric heating with high-frequency induction, *Pyroelectric Monthly Bull.* No. 9, November, 1919. Anon., Northrup-Ajax frequency induction furnace, *Chem. and Met. Eng.*, vol. 19, 1918, p. 155. Anon., Electrical production of carbon-free alloys, *Chem. and Met. Eng.*, vol. 21, 1919, p. 258. see also Northrup, E. F., U. S. Patents 1,286,394; 1,286,395, Dec. 3, 1918; 1,297,393, Mar. 18, 1919; 1,328,386, Jan. 20, 1920; 1,380,133, Feb. 10, 1920; Wilcox, D., The Ajax-Northrup electric furnace: *Metal Ind.*, vol. 18, 1920, p. 213; Alloying precious metals electrically: *Metal Ind.*, vol. 19, 1921, p. 171.

¹⁸ Northrup, E. F., Principles of inductive heating with high-frequency current, *Trans. Am. Electrochem. Soc.*, vol. 35, 1919, p. 71; (abstract) *Met. and Chem. Eng.*, vol. 20, 1919, p. 381. High frequency induction steel furnace, *Chem. and Met. Eng.*, vol. 24, 1921, p. 309. Recent progress in high frequency inductive heating, *Trans. Am. Electrochem. Soc.*, vol. 39, 1921, p. 179; see also Clamer, G. H., The induction electrical furnace, *Jour. Franklin Institute*, vol. 190, 1920, p. 473; Kreutzberg, E. C., Melts metal for [unclear] in the mold (Cammens process): *Foundry*, vol. 49, 1921, p. 777.

brass melting furnace, of nichrome or Monel metal. This focus inductor may be grounded so that the high tension primary is within two grounded guards. The focus inductor also tends to increase the "close coupling" of the primary and secondary circuits, though it has the disadvantage of introducing something beside the material to be melted and the crucible containing it in the hot zone, which is against reliability, and limits the temperatures obtainable to the safe operating temperature of the material used for the inner portion of the focus inductor.

Condensers are fairly reliable and long lived; if properly housed outside the furnace room, they need not give much trouble. In order to avoid deterioration of the condensers by overheating, it is necessary, if the furnace is to be operated 24 hours a day, to provide two sets of condensers so that one may be cooling off while the other is in use, for artificially cooled condensers have not been constructed.

The power factor of a Northrup furnace on a Tesla circuit runs from 50 to 60 or 75 per cent.

The gap requires attention about every 500 hours to keep it in working order, and about 1 pint of denatured alcohol is used every 300 kw. h. For small installations, such as in the physics departments of universities or for some commercial work in the melting of small amounts of precious metals, the Tesla circuit is satisfactory.

The high-frequency current may also be obtained from an oscillatory current high-frequency generator, which has been built up to a 60-kw. size, or from a high-frequency alternator, which is already commercially built in 200-kw. or larger sizes, and would probably be the best commercial source. With the high-frequency alternator, by suitable use of static capacity, the power factor of the circuit can be made unity.

As high-frequency current is not at present generated or transmitted by central stations, a foundry wishing to utilize the Northrup furnace must either be content with a comparatively low-powered furnace and with the use of a condenser set or else install its own high-frequency alternator. The high voltage used at usual low frequencies would be dangerous to life, and great care is taken to construct the furnace so that it is grounded to avoid shock. It is probable, however, that at the high frequency employed these high voltages are not dangerous, as in a test made by the Ajax Electrothermic Corp. a rabbit withstood the full voltage without injury.

The Ajax-Northrup furnace is therefore, so far, useful for research in physics and metallurgy, especially as the furnace can readily be constructed as a vacuum or a high-pressure type and for an at present limited field of commercial uses. Over 30 Ajax-Northrup furnaces have been installed for laboratory use. The furnace can

not at present be classed as ready for general commercial use for brass melting under existing conditions. The ultimate possibilities of this type of furnace are great, but much development must be made on the production of high-frequency current before the possibilities in brass melting can be realized. The furnace itself is probably much closer to commercial usefulness than the generation of the current it requires, and it is now of commercial value for handling the more precious metals.

Small furnaces for commercial metal melting have been sold to Baker & Co. for melting platinum, to J. R. Wood & Co. for melting gold, four to the Philadelphia Mint for melting gold and silver (see Pl. XXV, p. 265), and a nose-tilting, three-phase, 60-kw., 550-pound furnace was put into operation, melting silver, at the plant of Handy & Harmon, in June, 1920 (see Pl. XXII, B, p. 262). By the use of a three-phase system it was possible to secure a decidedly larger capacity in a given condenser than before. The power factor is around 70 and the furnace melts sterling silver at about 250 kw. h. per ton.

The Director of the Mint¹⁹ speaks very highly of the Ajax-Northrup furnaces used at the mint for melting silver.

A recent form of the Ajax-Northrup, called an "electric crucible," consists of a conical hearth. This, for melting brass or other materials of high electrical conductivity which are not contaminated by carbon, is lined with graphite. Melting can be done by the current induced in such metals, but not so rapidly nor so efficiently as when a conducting lining is used. Materials of high resistance, such as iron and nickel, do not require the conducting lining.

The conical pot with its surrounding coil is mounted in a box-like container which may be lifted off bodily just like a crucible, and poured with a special shank, or it may be tilted for direct pouring. A 150-pound 25-kw. furnace of this type is being tried out at the Ajax Metal Co.

That the eddy current furnace itself is efficient is shown by a test of an 18-kw. furnace which melted copper at the rate of about 240 kw. h. per ton, using power from an Alexanderson alternator. With oscillatory current from a condenser set, the power consumption was at the rate of about 500 kw. h. per ton when the power losses in the transformers, condensers, leads, etc., were included.

The furnace gives a circulation of the charge, the metal flowing upward in the middle of the crucible and downward at the sides on account of electromagnetic forces.

The furnace therefore is like the Ajax-Wyatt in that the heat is at least partly generated in the metal itself, and in that a forced

¹⁹ Baker, R. T., Report of the Director of the Mint for the fiscal year ended June 30, 1920, Treas. Dept., Doc. No. 2874, 1920, p. 19.

circulation is set up. It has no secondary loop in which a freeze-up will ruin the lining, and all the metal can be poured or tapped from the furnace when made in the noncrucible type. It is therefore free from the limitations of the Ajax-Wyatt as to changing from one alloy to another, and it can handle pure copper as well as alloys high in zinc. It does not have to be kept hot over night. It is almost as efficient on a first heat as it would be after a week of 24-hour operation per day, because the thin heat insulation used in order to get "close coupling" is so small that there is very little heat storage in the walls. The use of thin walls may, of course, reduce its mechanical stability. However, the freedom from complications in the hot zones, no electrodes, no resistor, no secondary loop, makes it promise to have sufficient mechanical reliability. The condensers supplied with the Tesla outfit are guaranteed for one year. The price of the 60-kw. furnace, Tesla circuit outfit, for 220-volt, 60-cycle power, was given in June, 1920, as \$6,750 for the crucible lift-out and as \$7,000 for the trunnion tilting form. That of a 25-kw. furnace of the portable "electric crucible" type, holding 10 pounds of aluminum or 30 pounds of copper, was given as \$3,350 in 1921.

The combination of efficiency and flexibility makes the furnace perhaps the most promising of all for ultimate use, provided it can be satisfactorily developed into larger sizes and high-frequency current can be almost as readily and cheaply obtained as ordinary alternating current is now. As high-frequency current can not well be applied to the ordinary uses of electricity, the furnace, while an excellent application of scientific ingenuity, does not seem likely to displace the common alternating-current furnaces in the immediate future. Its present field is in the laboratory, as a tool for experimental work, and in the commercial melting of precious metals such as gold, silver, and platinum.

ELECTRIC MELTING OF ALUMINUM.

The foregoing description of the various electric furnaces has dealt mainly with brass. One more very important class of nonferrous alloys, the light aluminum alloys, should also be considered in connection with electric melting.

Detailed information on electric melting of aluminum is about as unavailable as that on electric brass melting was a few years ago, and it is necessary to consider this subject more as theory than from present practice.

According to data recently gathered by the Bureau of Mines,²⁰ about 45,000 tons of light aluminum alloys are produced in the United

²⁰ Anderson, R. J., *Light aluminum alloy castings*: Tech. Paper 287, Bureau of Mines, 1922, 50 pp., 4 pla.; *Analyze loss in aluminum shops*: Foundry, vol. 48, 1920, p. 989, vol. 49, 1921, pp. 16, 54, 104, 143, 188.

States every year, around 98 per cent being aluminum-copper alloys and only $1\frac{1}{2}$ per cent, or 750 tons, being aluminum-zinc or aluminum-copper-zinc alloys.

A single electric furnace of around 1-ton capacity could melt all the zinc-containing aluminum alloys produced in the country and not have to work much over 10 hours a day to do it. As it is only the zinc-containing aluminum alloys in which there is any noteworthy loss from volatilization, it is plain that the situation in regard to aluminum is diametrically opposed to that in brass. Oxidation losses do occur, and good metal is entangled in and skimmed off with the dross, but the net loss of metal in melting the standard No. 12 alloy is probably as low as 1 or 2 per cent. Moreover, those who have had the most experience in the electric melting of aluminum doubt whether, starting with the same charge, an electric furnace will save enough metal, more than a well-operated fuel-fired furnace, to make much difference in the cost.

Most aluminum is melted either in iron pots or in some type of reverberatory furnace. Where it is still melted in graphite crucibles, the crucibles have a relatively long life, hence two of the outstanding advantages of electric brass melting—reduction of metal losses and avoidance of high crucible costs—do not obtain in any marked degree in electric aluminum melting. On aluminum, therefore, electric melting must justify itself by quality, ease of control, decreased labor cost, or lower cost of electric power than of fuel required.

As to quality, everyone who has tried electric melting seems to agree that electrically melted aluminum is of good quality, and some seem to feel that it is or may be of superior quality. Perhaps because of the good general reputation of the electric furnace on the score of quality in handling other alloys, some aluminum founders are hoping that the electric furnace may do away with cracks, draws, and porous castings, or may at least eliminate the possibility of these troubles originating in the melting of the metal, thus leaving "only" the thousand and one molding and core causes that may produce these evils.

Work of a scientific character on testing the product for porosity²¹ indicates that the electric furnace offers possibilities, but even these tests were often discordant and contradictory. It can be said with considerable confidence that a reducing atmosphere is better for finishing steel and melting brass than an oxidizing atmosphere. But in view of the oxidizing atmosphere over an iron-pot furnace for aluminum and the fact that such furnaces can produce good metal, and that in a reverberatory type, oil-fired aluminum furnace the atmosphere may be made reducing, and that this type can also

²¹ Compare Bezenberger, F. K., and Wilkins, E. A., Apparatus used for the determination of the porosity of metals: *Chem. and Met. Eng.*, vol. 22, 1920, p. 1081.

produce good metal as well as bad, one is not justified on present data in jumping to a conclusion that an electric furnace need be good for aluminum because it may have a reducing atmosphere. Overheating in an electric furnace seems to favor the absorption of gases and the production of castings containing tiny pinholes in exactly the same manner as it does in fuel-fired furnaces.

Neither on the score of theory nor recorded foundry tests of electric aluminum furnaces can we then assume any definite direct advantage for the electric furnace, save perhaps in the elimination of hard spots in the castings due to iron scale from the iron pot, an advantage shared by fuel-fired hearth-type furnaces as well.

In ease of control, there is a real advantage, because of the great effect that the rate of solidification, which is chiefly controlled by the pouring temperature, has on aluminum. Though it has never been satisfactorily explained, and hardly fully proved, there is a general idea that aluminum once overheated, and it may be in either a reducing or oxidizing atmosphere, is thereby irreparably injured, even though it be poured at the proper temperature. If this be so, then an electric furnace of a type not too sluggish in its response to changes of power input might be an advantage, and its use at any rate should make careful pyrometric control of pouring temperatures easier.

FURNACES USED BY GENERAL ELECTRIC CO.

The General Electric Co. uses a small 75-kw. furnace like their older type of electric brass furnace, holding some 75 or 100 pounds of aluminum to melt a nearly pure aluminum which must flow very freely as it must run down through the slots in a motor rotor. This rotor casting is made in an ingenious centrifugal casting machine, and to secure the desired results the temperature and fluidity of the metal and the speed of the centrifugal device must each be just right. The control over the metal made possible by the furnace, and the fact that the furnace itself is automatically operated, is considered to make the obtaining of good results more certain and easy than with a fuel-fired furnace. This furnace has been in use over a year and a half without relining, the only difficulty being that an accretion of oxide in the hearth has cut down the hearth capacity. On account of the low pouring temperature of aluminum compared to brass or bronze, almost any type of electric brass furnace applied to aluminum should have very little difficulty in maintaining the refractories.

The General Electric Co. also uses a larger furnace of the same type of about 800 pounds capacity, 150 kw., for ordinary aluminum casting work. It has also designed a very large furnace holding several tons, which it hopes to try out experimentally. The design calls for a long, rather narrow hearth with a weir or a dividing wall with a hole in it, about halfway up, between the two parts of the hearth.

Metal is to be charged at the back, melted, and clean metal, free from dross floating on top, or from foreign particles heavier than the metal, is to run through into the fore part. The rate of melting in the rear part of the hearth and of superheating to pouring temperature in the fore part can be regulated by adjusting the position of the electrodes along the front or rear to generate more or less power as desired. This scheme offers some possibilities, but they must be tried out before their value is certain.

The General Electric Co. is building still another type of electric aluminum furnace to be tried out at the Pittsfield, Mass., plant. This is a rectangular tilting furnace, like a long box, the hearth occupying the lower part, the heating element being strips of nickel-chromium resistance ribbon, such as is used in the heat-treating furnaces for gun forgings.²² The heating element is placed not only on the roof proper but also on the sides above the hearth. Such a resistor will not stand operation at temperatures sufficient to melt brass, but it may handle aluminum at a repair cost that will not be prohibitive. One factor leading the General Electric Co. to build this type was a desire to secure comparative data on the effect of the oxidizing atmosphere this furnace will have against that of the reducing atmosphere in the arc-resistance type used both for brass and aluminum.

The General Electric arc-resistance furnaces for use on aluminum are the same as for brass, save that the transformer capacity and power input are lower for the same size shell than on brass. They give the following table for different sizes on basis of 24-hour operation, after furnace is fully hot:

TABLE 53.—*Maker's claims for General Electric aluminum furnace.*

Kw.	Hearth capacity, pounds.	Nominal pounds melted per hour.	Approximate kw. h. per ton.
200	650	500	525
800	1,000	750	500

One Baily standard 105-kw. furnace is regularly operating on No. 12 aluminum at the plant of the Dayton Engineering Laboratories Co., Dayton, Ohio. A test run on the furnace showed an output of about $1\frac{1}{2}$ tons per day, in $8\frac{1}{2}$ actual hours melting, plus 1 hour pre-heating the furnace, or 7 to 8 heats of 425 pounds each. The average power consumption ran from 625 to 675 kw. h. per ton. The Delco people state that they are very well satisfied with the furnace.

²² Johnson, L. F., How the powerhouse aids the forge, *Iron Trade Review*, 1919, p. 1223, Collins, E. F., Metallic resistor electric furnaces for heat-treating operations, *General Electric Rev.*, vol. 23, 1920, p. 433.

At the plant of Landers, Frary & Clark, New Britain, Conn., one of the 50-kw. rectangular Baily furnaces was operated for about two months before it was taken out on account of trouble in maintaining the resistor trough, to be later replaced by the round type. This furnace, melting 9 hours per day and being preheated 4 hours, gave on 150-pound heats an output of $\frac{1}{2}$ ton per day, at 850 kw. h. per ton. The net metal loss was about 0.7 per cent.

One of the 105-kw. Baily furnaces was tried out some years ago by the Aluminum Co. of America for making ingot from pig aluminum, but was not kept in service, being considered too small for such work, and not showing any particular advantage as to metal losses.

A big 500-kw. tapping-type Baily furnace was later installed at the plant of the United States Aluminum Co., at Massena, N. Y. This type has been described by Miller.²³

It has two straight resistors, one on each side of the hearth, and held 3 to 4 tons of aluminum. In this furnace, according to the users,²⁴ under best conditions and continuous operation, the production was about 1 ton per hour at slightly under 500 kw. h. per ton. The furnace was primarily installed for making alloys high in zinc, and on such service it showed a notable decrease in metal loss over the fuel-fired reverberatory furnaces. But on the regular run of aluminum there was no decrease in metal loss. The resistor troughs have been a source of trouble, failing frequently, and the high heat capacity of the furnace makes it less efficient on intermittent operation, the power consumption averaging some 700 kw. h. per ton on the furnace as actually operated over a period of time.

As a consequence of the high power consumption the use of the furnace has been practically given up, and its owners contemplate changing it over into an annealing furnace, as such furnaces give very satisfactory results at the lower operating temperatures used in annealing.

It should be mentioned that a 105-kw. Baily furnace has long been in satisfactory operation at the Lumen Bearing Co., Buffalo, N. Y., on an alloy of around 85 per cent zinc, 10 per cent copper, 5 per cent aluminum, which is poured at about 700° C. The furnace gives around 5,500 pounds per 10-hour melting day, using about 660 kw. h. in the daytime and 330 at night to keep the furnace hot, or about 360 kw. h. per ton on this low-melting alloy. The use of the large 1,000-kw. Baily furnace installed at the Anaconda Copper Mining Co. has been discontinued.

A few isolated heats of aluminum have been made in Snyder direct-arc furnaces, but no data of consequence are available. If

²³ Miller, D. D., The remelting of aluminum pig in the electric furnace, *Chem. and Met. Eng.*, vol. 19, 1918, p. 251.

²⁴ Vail, A. E., Personal communication, Jan. 21, 1920.

the direct-arc furnace will produce a satisfactory quality of metal, without undue gas absorption in the overheated metal under the arc, such a furnace should work well. The common aluminum alloys, outside of the zinc-containing alloys, are relatively non-volatile, and in general, direct-arc furnaces are useful for such alloys. This question ought to be investigated, and the Bureau of Mines hopes to study it.

The Rennerfelt indirect-arc furnace in a small 50-kw. size was used by the Acieral Co. of America in making up "hardeners," but not in melting the aluminum alloy itself, before that firm went out of business. Similar hardeners are now handled in this furnace, using 100-kw. transformer capacity, at the Bausch Machine Tool Co., Springfield, Mass. No real tests of stationary indirect-arc furnaces on aluminum alloys themselves are on record. On the basis of some small test runs in the 50-kilowatt furnace, made by starting with the furnace at the temperature used for making hardeners, and therefore well above aluminum temperatures, it was calculated that in a 1-ton furnace 310 kw. h. would be required, presumably on 24-hour operation. These figures were later revised to 350, but neither figure is based on adequate test.

One of the standard Detroit rocking furnaces rated at 300 kw. was given a thinner lining than for use on brass, the melting chamber being 49 inches in diameter by 41 inches, the furnace thus lined holding 1,700 pounds of aluminum, and was tested at the C. B. Bohn Foundry Co., Detroit. The power input was raised to 325 to 375 kw., and on about 60 tons of No. 12, the actual melting time averaged about 1 hour, 10 minutes per heat, corresponding to an output of about 5 tons per 10-hour day, at about 475 kw. h. per ton. The net metal loss averaged about 0.85 per cent, running about 0.4 per cent on American ingot, scrap No. 12 and gates, and up to 1.3 per cent on some French ingot that appeared to contain some nonmetallic impurities that were weighed in as metal. It is also possible that there was some formation of aluminum carbide due to piling the charge too close to the arc in the runs on the French metal which were made first. The product from both the French and American metals was of satisfactory quality. A 1,200-pound Detroit furnace is operating on aluminum at the Storm Peak Potash Co., Denver.

The Ajax-Wyatt induction furnace has not been tried on aluminum.

It will be noted that all the data on aluminum show a high power consumption per ton of metal compared to that required by the same furnace on brass. The figures range on 10-hour operation from 850 kw. h. per ton in the 50-kw. Bailly on 150-pound heats, through 650 kw. h. per ton in the 105-kw. Bailly on 425-pound heats, to 475 kw. h. per ton in the 375-kw. Detroit on 1,700-pound heats, and the 24-hour operation figures for the 3 to 4 ton Bailly tapping furnaces,

and for the 1,000 to 1,400 pound General Electric furnaces, are 500 kw. h. per ton under the best conditions. Roughly, a given size and type of furnace requires at least 50 per cent more power per ton to heat aluminum to 700° C. than it does yellow brass to 1,050° C.

In fact, theory calls for the expenditure of about twice as much energy as, averaging the available data, 100 per cent melting efficiency calls for about 150 kw. h. per ton for yellow brass²⁵ and 300 kw. h. per ton for No. 12 aluminum.²⁶

In fuel-fired furnaces the difference in fuel consumption is not so marked because such furnaces operate at a higher efficiency the lower the temperature, so that a given fuel-fired furnace can probably produce a ton of aluminum with not much more fuel than it would use on yellow brass.

As there are no such savings in metal losses and in crucibles as in brass melting, and there is no definite proof of improved quality of metal, the value of electric furnaces for aluminum depends on whether or not an aluminum melter can melt cheaper by paying perhaps \$7 to \$10 or more per ton—depending on the cost of power and the size and type of electric furnace chosen—for electric power plus the interest and depreciation charges per ton on an expensive furnace (that will give only a small output, compared to brass, of aluminum) than by paying his present fuel costs. At 1920 fuel prices he might often do so, but he seldom has the incentive of so great a saving as to make a change imperative. On the other hand, the larger aluminum founders may often have so large an output that the large amount of electric power they would use for melting would come at a low figure and make a respectable saving. Similarly, plants melting brass electrically could add electric furnaces for their aluminum and could often get the added amount of power required at a rather low figure. Convenience, coolness, and cleanliness of the electric furnace might also be deciding factors.

But any use of electric furnaces for aluminum comparable to that for brass will be a slower development, unless it is definitely proved that some electric furnace operated in some particular manner will really improve the quality of the metal and will materially cut down the percentage of defective castings. There is a good chance that this may be possible, but it has not yet been demonstrated. Another

²⁵ Compare Richards, J. W., Electric power required to melt metals, *Trans. Am. Brass Founders' Assn.*, vol. 4, 1910, p. 95. Clamer, G. H., and Hering, C., The electric furnace for brass melting, *Trans. Am. Inst. Metals*, vol. 6, 1912, p. 101. Hansen, C. A., Electric melting of copper and brass, *Trans. Am. Inst. Metals*, vol. 6, 1912, p. 112. Also unpublished data from Bridgeport Brass Co.

²⁶ Compare Laschtschenko, P. N., Specific heat of aluminum: *Jour. Russ. Phys. Chem. Soc.*, vol. 10, No. 56, 1914, p. 311; Merica, P. D., Aluminum and its light alloys: U. S. Bureau of Standards, Circular 76, 1919, p. 27; Wüst, F., Meuthen, A., and Durrer, R., Thermal constants of technical materials: *Zeit. Instrumentenkunde*, vol. 39, 1919, p. 294; *Chem. Abstr.*, vol. 14, 1920, p. 1241.

possible use for electric melting furnaces which has yet to be thoroughly demonstrated is in melting cathode copper.

DIRECT-ARC AND OTHER FURNACES FOR MELTING CATHODE COPPER.

Lyon and Keeney²⁷ point out that the purity and conductivity of cathode copper are reduced by the impurities introduced in reverberatory melting and that it is possible to melt it in the electric furnace without the absorption of impurities.²⁸ The possible application of the various types of furnace available in 1914 is considered. The horizontal ring induction furnace is barred out because of troubles from pinch effect due to the high conductivity of the material; the pinch-effect furnace is similarly barred because of the low voltage required; the stationary indirect-arc furnace is considered a possibility but on account of the cost of upkeep is not thought to be as promising as a direct-arc type with a nonconducting bottom, such as the Heroult.

Possible loss of copper by volatilization, according to Lyon and Keeney, can be prevented by the use of a slag. In experimentally testing the operation of a direct-arc furnace on copper, however, it should be recalled that there have been severe and almost fatal cases of copper poisoning from melting copper under a direct arc. Suitable ventilation should be provided, and masks or respirators supplied, at least until it has been shown that, under the conditions under which the furnace will be operated, there is no possible danger from poisoning. The power consumption is calculated to be about 300 kw. h. per ton in a 25-ton direct-arc furnace. No commercial application of the electric furnace to the melting of cathode copper appears to have been made. The superintendent of a large copper rolling mill said in 1917 that the remelting of cathode copper, even without loss of conductivity, must be done at a cost of not over \$1.50 per ton; so that power would have to be considerably cheaper than one-half cent per kw. h. to make electric melting possible.

Baily says²⁹ that cathode copper can be melted electrically without blowing or poling and with only the sulphur from the copper-sulphate electrolyte contained in the cathode as charged and cites two heats in a Baily 105-kw. furnace. The cathodes were sheared to separate the edges, which occlude more electrolyte, from the centers.

²⁷ Lyon, D. A., and Keeney, R. M., Melting cathode copper in the electric furnace: *Am. Inst. Min. Eng., Bull.* 92, August, 1914, p. 1791.

²⁸ Lyon, D. A., and Keeney, R. M., Smelting copper ores in the electric furnace: *Am. Inst. Min. Eng., Bull.* 80, August, 1913, p. 2117.

²⁹ Baily, T. F., Resistance-type furnaces for melting nonferrous metals: *Chem. and Met. Eng.*, vol. 21, 1919, p. 11; *Trans. Am. Electrochem. Soc.*, vol. 35, 1919, p. 411.

Quality of copper melted in a Bailly furnace.

	Charge.	Analysis.			Conductivity.
		Cu.	O.	S.	
Heat A.....	Cathode edges.....	99.904	0.084	0.0021	Per cent. 100.1
Heat B.....	Cathode centers.....	99.950	0.036	0.0012	100.8

Bailly says that the large resistor type of furnace tried for zinc cathodes will be used for melting cathode copper. The contrary opinion has already been cited of a copper rolling mill which has used a 105-kw. Bailly furnace for brass, but does not consider the resistance type worthy of consideration for large-scale copper melting. Some interest, however, has been shown by the larger melters of cathode copper in the possible adaptation of General Electric and Foley furnaces for this purpose.

The General Electric furnace has been operated at the United States Copper Products Co. on new copper with some heavy tube scrap. The quality of the product is said to be satisfactory when no metallic impurities are present and the copper is poled prior to phosphorizing. The furnace is said to be much less satisfactory for pouring a pure, tough pitch, unphosphorized copper.

A Detroit rocking indirect-arc furnace has been used for melting cabbaged copper wire and scrap copper into wire bar, and its users state that metal of 100 per cent conductivity, Mathiesen scale, is produced without the use of fluxes or deoxidizers. The charge must, of course, be free from metallic impurities. The Bennett contact-resistance furnace is also said to be satisfactorily employed in melting pure copper.

The electric furnace has a big advantage over crucibles for melting brass in that it can readily produce equal quality in larger melting units, but it is at a corresponding disadvantage in melting cathode copper in that no electric melting furnace has yet been built with a capacity of over 40 tons of molten steel, and few, if any, with over 10 to 15 tons capacity have been built to melt cold stock, whereas reverberatory copper furnaces often take 200 tons and upward per charge.

Moreover, it is stated⁸⁰ that recent advances in reverberatory practice have allowed the use of high-sulphur oil for reverberatory melting of cathodes without trouble from sulphur in the wire bars.

The upkeep of the refractories in a direct or indirect arc furnace, the only types likely to prove efficient enough in the use of power

⁸⁰ Hydrometallurgy and pyrometallurgy of copper in 1919, Chem. and Met. Eng., vol. 22, 1920, p. 7.

unless induction furnaces are developed enough to be applicable, is almost certain to be more difficult than in a fuel-fired reverberatory, and until this factor can be overcome the large-scale electric melting of cathode copper does not seem likely in the near future.

CHOICE OF A FURNACE.

In choosing an electric furnace for a given plant, or in selecting two furnaces for comparative test, the first thing to consider is the alloys to be melted, so that the furnace chosen may be metallurgically fit for the work in hand and for the range of alloys. For nickel chromium and Monel metal one can use the Heroult, Snyder, Greaves-Etchell, or the Rennerfelt, and possibly the rocking type or the General Electric. The two latter would be at a disadvantage because the refractories ordinarily used in them are not meant for use at such high temperatures. The possible application of the direct-arc type to pure copper depends on the true answer to the yet unsolved question as to whether there will be poisoning of the workmen by volatilized copper. The General Electric furnace, though used on copper, begins to have considerable trouble in maintaining the roof on account of the high temperature. On gun metal one can probably use any of the furnaces. On "85-5-5-5," the point in zinc content at which a direct-arc furnace will not cause too great loss of zinc, has probably been passed.

On 74 copper, 4 tin, 5 lead, 17 zinc one could use any type save the direct-arc, but the Rennerfelt begins to have trouble through volatilization of zinc, and the lead content is near the upper limit for safety of the present lining of the Ajax.

Another point demanding consideration on alloys high in lead or on yellow brass is whether a thorough mixture of the alloy is essential. If it is, the rocking type on the high lead alloys and the Ajax and rocking furnaces on yellow are indicated, as against the types that require hand stirring to insure thorough mixing.

When the furnaces that are entirely fit metallurgically are chosen for a given alloy, the metal losses should be low, and practically equal in all the fit types, so that the choice of an individual furnace rests on other factors than the metal loss, though that may often be the determining factor in choosing electric furnaces as a class to replace fuel-fired furnaces.

The range of alloys to be melted in the same furnace must be considered. For example, a firm making both ordinary red brass and yellow brass could melt the former in a Rennerfelt and the latter in an Ajax, and if it had sufficient production to keep the Rennerfelt always working on red and the Ajax always on yellow, it would have a good combination in the use of both furnaces. But if the same

furnace must be able to melt a heat of red, followed by one of yellow, neither furnace will do, and the choice must be made from the furnaces that will handle both alloys. The metallurgical fitness of the various furnaces is shown in Table 54.

When the possible furnaces have been separated from those that are unfit on the basis of metallurgical ability and versatility, the choice can still be narrowed down further by considering the furnaces as to their fitness on other lines. A rolling mill or a smelting and refining plant that can operate 24 hours a day and can use almost any amount of metal any time it is ready to pour has a different problem from that of a small jobbing shop that plans to run but one shift per day and perhaps requires heats of various alloys and in various amounts and can not dispose of a very large heat at one time. The next thing to consider, therefore, is the daily production desired and the size of the heats needed. A further sorting of the furnaces may be made to select a type and a size of a given type suitable for the work. A rolling mill may desire to pour direct from furnace to mold and hence chooses a furnace from which such pouring can be done, and selects one taking a charge that will just fill a certain number of molds. The tendency there might be toward not too large sizes and toward the more compact furnaces, as pouring direct from these is more easily controlled. Another rolling mill may wish to break away from small charges or small ingots to see if economies can not be introduced by large scale units, so that they choose the furnace taking the largest charge from among the otherwise suitable ones.

TABLE 54.—*Applicability of various types of electric furnace to various alloys.*

Alloys.									Types of furnaces.						
Fe.	Cr.	Ni.	Cu.	Sn.	Pb.	Zn.	Ag.	Al.	Direct arc.	Stationary indirect arc.	Moving indirect arc.	Reflected heat.	Contact resistance (Bennett).	Vertical ring induction.	High frequency.
	15	85							O. K.	O. K.	O. K.?	? 1	?	?	O. K.?
5		67	28						O. K.	O. K.	O. K.	? 1	?	?	O. K.?
		25 5	75 95						? 2	O. K.	O. K.	O. K.	O. K.?	?	O. K.?
			100						? 2	O. K.	O. K.	O. K.	O. K.	No-3	O. K.
			90 88	10 10		2			O. K.	O. K.	O. K.	O. K.	O. K.?	? 3	O. K.
			80 76 68	10 2 4	10 18 26		3 2		O. K. 5	O. K.	O. K.	O. K.	O. K.?	No-4	O. K.
			85	5	5	5			No-5	O. K.	O. K.	O. K.	O. K.	O. K.	O. K.

TABLE 54.—*Applicability of various types of electric furnace to various alloys—Continued.*

Alloys.										Types of furnaces.						
Fe.	Cr.	Ni.	Cu.	Sn.	Pb.	Zn.	Ag.	Al.		Direct arc.	Stationary indirect arc.	Moving indirect arc.	Reflected heat.	Contact resistance (Bennett).	Vertical ring induction.	High frequency.
			74 80 70 67 61 65 60	4	5	17 20 30 30 38 34 40				No-5	No-5	O. K.	O. K.	O. K.	O. K.	O. K.
		18	58			24				No-5	No-5	O. K.	O. K.	?	O. K.	O. K.
			3 2 8 10 10			15 80 80 90 85		82 68 92 90 5		No-5 ? ? ? No	? 5 O. K.? O. K. O. K.?	O. K. O. K. O. K.? O. K.?	O. K. O. K. O. K.? O. K.	? ? ? ?	? 3 ? ? ?	O. K. O. K. O. K. O. K.?

O. K. —Furnace metallurgically satisfactory for this alloy.

? —No data on hand.

O. K. —Probably satisfactory, but no record of trial.

No —Furnace not satisfactory for this alloy.

1 —Temperatures probably rather high for good refractory life.

2 —Question of copper poisoning by fume from direct arc not yet settled.

3 —Induction furnace on this alloy would require resistor tube of special design.

4 —Lead too high for good life of usual lining of induction furnace.

5 —Furnace causes loss of volatile metal from this alloy.

A firm that can dispose of metal rapidly desires to use the furnace that will give the greatest production in the one, two, or three shifts they plan to work, and the high rate of production may be most important in relation to labor per ton of melt, to floor space, or to the original investment in the furnace.

ELECTRIC POWER AVAILABLE.

If electric power is cheap, convenience, low labor cost, or the avoidance of electrode cost, may indicate a choice of a furnace relatively inefficient in use of power. If power is dear, the more efficient the furnace is in the use of power the more likely it is to be preferred. If a plant is to take power from a small central station, the size and nature of the furnace load must be such that the central station will accept the load. Furnaces otherwise better fitted for the plant's need may therefore have to be passed over in favor of a furnace with electrical characteristics suitable for the location. Again, between two suitable furnaces, a choice may be made on the basis of upkeep cost, life and cost of lining, electrode consumption and breakage. It may be cheaper to use a very high-powered furnace and to replace refractories often in order to get a high rate of production, or it may be cheaper to use a low-powered, inefficient furnace with a low rate of production to avoid refractory troubles; more likely, it is better to choose one that falls between the two extremes.

SPECIAL CONDITIONS.

Finally, after the most suitable electric furnaces have been selected, a comparison should be made for special conditions: The alloys melted, the number of shifts worked, the power and fuel costs, the labor costs, the upkeep costs, the metal losses, and their value, and the health and safety conditions of fuel-fired furnaces already in operation or chosen from the various available types and sizes of fuel-fired furnaces in a fashion similar to that employed in selecting the type and size of electric furnace.

In this comparison, the first cost of the electric furnace must not be forgotten, though it is often neglected in the comparisons of melting costs put out by electric furnace makers. One can either include an interest and depreciation charge per ton based on the sum of these items per year, divided by the yearly output, or he can omit it, and then figure, from his calculation of the savings shown by electric melting over fuel-fired furnace melting, how long it will take, at the rate of production of the particular electric furnaces considered, to pay for each of them and how much they will thereafter save him.

On account of the initial cost and the interest charge one should not install so many electric furnaces that some will stand idle most of the time, being there simply to handle small peaks of production in excess of the normal; for excess production a plant should utilize fuel-fired furnaces of lower initial cost.

It is impossible to draw up any accurate blanket comparison of melting costs in fuel-fired furnaces and electric furnaces. Melting cost in fuel-fired furnaces varies with a multitude of conditions, and so does melting cost in electric furnaces. When the varying conditions are known, a dollar-and-cents comparison is the most striking and convincing method possible.

But such figures are of little real value unless they are calculated for a particular rather than for a general case, and to give general figures in this report is deemed inadvisable.²¹ The cost figures given in the advertising literature of some furnace makers, therefore, will not be quoted. Instead, in the discussion of the performance of the various furnaces, such data as could be collected have been given as would allow an individual plant to make the calculation of melting costs for its own conditions.

When electric brass melting is a little older, valuable comparisons can be made of the performance of two or more types and sizes of electric furnaces in the same plant, as then the conditions of operation of both types can be made similar. A number of firms have two or more types of electric brass furnaces, and one firm has tried

²¹ Compare Electric furnace progress, *Metal Ind.* vol. 17, 1919, p. 280.

five makes of electric furnaces, but it is unusual that more than one type has been installed long enough so that accurate data are available on the different types or else the different types have not been tested on the same classes of work. Valuable, accurate, comparative data on performance should become increasingly available from these plants in the future.

After a proper type and size of electric brass furnace has been chosen to meet the specific conditions of a particular plant, the furnace should be installed and operated to the best advantage.

INSTALLATION OF AN ELECTRIC FURNACE.

There are certain points in the installation which require the attention of some one with a knowledge of electrical engineering, as applied to electric furnaces, rather greater than that possessed by the ordinary firm of electrical contractors, so that the advice of either the furnace maker or the central station should always be taken rather than to depend entirely on a plant electrician or on a contractor. These points mostly center around the fact that carrying alternating currents of, say, 2,000 amperes is a different matter from that of carrying the much smaller currents of the ordinary power or lighting circuit.

Compared to the problems²² involved in carrying currents of 20,000 amperes often required by large ferro-alloy or iron-smelting furnaces, or even current of 5,000 to 10,000 amperes, as taken by large steel-smelting furnaces, the problems with the brass furnaces, even the largest, are fairly simple.

For example, in one of the Swedish iron-smelting furnaces, taking 3,000 kw., the losses between the transformers and the furnace were 475 kw. with one method of running the leads from transformers to furnace, while with another the losses were cut to 285 kw., the power factor being at the same time brought up from 70 to 90.

Beside the actual saving of energy, the change made 3,500 kv. a. of transformer capacity capable of sending more energy to the furnace than a capacity of 4,500 kv. a. sent before.

In another 4,000-kw. furnace, 490 kw. were lost between transformers and electrodes, 165 kw. in the leads themselves and 315 kw. in the structural ironwork near those leads. Power used in heating up

²² Compare Lindström, A., Leads for electric furnaces, *Met. and Chem. Eng.*, vol. 16, 1917, p. 683. Holmgren, F., Problems in electric furnace smelting, *Chem. and Met. Eng.*, vol. 22, 1920, p. 165. Anon., Low tension conductors for electric furnaces, *Elec. Rev.*, vol. 76, 1920, p. 287. Meyer, A. A., Electrical characteristics of electric furnaces, *Trans. Am. Electrochem. Soc.*, vol. 31, 1917, p. 97. Flinterman, R. F., Electric steel castings, *Trans. Am. Electrochem. Soc.*, vol. 33, 1918, p. 263. Anon., Frequency for and capacity of electric furnaces, *Elec. Rev.*, vol. 75, 1919, p. 1069. Editorial, Transformers for electric furnaces, *Elec. World*, vol. 75, 1920, p. 880.

leads or nearby ironwork never gets into the furnace, but it has to be paid for just the same.

In a Heroult steel furnace of about 1,300 kw. it was found that the voltage drop in one of the leads passing near some structural steel was over three times the drop in another similar lead more remote from the steelwork. The leads were originally made up of bus bars held apart to allow ventilation and radiation of heat by copper spacers, which made the whole lead essentially one big conductor instead of several separate conductors in parallel. By merely taking out the copper spacers and inserting insulating spacers the voltage drop was decreased, and the furnace was enabled to do more work and to make its heats faster and at a lower power consumption.

All this peculiar behavior of leads carrying heavy currents is due to the fact that alternating current travels in waves, first in one direction and then in the other. The more often the waves travel, the greater the complications.

With direct current, such as we have in storage batteries, electroplating, etc., the current travels steadily in one direction. But direct current can not be transformed from one voltage to another and hence must be generated at the voltage at which it is to be used. It must, therefore, be transmitted at low voltages and high currents. Alternating current, on the other hand, can be generated at many thousands of volts and transmitted over long distances over tiny wires at very low currents. Since the heating of a conductor depends on the current and not the voltage, and increases as the square of the current, this high-voltage, low-current transmission at low currents means that there is a huge saving both from the diminution of energy losses and from the decreased cross section of conductor required.

When the high-voltage alternating current has been brought to the point where the power is used, it is merely run through a transformer that can change it to the lower voltage desired, at a correspondingly higher current. For this reason all electric furnaces that need merely heat, and not the electrolytic action of the direct current, use alternating currents.

With direct current the same number of amperes of current passed through the same cross section of a copper conductor gives the same heating effect no matter what the shape of the conductor or whether it is near other conductors or not.

Direct current will magnetize nearby iron or steel, but it does not cause in them the losses due to inductive or eddy current that are caused by alternating current. With alternating current, where the current changes in magnitude from zero through the maximum to zero again, as well as in direction, 60 times a second on 60-cycle current, and 25 times on 25-cycle, and the changes in the current bring in

the effect of "reactance" or an inductive effect. With direct current a conductor opposes the flow of current by resistance only, while with alternating current it opposes that resistance plus reactance as well, and reactance is affected by the size and shape of the conductor and by the proximity of other conductors and of iron or steel. If to a big electric furnace for making calcium carbide, for example, carelessly installed, were supplied, in order, three different sorts of current at the very same voltage, the furnace might take 5,000 kw. on direct current, 4,000 on 25-cycle, and 2,500 on 60-cycle alternating current. By proper subdivision and interlacing of the leads, and by avoidance of nearby iron, the last two figures might be brought up much nearer the first.

It is not necessary to go deeply here into the various effects of reactance, such as the power factor, skin effect in conductors, eddy currents, and other losses in near-by iron or steel here, but it is worth while, however, to point out that these factors exist, and that disregard of them may mean that the cost of the leads between transformers and furnaces may be much higher than is necessary, and that avoidable losses of energy, costing as much per unit as energy usefully employed in the furnace, may take place every day the furnace is used.

The foundryman must trust the furnace maker or the central station to specify the proper size and shape of the leads and to see that they are properly installed. He should be sure that the knowledge of and experience with heavy current conductors of whomever he chooses to trust is adequate.

Generally speaking, the shorter the secondary leads are from high-tension transformer to the furnace and the farther they are from girders and structural ironwork the better. The ideal arrangement is to have the high-tension switchers, meters, etc., in a properly inclosed room, kept locked against intrusion in order to prevent danger to life from high-voltage current, with remote control switches on the furnace switchboard for throwing off and on high-tension current.

A fatal accident occurred on a direct-arc furnace of rather higher voltage than the most modern form, on which the primary switch was not controlled from the furnace. A workman was putting the furnace into shape after relining it and stood so that his body made contact between upper and lower electrodes. A fellow workman evidently became confused as to which of two switches at some distance from the furnace was the one that controlled a circuit that he wished to close, and he closed the switch to the furnace, and the man working on the furnace was killed. Under working conditions there is usually no danger whatever in an electric brass furnace properly installed, but

even the low voltages may be dangerous, as even a low-voltage, high-current circuit closed through the body may bring serious results. The safe way is to open the high-tension circuit entirely when one wishes to turn the current off, and to do this from a control board at the furnace itself.

All the commercially used electric brass furnaces can be installed so as to be safe.

The transformers themselves should be placed outdoors wherever possible, or else in special transformer rooms or cells. If it can be avoided, high-tension current should not come into the foundry itself.

To get short secondary leads, the furnaces are placed preferably close to a wall, with the transformer directly back of the wall on the other side. The higher the current in the secondary leads the more important it is for them to be short.

Besides the losses of energy in the leads there are losses in the transformers and in the electrodes, but they depend on proper materials, design, and proportioning, for which makers of the transformers and the furnaces are responsible.

As soon as a furnace is installed a comparison should be made between the reading of the kw.-h. meter on the primary, back of the transformer and leads, and the reading of that on the furnace switchboard by which the furnace is operated, if this, as is usually the case, is connected up on the secondary side and shows the energy that gets to the electrodes, but does not include losses in transformer and leads. If such a comparison shows not much over 5 per cent loss between the power on the primary side, that paid for, and that on the secondary, the installation is pretty good. A loss of 10 per cent would not be unusual if the secondary leads are very long. Higher losses would indicate that the installation could probably be improved.

REFRACTORY LININGS.

The first refractory lining for a furnace is usually supplied by the furnace maker. After that is worn out the user may and usually does, sooner or later, try some other lining, and he may experiment with all sorts of refractories and with large bricks, or even one-piece liners, with small bricks, and with rammed-in linings.

On account of changes in prices of refractories and varying freight rates, different refractories may be called for in different localities or in the same locality at different times. Each type of furnace has its own peculiar requirements, and the nature of the charge, the amount and nature of slag-forming nonmetallic impurities, and the way the furnace is run all have a bearing on the refractory problem.

The ideal lining would never wear out, would allow no heat to escape through it, and would itself take up no heat. No lining is

ideal in any one of these points. Each factor is of about equal importance. Long life is desirable, and the less often the furnace is down for relining the lower the cost, not only of refractories and of labor for laying them, but also of the overhead on a nonproductive furnace. But long life may very possibly be attained at too great an expense for expensive electric heat lost through the lining, and through lowered production due to this loss of heat. A lining of low-heat conductivity is more necessary in an electric furnace than in a fuel-fired one, so the lining is usually thicker than in fuel-fired furnaces of similar size for this reason.

But if an electric furnace is to be used only, say, 9 or 10 hours per day, the heat storage in a thick lining may be too great. The inside of the furnace lining must be heated up to or above, depending on the type of furnace, the pouring temperature of the metal before the metal can be poured. The temperature varies through the lining from inside to outside. Now, if the refractory has too high a heat conductivity, the lining must be heated nearly to this temperature for quite a distance back before it ceases to drain the heat away from inside, and before the inside gets up to full operating temperature. In some furnaces having great wall area in comparison to their metal capacity, huge amounts of heat must thus be drained from the inside and stored in the walls during the first two or three melts-of the day. This heat leaks away through the furnace shell when the furnace is idle nearly as fast as it does while the furnace is running until the lining gets pretty well cooled off, so the next morning the walls have again to be supplied with the heat they have lost through the night. On continuous operation the walls get "soaked full of heat," or the furnace has reached the "steady state," so that each heat comes out in the same time, and the energy supplied is used in useful work or is lost through radiation from the walls during the heat rather than in heating up the walls. On 9-hour operation on the first few heats of the morning it is necessary to supply not only the shell losses during that heat, but also energy to make up for the shell losses of the night before.

High heat storage therefore means low production and low thermal efficiency when the furnace is not run continuously. It might conceivably pay to make a furnace lining, say 2 inches thick and, if necessary, to water-cool the outside to keep the lining from melting on a furnace to be run but a few hours per day. If one lost an average of 20 kw. per hour for 24 hours with a thick lining, he has to supply 480 kw. h. in whatever time the furnace is run, say, 8 hours, 320 kw. of this being stored and lost at night. He could lose 80 kw. average per hour for 8 hours, and if the furnace had a storage of only 100 kw. h. he could not lose more than that in the remaining 16 hours,

or a total of 420 kw. h. He would then start with a dead cold furnace each morning, but he would be 60 kw. h. ahead on his day's run. St. John²² has shown several diagrams of the distribution of heat losses which bring out the important rôle played by stored heat.

It is the stored heat lost at night that cuts a large percentage figure in a small furnace and that prevents the usual types of externally heated crucible furnaces from having any chance of commercial usefulness. The lack of stored heat, due to the need for only a very thin heat-insulating wall, makes the high-frequency furnace with its internally heated crucible a possibility for intermittent work in small-sized furnaces.

The inside of the lining must be refractory enough not only merely to resist fusion but to stand up against the atmosphere, the metal, or the slag with which it is in contact.

But the most refractory materials commonly available have a high heat conductivity and a high specific heat; that is, they tend to give both high wall losses and high heat storage. If only such a thickness of this high-temperature refractory be used as will keep the temperature at its back down to some lower temperature at which some other less refractory material of lower heat conductivity and heat storage will stand up, the second refractory then becomes a better material for that layer of the lining than the first.

Past a certain thickness of the second refractory, a third material, still less refractory but of even higher resistance to heat flow and of lower heat storage capacity, then surpasses the second. Even more layers could be used.

Still better would be a brick that varied in composition and properties from one end to the other, as too many layers of very thin bricks give mechanical instability. Some distinct progress is being made in the experimental production of such bricks, especially carborundum fire-clay mixtures.

Nearly all electric furnaces use at least two layers of refractory material. The outside layer next to the shell is the easiest to select, as infusorial earth like "Sil-o-cel" and similar infusorial earth products have in high degree just the needed properties. Heavy asbestos sheets or asbestos cement are sometimes used.

A low-grade fire brick usually fills fairly well the requirements for the middle layer, but the inner layer is the real problem. Most electric brass furnaces use high-grade fire brick, especially those high in alumina. Some use silica brick. On account of the danger of spalling due to heating and cooling, which is shared by magnesite and silica, these materials are seldom used in furnaces for intermittent operation. Carborundum brick has been used in roofs and

²² St. John, H. M., Commercial testing of metallurgical electric furnaces, Chem. and Met. Eng., vol. 21, 1919, p. 388.

heating troughs. Its heat conductivity is rather high for use in the body of the furnace, though it finds some use for that purpose. Carborundum brick, especially the bonded type, has great resistance to spalling and abrasion. Hartmann and Kohler²⁴ show some striking photographs which bring out clearly the superiority of such brick, as well as of high-grade fire brick, as to spalling on cooling rapidly from 1350° C. over silica or magnesite brick, which spall very badly.

Chromite brick is of doubtful value in the furnaces that have a strongly reducing atmosphere, as it tends to be reduced into ferrochrome.

Zirkite brick, made from crude zirconia ore, is tantalizingly close to being a valuable refractory, but its price is too high to be justified until its properties are improved. When pure zirconia refractories become available they bid fair to give a most desirable combination of properties.

Alundum, or electrically fused alumina bricks, also have possibilities, though they are expensive and are not yet developed very far except for laboratory use.

Alundum cement, however, is of great value in some parts of some electric brass furnaces because of its combination of refractoriness, good bonding power, and its ability to remain an electrical insulator at temperatures at which other refractories become conductors.

Carborundum and alundum are, and pure zirconia refractories probably will be, themselves electric furnace products.

As laboratory experiments and still more important plant tests with improved refractories proceed, the life of electric furnace linings should be greatly improved, and relining costs per ton lowered. It is also hoped that a lining for the induction type can be found to allow that type to handle highly leaded alloys.

The makers are working with refractories of the type used for making graphite crucibles, with some hope of success.

Small amounts of heat may be theoretically saved by painting the furnace shells with aluminum paint or even by nickel plating them, as good reflectors lose less heat than dull black materials. Such a bright finish improves the appearance of a furnace and probably saves a few kw. h., as long as it stays bright, which is seldom very long in a foundry.

In comparing furnaces on the data so far available, preparatory to making a choice for installation in a given plant, the purchaser must consider the ability of the various furnaces to meet the plant conditions imposed, and the suitability of the furnaces as a load on the generating station that will supply the power.

²⁴ Hartmann, M. L., and Kohler, J. F., Physical characteristics of specialized refractories, *Trans. Am. Electrochem. Soc.*, vol. 37, 1920, pp. 349, 355.

CENTRAL STATIONS AND THEIR RELATION TO THE BRASS INDUSTRY.

Few brass foundries or small rolling mills are likely to extend their own generating stations, if they have any, but are more likely to buy their furnace power from a central station when they install electric furnaces. Hence the interest of the central station in electric brass furnaces as a means of extending their service is and should be keen. Not a little of the progress made in electric brass melting so far is due to the work of the central stations, although there are many central stations that have so far neglected the electric furnace as a load builder. Generally, however, the central station is glad to cooperate with the foundry and to have its engineers aid the foundrymen in the selection of a suitable furnace and in its proper installation.⁵⁵

A wise plan, followed by the more progressive stations, is to assign an engineer or group of engineers to specialization on electric furnace problems as they affect the central station. A little advice given by an electrical engineer of its staff, competent to handle the problems put up to him by the foundry, may result in the connection of a furnace or group of furnaces to the power company's lines, aggregating a kilowattage that it would take a good many installations of lights and motors to equal, and giving a load with a higher load factor and greater general desirability than the average industrial load.

As the central station is the usual source of power, it is of interest to note that the price of power varies with the amount used, so that under one condition the cost per kw. h. may be very different from that under other conditions in the same plant.

VARIATION IN COST OF ELECTRIC POWER.

Industrial power contracts usually have two factors, the demand charge and the energy charge. The first pays the power company for the equipment it must maintain to supply the maximum power needed; the second depends on the total amount of power used.

Suppose we have a maximum demand of 300 kw. and that the average power consumption per ton of metal is 335 kw. h. per ton on 9-hour operation, 275 on 18-hour, and 250 on 24-hour; the total power used per day is then about 2,000, 3,565, or 5,250 for the three cases. In a 25-day month this means 50,000, 91,000, 130,000 kw. h. per month.

Assume that the plant, before it installed its electric furnace equipment, had a maximum demand in lights and motors of 200

⁵⁵ Compare St. John, H. M., Commercial testing of metallurgical electric furnaces, *Chem. and Met. Eng.*, vol. 21, 1919, p. 377. Editorial article, Some practical data on operating electric brass furnaces, *Elect. Rev.*, vol. 76, 1920, p. 215. Wilcox, E. A., Electric furnace power from the standpoint of the central station, *Trans. Am. Electrochem. Soc.*, vol. 37, 1920, p. 589. Smith, A. C., The electric-arc melting furnace and the central station electric company, *Trans. Am. Electrochem. Soc.*, vol. 37, 1920, p. 701.

kw., and use 20,000 kw. h. per month for those purposes. Taking a concrete case where the power contract calls for a demand charge of \$1.80 per kw. per month for the first 50 kw. and \$1 per kw. per month for all over 50 kw., the energy charge schedule is: 2,500 kw. h. per month at 2 cents per kw. h.; for the next, 3,500 kw. h. per month at 0.8 cent per kw. h.; for the next, 310,000 kw. h. per month at 0.5 cent per kw. h.; all over 347,500 kw. h. per month at 0.4 cent per kw. h.

The 200-kw., 20,000-kw. h. lighting and motor power cost as follows:

$$\begin{array}{r} 50 \times \$1.50 = \$75.00 \\ 150 \times 1.00 = 150.00 \\ \hline \end{array}$$

225.00 Demand.

Total, \$415 or 2.07½ cents per kw. h. used.

The plant now has 500 kw. maximum demand and 70,000, 111,000, 150,000 kw. h. used per month on three assumptions.

$$\begin{array}{r} 50 \times \$1.50 = \$75.00 \\ 450 \times 1.00 = 450.00 \\ \hline \end{array}$$

525.00 Demand.

Energy—Case 1----- $2,500 \times \$0.02 = \50.00

$$35,000 \times 0.008 = 280.00$$

$$32,500 \times 0.005 = 162.50$$

$$\hline 70,000$$

$$2,500 \times \$0.02 = \$50.00$$

$$17,500 \times 0.008 = 140.00$$

190.00 Energy charge.

$$\hline 492.50 \text{ Energy.}$$

$$525.00 \text{ Demand.}$$

1,017.50 Total charge.

Less 415.00 Previous charge for lights and motors.

602.50 Total charge for electric furnace power.

$$\begin{array}{r} \$602.50 \\ \hline \end{array} = 1.205$$

50,000

The electric furnace power costs 1.205 cents per kw. h.

Case 2. There is the same demand charge, but the energy charge is—

$$2,500 \times \$0.02 = \$50.00$$

$$35,000 \times 0.008 = 280.00$$

$$73,500 \times 0.005 = 367.50$$

$$\hline 111,000$$

$$\hline 697.50 \text{ Energy.}$$

$$525.00 \text{ Demand.}$$

1,222.50 Total.

Less 415.00 Lights and motors.

807.50 Total charge for furnace power.

$$\begin{array}{r} 807.50 \\ \hline \end{array} = 0.89$$

91,000

The furnace power costs 0.89 cent per kw. h.

Case 3-----	2,500×\$0.02 =	\$50.00	
	35,000× 0.008=	280.00	
	112,500× 0.005=	562.50	
	<u>150,000</u>	<u>892.50</u>	Energy.
		<u>525.00</u>	Demand.
		<u>1,417.50</u>	Total.
	Less	<u>415.00</u>	Lights and motors.
		<u>1,002.50</u>	Total change for furnace power.

1,002.50

=0.79.

130,000

The furnace power costs 0.79 cent per kw. h.

Case 4. If the plant had three times the furnace installation figured above and used it 24 hours a day, or 900-kw. furnace demand (1,100 kw. total); 450,000 kw. h. per month furnace energy (470,000 total), the charge would figure:

50×\$1.50 =	\$75.00	\$2,370.00	Energy.
1,050× 1.00 =	1,050.00	1,125.00	
	<u>1,125.00</u>	<u>3,495.00</u>	Total.
2,500× 0.02 =	50.00	Less	415.00
35,000× 0.008=	280.00		Lights and motors.
310,000× 0.005=	1,550.00		
122,500× 0.004=	490.00		3,080.00
470,000			Total for furnace power.

2,370.00 Energy.

3,080.00

=0.685

4,500.00

The electric furnace power costs 0.685 cent per kw. h.

Tabulating these we get:

Case No-----	1	2	3	4
Hours -----	9	18	24	24
Number of furnaces-----	1	1	1	3
Kw. h. per ton-----	335	275	250	250
Power price, cents per kw. h-----	1.205	0.89	0.79	0.685
Cost per ton for power-----	\$3.92	\$2.45	\$1.98	\$1.71

This table shows the advantage of continuous operation of electric furnaces, as well as that of large installations. The exact figures will vary in each particular case, but the ratios will remain approximately the same.

The methods of determining the maximum demand vary in different localities. Often there is a penalty, a higher demand charge if the power factor is below a certain figure, usually 70 per cent, and sometimes a premium if it is over a certain figure. The methods of determining the power factor, for rate-making purposes, may also vary.

It is sometimes possible to obtain a lower rate by buying "off-peak" power; that is, by taking power at any time save for two or three hours per day when the power house has its heaviest load. By suitably arranging the melting schedule, a foundry could sometimes profit by this fact. The peak-load hours are different with different central stations.

Variations in the cost of coal to the power plant are sometimes a factor in the rates paid by the user as well as in the costs of the central station. The schedule of rates should be carefully studied and analyzed before choosing a furnace and in determining the best operating schedule when it is installed.

When electricity is generated by water power, it is bought under schedules somewhat similar to those on which steam-generated power is bought, but the demand charge is usually almost all-important while the energy charge is relatively small. That is, if one's maximum demand is 1,000 kw. it costs nearly as much for power to let the plant lie idle and use no power at all as it does to use 1,000 kw. 24 hours a day. The electrochemical manufacturers at Niagara Falls, for example, strive to keep their average demand just as close to the maximum demand as possible every minute.

The cost of hydroelectric power in this country, to large users who operate 24 hours a day, 365 days a year, as many of the electrochemical industries at Niagara Falls and other similar centers, will probably vary from \$20, perhaps less, to \$40 per kw. year.

Suppose a plant pays \$25 per kw. year and has a transformer capacity of 10,000 kw. It then pays \$250,000 per year for its power, whether at any time it is using the whole 10,000 or none at all.

The ratio of the average load in kw. to the kw. capacity of the plant is called the "load factor." If a 10,000-kw. plant averages only 5,000 kw. actual load, its load factor is 50 per cent and its power cost, when paid for on a kw.-year basis, is twice as much per kw. h. actually used as it would be at 100 per cent load factor. The load factor, therefore, is a most important consideration to electrochemical plants continuously operating. The load factor also plays a part, though not nearly so great a part, in the ratio of the "demand" and "energy" charges of the power bill for steam-generated electric power bought by the small user on a kw.-h. basis. These points are discussed by a number of writers.²⁶

²⁶ Steinmetz, C. P., Characteristics of electrical energy as affecting chemical industries, *Trans. Am. Electrochem. Soc.*, vol. 25, 1914, p. 29. Sothman, P. W., Notes on power transmission and transmission economics, *Trans. Am. Electrochem. Soc.*, vol. 25, p. 42, 1914. Fitzgerald, F. A. J., Some economies in the use of energy for electric furnaces, *Trans. Am. Electrochem. Soc.*, vol. 25, 1914, p. 53. Horry, W. S., Power for electric furnace work, *Trans. Am. Electrochem. Soc.*, vol. 25, 1914, p. 59. Longwell, H. E., Power problem in the electrolytic deposition of metals, *Trans. Am. Electrochem. Soc.*, vol. 25, 1914, p. 73. Newberry, F. D., Sources of direct current for electrochemical

Since, on the ordinary price schedule for steam power, the more power is used the lower the rate becomes, so that an extra furnace load added to an existing one really uses less expensive power than the earlier installation used, it is plain that firms who melt both steel and brass and already use electric steel furnaces, are in a favorable position to melt their brass electrically at low cost. Similarly, after a brass and aluminum foundry has turned completely to electric brass melting, the price of the extra power needed to melt also its aluminum electrically may be low enough that the electric melting of aluminum may be more attractive than otherwise.

Berlin,⁸⁷ has discussed fully the various factors entering into rate schedules. He considers the problem of tandem operation of two furnaces, in order to decrease the maximum demand, one furnace being poured and charged while the other is melting, so that power is only on one furnace at one time. He concludes that it will not pay on steel furnaces. It is possible, however, that it might pay on brass furnaces. One refining plant operating 24 hours per day plans to try running two 1-ton rocking furnaces in this way. Since a 1-ton brass furnace only has to melt about an hour per heat, and it usually takes half an hour to charge and pour, two furnaces thus run would produce 24 tons per day instead of 30 to 32 tons. The power cost per ton will probably not be materially different from that under unrestricted use of both furnaces. As only one transformer will be needed, the first cost of the equipment would be somewhat decreased though not in respect to output. The labor cost is expected to be decreased. Whether this will pay or not can only be told by trial. It would certainly not pay on other than 24-hour operation.

In order to fit the operation of the furnace to the particular schedule on which power is bought, in order to secure power at the lowest cost per ton of product and to operate it to the best advantage, the behavior of the furnace under different operating conditions must be accurately known and is best determined by thorough tests.

TESTING ELECTRIC FURNACES.

A complete test of an electric furnace is as important a matter to the foundry as a complete boiler test is to a power plant. By such a test the possibilities and limitations of the furnace may become very much clearer and better understood.

processes, *Trans. Am. Electrochem. Soc.*, vol. 25, 1914, p. 85. Turnbull, R., Electric furnace regulators, *Trans. Am. Electrochem. Soc.*, vol. 25, 1914, p. 97. Discussion on all of the above, *Trans., Am. Electrochem. Soc.*, vol. 25, 1914, p. 106. Lyon, D. A., and Keeney, R. M., Electrometallurgical industries as possible consumers of electric power, *Trans. Am. Electrochem. Soc.*, vol. 28, 1915, p. 139. Beckman, J. W., The electrochemical possibilities of the Pacific Coast, *Trans. Am. Electrochem. Soc.*, vol. 28, 1915, p. 169.

⁸⁷ Berlin, W. G., Power problems from the standpoint of the furnace operator, *Trans. Am. Electrochem. Soc.*, vol. 37, 1920, p. 367.

The test resembles a boiler test in many ways, but it may be even more intricate and requires fully as much care. A few plants have made such thorough tests which show the complete distribution of the energy delivered and trace completely the losses of each component of the alloys melted. The tests gave information of value in showing how to operate and to control the furnace in regular practice.

St. John²⁸ has given a complete and useful outline of the way such tests should be conducted. The reader is referred to his article for full information. A few points, however, will be mentioned.

The energy required to melt a given charge depends first of all on the specific heat and heat of fusion of the mixture,²⁹ on its melting point, and how far above the melting point the metal has to be heated. No one would expect to melt Monel metal with as little energy as is required for yellow brass. The nature of the charge also has an effect. Neither all large ingot nor all fine borings will melt as quickly and easily as a judiciously chosen mixture of light and heavy material.

The presence of nonmetallic material, such as sand on gates, may make a great difference in the performance of the furnace, as it may leave a layer of dross or slag over the metal, which in many types of furnaces baffles the flow of heat to the metal and requires more time and more energy to finish the heat. Large amounts of oil make it impossible to close the furnace tightly until it is distilled off, and the soot from its decomposition blankets the charge.

Hence one plant melting a charge of dirty borings and sandy gates, another one of large clean ingot, and a third one of medium-sized scrap, may for an alloy of exactly the same composition in the same type of furnace require quite different amounts of energy. Figures for power consumption, therefore, should be accompanied by a description of the charge.

The pouring temperature is also important. One may cast phosphor bronze into very large gears or into ingots at not much more than 50° F. above its melting point, or one may cast it into small bearings, many on a gate, at 550° F. above its melting point. Smaller variations are met on other alloys, but a smelting and refining plant making ingots only, and a foundry making tiny saddlery hardware or lock parts, will pour their metal at widely differing temperatures.

²⁸ St. John, H. M., Commercial testing of metallurgical electric furnaces, *Chem. and Met. Eng.*, vol. 21, 1919, p. 377.

²⁹ Compare Richards, J. W., The electric power required to melt metals, *Trans. Am. Brass Founders' Assn.*, vol. 4, 1910, p. 95. Clamer, G. H., and Hering, C., The electric furnace for brass melting, *Trans. Am. Inst. Metals*, vol. 6, 1912, p. 95. Hansen, C. A., Electric melting of copper and brass, *Trans. Am. Inst. Metals*, vol. 6, 1912, p. 110. Gillett, H. W., and Norton, A. B., Approximate melting points of some commercial copper alloys, *Tech. Paper 60*, Bureau of Mines, 1913, 10 pp.

Thornton and Hartley⁴⁰ calculate that heating brass to 1,100° C. would increase the gas consumption in a gas-fired furnace 20 per cent over that required to heat it to 1,000° C. Determination of the pouring temperature is essential in comparing the performance of different furnaces on the same alloy or the same furnace on different alloys.

Pyrometers for this purpose are now available⁴¹ and one should be used in every electric brass furnace test of any pretensions to accuracy and completeness.

METERS AND RECORDS.

It is so easy to measure everything one wishes to know about the electric power used in a furnace that there is little excuse for incomplete tests. Every furnace should have at least its individual kw. meter to show the rate at which the furnace is taking power at any moment, and its own kw. h. meter, readable to 1 kw. h. The latter is sometimes left out of the equipment to save its cost, as the central station's kw.-h. meter, on which the power bills are calculated, is supposed to take its place. But this is poor economy, for without a kw. h. meter on each furnace one can not know just how much energy the furnace has used in a given period unless no other load is metered by the same meter. Unless a kw.-h. meter is provided for each furnace, and located on the furnace switchboard, the furnace has to be controlled on a time basis, or by opening it and estimating the temperature of the melt. In most cases far better control is obtained by running on a kw.-h. schedule that has been worked out on the basis of experience, very close temperature control being possible on 24-hour operation, and very good approximations on single-shift operation.

With the induction furnace an ammeter may be substituted for the kw. meter, but the individual kw.-h. meter should be used in that case also. On many types of furnaces a voltmeter and an ammeter for each phase are also required, and a direct-arc furnace might well use power-factor meters also. Graphic meters are of value for permanent records.

The electric furnace, due to the ease of metering, offers, as Miller⁴² points out, every opportunity for the easy recording of operating data, so that if the furnace falls behind its standard performance for

⁴⁰ Thornton, H. M., and Hartley, H., Metal melting, *Jour. Inst. Metals (British)*, vol. 17, 1917, p. 217.

⁴¹ Compare Gillett, H. W., Pyrometers for molten brass and bronze, *Trans. Am. Inst. Metals*, vol. 8, 1914, p. 340. Editorial, Pyrometers for brass and bronze makers, *Chem. and Met. Eng.*, vol. 21, 1919, p. 114. Kinnison, C. S., An alloy which resists molten brass and bronze, *Chem. and Met. Eng.*, vol. 21, 1919, p. 608.

⁴² Miller, D. D., Electric furnace for heating nonferrous metals, *Jour. Am. Inst. Metals*, vol. 11, 1917, p. 271.

even a single heat the cause can be sought, found, and remedied more easily than with most fuel-fired furnaces.

METAL LOSS TESTS AND CALCULATIONS.

Metal loss is perhaps the most difficult thing to ascertain in testing any brass furnace, in making records of its operation, or in calculating its money value as an item in melting costs.

In testing the furnace itself the important points are the primary recovery, the percentage of the total weight of metal charged that is obtained in the ladle, the secondary recovery from the slag or dross, and the net loss. Also, how does the value of the primary recovery, the secondary recovery, and the net loss compare with that of the original charge?

For a complete answer both the total weight and the composition of the original charge, the metal in the ladle, and the amount of the secondary metal recovered must be known in order that the net loss—the metal which is gone and can neither be weighed nor analyzed—may be known accurately.

Various conditions often make it hard to attain the desired accuracy. If the charge contains sandy gates or oily, dirty borings it is difficult to be sure that a sample of either used for the determination of the nonmetallic materials is representative, and such a determination is often only an approximation.

On a hearth-type furnace one has to contend with an absorption of metal, often a preferential one as to lead, for example, in a fresh hearth. There is sometimes slight retention of metal in the hearth from heat to heat, requiring great pains in scraping out the furnace and taking up time that would not normally have to be taken when making successive heats on the same alloy. Extra pains to get accurate figures on metal loss may thus delay the operation of the furnace and influence its production and power consumption figures, thus sacrificing accuracy on that part of the test. To get really accurate figures, a metal loss test on a hearth type furnace should extend over a considerable number of heats.

All spill and spatter in pouring must be collected and weighed up for each heat. When the metal is in the ladle the question arises as to how it should be weighed, whether by difference between weights of empty and full ladle,⁴⁸ which involves inaccuracy in the weighing of hot objects and requires care lest slag, dross, and skulls be weighed as something they are not, by pouring into ingots and weighing those plus the pouring spill and spatter, which means that in a sand-castings shop the metal can not go into regular production, or by weighing as castings, in which case difficulties arise due to the necessity

⁴⁸ Compare Tonkin, E., Stand for checking melting losses, Foundry, vol. 45, 1917, p. 20.

for keeping all the castings, gate, spill, and overmetal from a heat together, and to the presence of adhering molding and core sand if the rough castings are weighed or to the loss of metal in cleaning up if the cleaned castings and gates are weighed.

Metal loss figures obtained by comparing metal bought with product shipped, though valuable for cost accounting, do not show furnace performance, for they include the remelting of gates and defective castings.⁴⁴

Pouring into ingots undoubtedly gives the most accurate figure.

Some people, like Collins,⁴⁵ prefer to calculate their metal losses from chemical analysis. Such calculation is a useful check when analysis is to be made anyway, but it is cheaper, and in general, far more accurate, to weigh in the ladle than to analyze. Particularly with furnaces that do not mix the metal automatically, calculation from analysis is unsafe, as there is danger of segregation in the molten metal. Added to that is the possibility of further segregation in the solid metal with attendant possibility of nonrepresentative sampling. On top of these is the inaccuracy of chemical analysis; it takes only a difference within the ordinary analytical limits of error to calculate to a material difference in weight. St. John⁴⁶ points this out and shows that though chemical analysis is useful in checking up the value of the lost metal, to calculate metal losses wholly on analysis involves the fallacy that none of the element used as a basis for the calculation, usually copper, is lost; the fact is, some copper is really lost by oxidation and slagging. This loss may be very little in an electric furnace, but it may be greater in an open-flame oil furnace with which the electric is being compared as to metal losses.

This whole matter has been rather thoroughly discussed,⁴⁷ and it has been shown that while some mills charge up metal loss as having the same value per pound as the charge, and others charge it up at the value of zinc only, neither is correct, the lost material being made up of all the constituents of the alloy, with the more volatile ones preponderating.

The volatilization loss in a fuel-fired furnace is largely zinc and lead and could be mostly made up by the addition of more zinc and lead. It is not fair to the fuel-fired furnace, therefore, to charge it with the full price of the alloy. Neither is it fair to the electric fur-

⁴⁴ Compare Walter, C. M., Metal melting by high pressure gas: Jour. Inst. Metals (British), vol. 17, 1917, p. 197.

⁴⁵ Collins, E. F., Melting some nonferrous metals and their alloys in the electric furnace: Jour. Cleveland Eng. Soc., vol. 11, 1919, p. 311.

⁴⁶ St. John, H. M., Commercial testing of metallurgical electric furnaces, Chem. and Met. Eng., vol. 21, 1919, p. 384.

⁴⁷ Discussion on paper by Guenther, E. B., Use of producer gas for melting yellow brass: Trans. Am. Inst. Metals, vol. 8, 1914, p. 320.

nace to assume that all the metal saving it shows over the fuel-fired furnace is cheap zinc and lead, when some of it may be copper and tin.

Complete data on losses by weight and on the composition of the lost metal must be obtained if the figures on the value of the metal lost or saved are to be truly accurate.

In considering the metal savings of the electric furnace one must take into account the fact that not only is the net loss usually reduced, but the gross loss is also generally reduced. In fuel-fired practice metal spilled into the ashes or running out of a pot broken in the fire and that entrained in the heavy slag of most open-flame oil furnaces has to be recovered. This metal must be charged with the cost of its recovery and of its remelting if it is pure enough to be remelted direct, and must be charged with the cost of the bad effect of any harmful impurities it picks up.

If it is not suitable for remelting, but must be refined after its recovery and concentration, it must bear the refining charge. In this last case often the only metal recovered after refining is a low-grade copper, all the other metals having been lost.

One brass mill on a run of 14½ tons of 68 per cent copper, 32 per cent zinc in pit fires, had a gross loss of 4.5 per cent, recovered 3.1 per cent from the ashes, giving a net loss of 1.4 per cent. This run was made in 1917 when metals were very high, copper being figured at 30 cents and zinc at 14 cents, making a pound of the brass cost 24.9 cents for its contained metal. The 3.1 per cent of recovered metal was valued by this mill at this time at only 16 cents per pound, after paying recovery charges, so that the drop in value was 7.9 cents per pound, or 24.5 cents per 100 pounds melted.

The money loss from the deterioration and cost of recovery of the recovered metal was, then, greater than the value of the net loss taken as all zinc, and 70 per cent as much as the value of the net loss if this were taken as composed of the same proportion of copper and zinc as the alloy itself.

Several other mills operating crucible fires on yellow brass calculate that in coke-furnace practice 1½ per cent of the charge is spilled into the ashes, and of this 60 per cent is recovered—that is, 0.9 per cent of the original charge is recovered and 0.6 per cent is lost. This 0.9 per cent recovered is said to be worth only 75 per cent of the value of its copper content and, besides, must be charged with the cost of its recovery.

On the basis of the value of the liquid metal obtained ready to pour, the electric furnace generally shows as much money saving due to its lower gross loss as it does due to its lower net loss. Both savings effect real conservation.

ELECTRIC BRASS FURNACES AS A FACTOR IN NATIONAL CONSERVATION.

There are many points in which electric brass melting deserves consideration as a factor in national conservation, and it is its importance as such a factor which has drawn and held the attention of the Bureau of Mines to the general problem.

METAL SAVINGS.

As has been shown above, the electric furnace effects material saving of the metals melted, both by reduction of the net loss and of the gross loss. The possible saving, when the brass industry adopts electric melting as the standard method, will be well worth while, even though the country's resources are large in all the metals, except tin, that enter into brass and bronze. The labor and the transportation facilities required for the production of some 6,000 tons of metal that could be saved by electric melting could then be applied to other needs.

SAVING OF CRUCIBLES.

The avoidance of crucibles not only cuts out a large item of expense but helps to make the United States independent of Ceylon graphite and Klingenberg clay, the shortage in which was so acute during the war that the Bureau of Mines took up the study of the domestic supply of crucible graphite and clays.⁴⁸ The carbon and graphite electrodes and other furnace parts required are made from domestic sources, as graphite electrodes are best made of artificial graphite, itself an electric furnace product.

SAVINGS IN LABOR AND HEALTH.

The reduction of the amount and severity of the labor required and the improvement in the health and safety of workmen due to the comparative coolness and freedom from deleterious fumes and to a reduction in the hazard of burns are as truly conservation as are the more material factors.

SUBSTITUTION OF FUELS.

It is in its bearing on that part of the conservation problem relating to fuel that the electric brass furnace is perhaps of the greatest ultimate importance.

Even under existing conditions, for the electric furnace to use steam-generated power helps the fuel situation to some degree, as it allows the substitution of bituminous coal, burned in the power plant, for coke, anthracite coal, or fuel oil, burned direct at the foundry in

⁴⁸ See White, U. B., Principal Investigations of the Bureau of Mines, Mimeographed report, June, 1919, pp. 14-26.

fuel-fired furnaces. It is true that in present steam-plant practice the by-products of bituminous coal useful for dyes, explosives, and fertilizers are not recovered; but, as Rideal⁴⁹ points out, these can be saved if the steam plant goes through the producer-gas step and enters the by-product chemical industry.

This question, however, has been investigated by the British Ministry of Munitions,⁵⁰ and the conclusion is reached that under present British conditions by-product recovery is not economical.

At any rate, there is a saving in distribution between the shipment of coal to a large central station and the distribution thence of electric power to foundries, over the shipment of foundry fuel to each foundry, with the trackage, switching equipment, and tying up of rolling stock involved.

SUBSTITUTION OF OIL.

Of real importance is the substitution of steam-generated electric power for direct oil-fired furnaces. Great as are our oil resources they are all too small for the imminent future demands upon them for gasoline, lubricants, and especially for fuel for the Navy and merchant marine. The oil-burning ship, whether it burns oil to raise steam or uses it in Diesel-type engines, is the modern ship. The Navy must have oil and the merchant marine should have it. Just as the electrification of railways could reduce the need for the oil-burning locomotive, so the electrification of brass furnaces could cut down the need for oil-fired furnaces.

Folsom⁵¹ points out that to supply our new merchant fleet it is imperative that the consumption of oil on land be reduced.

The Director of the United States Geological Survey says:⁵²

The position of the United States in regard to oil can best be characterized as precarious. * * * The new demand of our shipping program alone involves fuel oil in quantities equivalent to nearly one-half of the present domestic output. A country-wide thrift campaign needs to be waged. * * *

The last 100 years has resulted only in the exhaustion of less than 1 per cent of the country's coal resources, while in the 60 years since our oil production began 40 per cent of the available oil has been brought to the surface and consumed.

The former Director of the Bureau of Mines⁵³ also pointed out the seriousness of the domestic oil situation.

⁴⁹ Rideal, E. K., *Industrial electrometallurgy*, 1919, p. 19.

⁵⁰ Report to British Ministry of Munitions, Direct firing and by-product power production, *Iron Age*, vol. 105, 1920, p. 1234.

⁵¹ Folsom, D. L., Fuel oil for ships after the war, *Am. Inst. Mining Eng., Bull.* 144, December, 1918, p. xvi. Rear Admiral Dumas, British Navy, quoted in *Eng. Min. Jour.*, vol. 104, 1920, p. 249.

⁵² Smith, O. G., A foreign oil supply for the United States, *Mining and Metallurgy*, No. 157, January, 1920, sec. 4.

⁵³ Manning, Van H., International aspects of petroleum industry, *Mining and Metallurgy*, No. 158, February, 1920, sec. 24.

A careful analysis of the situation has been made by Gilbert and Pogue,⁵⁴ and they conclude that the sooner the enlarging use of fuel oil for steam raising and heating be turned into a narrowing use the better.

It seems evident that the substitution of oil by coal-generated electric power is a move in the right direction.

SAVING OF COAL.

By reason of the efficiency of the electric furnace, it is possible also to effect a saving in the actual number of heat units required to melt a given amount of metal.

If the power is generated either from coal, through a steam boiler and turbo-generator, or through a gas producer and gas engine, from gas tar through a Diesel-type engine and generator in a reasonably efficient, large, central power plant, the efficiency of the thermally more economical of the available electric furnaces, if properly operated, is so good that electric melting may require fewer B. t. u. in the power plant fuel than would be needed for melting in fuel-fired furnaces.

The most modern central stations can deliver to their customers 1 kw. h. for 25,000 B. t. u.—that is, the equivalent of 2 pounds of bituminous coal. Three pounds of coal may be taken as an average figure when less efficient central stations are included,⁵⁵ although both lower and higher figures are taken by various authorities.⁵⁶

In crucible-pit fires, 600 pounds of coke per ton of yellow brass melted, or 800 pounds per ton of red, is fair practice; that is, if yellow brass is melted at 200 kw. h. per ton and red at 265 kw. h. per ton, each kw. h. being generated by 3 pounds of coal, no more fuel is used than would be with direct firing.

It has been shown that these figures can be reached on 24-hour operation on the alloys to which they are fitted by the larger sizes of direct-arc, stationary, and moving indirect-arc, and by induction furnaces, each furnace being, of course, used on alloys to which it is

⁵⁴ Gilbert, C. G., and Pogue, J. E., *Petroleum, a resource interpretation*: Smithsonian Institution, U. S. Natl. Museum, Bull. 102, part 6, 1918; compare Rice, G. S., *Review of the coal situation of the world*, Bull. Am. Inst. Min. Eng., 133, 1918, p. 131. Arnold, B., *Conservation of oil and gas resources of the Americas*, Econ. Geol., vol. 11, 1916, pp. 203, 299. Requa, M. L., *Petroleum resources of the United States*, Sen. Doc. 363, 64th Cong., 1st sess., 1916. U. S. Navy, *Oil conservation, a national necessity*, Ann. Rept. Sec. of the Navy, 1916, p. 33. Dunn, F. B., *Industrial uses of fuel oil*, 1916. Editorial note, *Crude oil imports set up a new record*, Foundry, vol. 48, 1920, p. 298; Editorial note, *Fuel oil Prices*, Iron Age, vol. 105, 1920, p. 1101.

⁵⁵ Compare Benzinger, R. F., *Solving the ever-increasing cost of central-station operation*, *The Electric Furnace*, vol. 1, No. 2, p. 16. Cook, C. S., *Power production for electrochemical purposes*, Trans. Am. Electrochemical Soc., vol. 35, 1919, p. 67.

⁵⁶ Compare Report to British Ministry of Munitions, *Direct firing and by-product power production*, Iron Age, vol. 105, 1920, p. 1284. Volgesang, A. T., *St. Lawrence power*, Power, vol. 51, 1920, p. 397. Report of Committee, Am. R. R. Assn., *Mining and Metallurgy*, No. 161, May, 1920, p. 7.

fitted. They can probably not be reached by even the more efficient of the reflected-heat furnaces.

In a locality where the central station was so shortsighted as to disregard the advantages to itself in an electric furnace load, and to accept the tentative suggestion of an editorial in a recent electrical journal⁵⁷ that the value of the service rendered by electric furnaces was so high that they could stand being charged rates "approximately those for lighting service" (if the public service commission of the State did not, as it probably would, prevent the charging of such an unjust rate), the combination of high rates and an inefficient furnace would soon cause its user to change to a more efficient type or to go back to fuel-fired furnaces, just as happened in a steel foundry. This plant was using a Heroult electric furnace for steel castings, but when the cost of power rose to nearly 2 cents per kw. h.,⁵⁸ the electric furnace was sold and an open-hearth installed. The power cost was the item that necessitated the change, even at the expense of a slightly less high quality of steel, the open-hearth steel, in this case, being of good enough quality for the purpose in hand.

Whether or not economic conditions will allow the use of an inefficient furnace, on account of the need of fuel conservation, it is a waste to utilize a furnace less efficient in the use of electric power for any certain purpose where a more efficient one as to the use of power will do the work at as low an over-all cost to the user. The less efficient of two electric furnaces may show a great reduction in melting cost over the old fuel-fired furnaces, but with equal melting costs in the two motives of conservation should lead to the choice of the more efficient furnace.

For this reason special stress has been laid in this report on the difference in performance as to power consumption per ton of different types of furnaces, and with different methods of operation. It is true that as far as a cost sheet goes other advantages may sometimes make up the difference between a furnace melting at 300 kw. h. per ton, and another at 450 kw. h. per ton, but the balanced costs fail to indicate that the first uses but two-thirds as much as the second from the Nation's coal pile. In reference to desirability of economical utilization of coal, see Gilbert and Pogue.⁵⁹

WATER POWER.

Generally speaking, the brass industry in this country is at such a distance from developed water power or from that likely to be developed in the near future, that the general utilization of hydro-

⁵⁷ Electric furnace rates, *Electrical World*, vol. 75, 1920, p. 767.

⁵⁸ Melting changes bring lower costs, *Foundry*, vol. 48, 1920, p. 253.

⁵⁹ Gilbert, C. G., and Pogue, J. E., Coal, the resource and its full utilization: Smithsonian Institution, U. S. Natl. Museum, Bull. 102, part 4, 1918.

electric power for brass melting will have to be accomplished by the location of new plants within economical transmission distance of existing or future water-power development, or by the tying-in of water power and steam plants into a large network or such "super-power system" as has been advocated by former Secretary Lane.⁶⁰

Murray calculates⁶¹ that though the investment required by the proposed superpower network would be \$1,250,000,000, including the cost of changing from steam to electricity in railroads and industrial plants, it would return 24 per cent on the investment. It seems inevitable that this great improvement will sooner or later be made.

Another power project that may develop in the future is that on the St. Lawrence River, the main idea being to deepen the river to allow passage of seagoing vessels. Hoover⁶² cites this project as being "entirely feasible," capable of relieving the railroads from their peak load, and estimated to save 5 to 6 cents per bushel on the transportation of grain. In such a development a large amount of hydroelectric power would be made available, which would be free from the seasonal variations of some hydroelectric projects on account of the natural storage of the great lakes. Vogelsang⁶³ estimates that the power available as the United States share of that developed would be equivalent to 9,750,000 tons of coal per year and would require nearly 8,000 fewer workmen in the hydroelectric plants than would be required to develop the same power by steam.

With such huge projects as these in the East adjudged feasible and desirable by eminent engineers and economists and the advance of the already large hydraulic developments in the West, adequate electric power supply seems bound to come in the future.

Until hydroelectric power is thus made available electric brass furnaces will continue to utilize steam-generated power in the great majority of cases in this country.

Some other countries, though they do not melt much brass compared to the United States, will probably be able to apply hydroelectric power to brass melting when they start to utilize the electric brass furnace.

Sweden and Norway⁶⁴ have long been noted for their hydroelectric developments, which were forced upon them by the scarcity

⁶⁰Anonymous, Secy. Lane's proposal for power resource survey: *Elect. World*, vol. 73, 1919, pp. 282, 331; see also Report of Committee of Am. R. R. Assn., *Mining and Metallurgy*, No. 161, May, 1920, p. 7; Murray, W. S., *Superpower Survey report*: U. S. Geol. Survey Prof. Paper 123, 1921, 261 pp.

⁶¹Murray, W. S., *Economical power*: *Jour. Am. Inst. Elect. Eng.*, vol. 39, 1920, p. 27; *Economical supply of electric power for the railroads and industries of the North Atlantic seaboard*: *Jour. Am. Inst. Elect. Eng.*, vol. 39, 1920, p. 219.

⁶²Hoover, H., *Some notes on agricultural readjustment and the high cost of living*, *Sat. Eve. Post*, vol. 192, 1920, Apr. 10, p. 46. See also Craig, C. P., *From the Great Lakes to the Atlantic*, *Sat. Eve. Post*, vol. 192, 1920, June 26, p. 40.

⁶³Vogelsang, A. T., *St. Lawrence power*: *Power*, vol. 51, 1920, p. 397.

⁶⁴Compare Beckman, J. W., *Power developments in Scandinavia*, *Trans. Am. Electrochem. Soc.*, vol. 37, 1920, p. 403.

of fuel, and for their application of electric furnaces to the iron industry.

Italy has a highly developed water-power system, and there is to-day⁶⁵ a larger production of electric steel in Italy than of Bessemer steel.

In 1920 France⁶⁶ was expected to utilize about 1,600,000 h. p., or twice what she utilized in 1913. According to Keller,⁶⁷ the various plants of Keller Leleux and the Government plants designed by Keller were using electric furnaces on war production of pig iron, steel, carbide, etc., requiring about 30,000 kw., most of which was from water power.

The situation as to electric power in England has not been good, there being practically no water power, and according to Rideal⁶⁸ the average of some 600 power stations being only 5,000 h. p., or, as he states, one-quarter the size of an economical generating unit and one-thirtieth of the size of a really economical central station. Centralization into larger units is contemplated.

The fuller utilization of our own water-power resources has long been recognized as desirable and in the near future necessary.⁶⁹

The long-delayed constructive legislation which will make this a certainty for the future is now in operation, although on account of the large capital investment and the time required to construct hydroelectric plants some years will elapse before the full benefits can be realized. Comparatively prompt increase in utilization of water power is nevertheless assured.

OPERATION OF ELECTRIC BRASS FURNACES.

When cheap hydroelectric power becomes generally available for use with electric brass furnaces in this country in the future the economical use of power may become subordinate to other points in furnace operation, but as long as electric power is relatively costly the selection and operation of a furnace must take into account the fact that the power bill is one of the largest items of melting cost. How to operate with a low-power consumption, how to increase production, keep upkeep and repair cost down, and how to hold the metal losses low and the quality of the product high are the main questions confronting the electric furnace operator.

⁶⁵ V. D., 1000 fours électriques a acier dans le monde, Jour. du four électrique, vol. 28, 1919, p. 114.

⁶⁶ Toublett, Lt. Comdr., French postwar mineral resources, Mining and Metallurgy, Bull. 157, Jan. 1920, Sect. 1, p. 21.

⁶⁷ Keller, C. H., Synthetic electric furnace cast iron, Trans. Am. Electrochem. Soc., vol. 37, 1920, advance copy.

⁶⁸ Rideal, E. K., Industrial electrometallurgy, 1919, p. 15.

⁶⁹ Compare Dunn, G., The water power situation, including its financial aspect: Trans. Am. Inst. Elect. Eng., 1916, p. 575; Gilbert, C. G., and Pogue, J., Power, its significance and needs: Smithsonian Institution, U. S. Natl. Museum, Bull. 102, part 5, 1918.

To keep the metal loss low and the quality of the product high, the fundamental principle is to choose a furnace that can be run tightly closed, to keep air out and volatile metals in, and then to take care that it is really so run. When the charge contains oily borings it is impossible to operate any furnace tightly closed till the oil is driven out. But the oil vapor prevents the entrance of air as long as it is coming off, so if the furnace is plugged up as soon as the oil is off, little metal vapor is lost, as the bulk of the oil is out before the metal is melted and hot enough to vaporize much of the volatile constituents of the alloy.

Low upkeep and repair cost for refractories depends on the care and attention given the furnace, especially in the matter of overheating the lining or roof. Overheating is mainly due to the use of too high a rate of power input. But the higher the rate of power input the greater the output of product and the lower the energy cost per ton. Careful balancing between these two factors has to be made.

The rate at which the furnace can safely take and the transformers safely supply energy should be properly balanced so that there is enough transformer capacity to allow the furnace to do its best work, but not so much excess capacity as to involve a useless expense for the purchase of too large a transformer. On arc furnaces a balance has to be struck between a power factor that is high enough to make the arc so unsteady as to require too much attention or to make it too easy for the operator to allow the furnace to take too much power, and one so low as to involve waste by requiring larger and more expensive leads or a power factor penalty on the power bill. Automatic electrode control on arc furnaces and pyrometric records of roof temperature in the resistance type aid in allowing the operator to hold the maximum safe rate of input. Automatic electrode control is seldom justified in the installation of a single brass furnace, as it does not often cut the labor cost appreciably. But on a battery of two or more arc-type furnaces automatic electrode control, where it is possible to apply it, will usually pay for the investment very promptly by saving labor.

In order to allow holding the power input up to the maximum rate, the furnace operator must have all the meters to guide him that the particular type of furnace in use requires. Graphic meters and the keeping by the operator of a regular furnace log, showing time per heat spent in charging, melting, and pouring, in waiting on account of outside delays, and delays due to the furnace itself, kilowatt-hours used, weight charged and obtained, electrode consumption, etc., are most valuable in allowing the management to see to it that the furnace is operating properly.

To get a high production with the minimum power per ton one should use a furnace of the largest size his demand for metal will

allow, or that he can take the metal away from promptly, as the larger the capacity of a given type of furnace the more efficient it is. But if a large furnace has to be poured through a number of small ladles, the ratio of idle time to operating time may be so great that a smaller and more quickly emptied furnace may give better results.

On account of the high first cost of the electric furnace its overhead per idle hour is high. Moreover, the furnace while idle is leaking heat from its walls that has to be replaced during the next heat instead of going to the melting of metal. This leakage occurs not only between heats during the day, but at night also, so that there is usually a great difference in the efficiency of a furnace on 9-hour and on 24-hour operation.

The more nearly the furnace operates continuously the more cheaply it will operate. On continuous operation there is far less difference between the power consumption per ton of the more and the less efficient types than there is on 9-hour operation. Two-shift, 16-hour operation shows a greater improvement over one-shift operation than 24-hour over 16-hour operation. If melting costs are a main factor, and the quality of the castings is good with night work, as in rolling mills and smelting and refining plants, the tendency should be toward two-shift operation. In a shop making intricate cored castings in sand, unless artificial illumination can be provided that will keep the defective castings from night work down to the daylight figure, single-day-shift operation will prevail.

In single-shift operation, delays in pouring, waiting for the ladle crane, or holding the metal in the furnace because molds are not ready, operate to the great detriment of high furnace production and low power cost. Of course, melting is only one of the factors in foundry costs, and it may be better in the long run to operate on the basis of molding as the main factor, holding the metal in the furnace to wait for molds if necessary, and fitting the rate of melting to the rate of molding, but it should be clearly recognized that this handicaps a furnace greatly. Even for such operation the electric furnace has advantages; in the types without too great heat storage so that metal held in the furnace does not continue to heat up too rapidly from the heat stored in walls and roof, it is possible to hold the metal longer without damage to its quality than in fuel-fired furnaces.

If metal is to be held in furnaces of high-heat storage after it is ready to pour, additions of cold metal may have to be made to keep the charge from getting too hot before it is poured.

The average foundry using electric furnaces does not appear to realize how a few minutes delay waiting for the next charge, a few minutes more waiting for crane or ladle, and a few more holding the metal waiting for molds may add up in the course of a day to time enough to make another heat. Any furnace will illustrate this fact.

Take an actual day's run of a small Rennerfelt on red brass for example:

TABLE 55.—Operation of a furnace with and without delays.

Heat.	Charge.	Total time.	Arc on.	Idle time.	Kilowatt hours used.	Kilowatt hours per 100 pounds.	Notes.
	<i>Pounds.</i>	<i>Hrs. Min.</i>	<i>Hrs. Min.</i>	<i>Hrs. Min.</i>			
1.....	497	2 45	2 44	40	211	42	Furnace at red heat from previous day's run.
2.....	522	1 45	1 25	20	130	25	
3.....	527	2 55	1 10	1 45	105	20	Delay waiting for molds.
4.....	523	1 25	1 05	20	106	20	
5.....	394	1 25	1 10	15	94	25	Waiting for molds, so held power input low.
Total Average	2,463	10 15	6 55	3 20	646	26	520 kilowatt hours per ton.

This could have been run as follows by eliminating all delays:

Hours.	Charge.	Total time.	Arc on.	Idle time charging and pouring.	Kw. h.	Kw. h. per 100 pounds.
	<i>Pounds.</i>	<i>Hrs. Min.</i>	<i>Hrs. Min.</i>	<i>Hrs. Min.</i>		
1.....	525	2 10	1 55	15	215	41
2.....	525	1 40	1 25	15	130	25
3.....	525	1 25	1 10	15	105	20
4.....	525	1 20	1 05	15	100	19
5.....	525	1 20	1 05	15	100	19
6.....	525	1 20	1 05	15	100	19
Total.....	3,150	9 15	7 45	1 30	750	a 24

a Average, or 480 per ton.

The elimination of delays thus gives a 25 per cent increase in output in an hour's less time and decreases the power consumption 40 kw. h. per ton.

The difference between the results of a test run, made to show the maximum performance of the furnace, with everything arranged for rapid operation and those in regular operation under "foundry handicaps" is often very marked, and the maker's guarantees as to furnace performance under proper operation may be far better than the figures on everyday performance if the furnace is not properly operated.

Test runs should always be made and the best possible performance of the furnace determined. If the foundry does not closely approach this performance in regular operation, the manager can figure out how much his failure to provide the furnace with the conditions under which it can best operate is costing him. There is always a point at which rigid putting of the furnace performance first and requiring the other foundry operations to be arranged mainly with respect to efficient melting is more expensive in the long

run, and it is not always best to try to get 100 per cent of the possible performance from the furnace. But the extra saving possible by giving the furnace proper operating conditions and getting high production per furnace is generally large enough to demand consideration, because of high interest charges per ton of product when the expensive electric furnace is not allowed to give maximum production, and of the lowered power cost per ton when production is high.

It is, then, of greater relative importance among all the different foundry operations that an electric furnace be operated under conditions as favorable as possible to it than in the case of fuel-fired furnaces.

VALUE OF MECHANICAL CHARGING.

It saves much time to be able to dump the charge in mechanically from above instead of having to throw or shovel it in through a door in the side or end of the furnace. Such furnaces as the Snyder, the Detroit, and the Ajax that can be charged from the top, if installed with mechanical charging in view, or located so that no craneway or other structure over the furnace renders mechanical charging out of the question offer opportunity for time saving. Other types of furnaces which must be charged by hand through a door should have doors of generous size. If hand-charging must be used, chips, unless in relatively small amount, when they may be useful for charging first to form a cushion to prevent damage to the refractories when charging heavy ingots, may be briquetted,⁷⁰ and bulky material, like wire, should be compressed or "cabbaged."

Generally speaking, the furnace should be at a red heat before the first charge of the day is put in, this being obtained by preheating the empty furnace if it has cooled too far since last used.

The best results are obtained with a tightly closed furnace, and it is usually better to put the whole charge in at once instead of melting part, then opening the furnace to add the rest. If a large amount of cold metal is added to a relatively small mass of molten metal, the solid material may sink in the molten mass and freeze the whole into one solid chunk, which is harder to melt than a piled-up mass of smaller pieces.

Most furnaces should not be opened between charging and pouring, though if the charge contains too much nonmetallic material that may form a blanket of dross or a mirror-like slag, it is sometimes better to open the furnace and skim the slag or dross as soon as the charge is molten, as such a blanket or cover may retard the heating of the metal. Provision of sufficient heated ladles to allow

⁷⁰ Compare Stillman, A. L., Briquetting of nonferrous light metal scrap. Jour. Am. Inst. Met., vol. 11, 1917, p. 193.

emptying the furnace rapidly and the arrangement of ladle hoists and trolleys or craneways so that one ladle can be poured immediately after another, are important factors in time saving.

There are various details in which the operation of special types of furnaces require special procedure. In the Snyder the bottom electrode must be kept in metallic contact with the charge. In the rocking and revolving types care must be taken not to rock too far or to revolve too soon to prevent electrode breakage. In the Baily the furnace must be heated empty at night if run but one shift per day. In the Ajax the charge must be kept poked down and a layer of charcoal kept on the metal to prevent access to air. All furnaces are alike, however, in that the more steadily they are kept at melting and the less time they are allowed to be idle, the better their performance. On all, also, it is desirable that the temperature be controlled as closely as possible that the metal be not heated hotter than it is necessary to pour the casting desired properly. Needless superheat simply wastes time and energy.

PREHEATING THE CHARGE BY FUEL.

It is often suggested⁷¹ that brass melting should be done in two stages, first, heating the metal to a red heat by fuel and then transferring the hot metal to the electric furnace for melting.

Theoretically, this has many advantages. It is well known that the lower the temperature to which material is to be heated the more expensive it is to generate that heat electrically in competition with fuel.⁷² That is, the thermal efficiency of fuel-fired furnaces is the higher the lower the temperature to which their contents are heated. Or it would take fewer B. t. u. to heat brass to, say, 750° C. (1,400° F.) by direct fuel firing than by electric heating, not because the electric furnace is less efficient at the lower temperatures but because the fuel-fired furnace is more so.

If, then, we could preheat the metal by fuel and could charge the hot metal into the electric furnace we would do only that part of the work in the electric furnace which the electric furnace can do most efficiently. We would increase the output of the electric furnace and thus decrease the cost per ton for interest on the investment in the costly electric furnace. Volatilization of zinc and lead from brass is very small indeed until the alloy is melted and superheated; therefore, the metal loss would not be increased. If the

⁷¹ Compare Clamer, G. H., and Hering, C., The electric furnace for brass melting, *Trans. Am. Inst. Metals*, vol. 6, 1912, p. 97. Hansen, C. A., Electric melting of copper and brass, *Trans. Am. Inst. Metals*, vol. 6, 1912, p. 127. Hering, C., Ideals and limitations in the melting of nonferrous metals, *Jour. Inst. Metals (British)*, vol. 17, 1917, p. 250.

⁷² Compare Johnson, W. McA., and Sieger, G. N., Electric furnaces, their design, characteristics and commercial application, *Metal. and Chem. Eng.*, vol. 11, 1918, p. 506.

scheme could be applied to oily borings, the oil would be driven off in the preheating and would not interfere with the operation of the electric furnace.

Only such electric furnaces as are capable of mechanical overheating would be suitable for such a process. The success of the scheme would depend on how well the mechanical details were worked out and on whether or not harmful oxidation could be prevented in the transfer of the material from the preheating furnace to the electric furnace.

The synchronizing of another operation to the regular sequence of melting and the extra equipment and attention required would be a drawback, but there are many cases where the scheme would be worth a trial.

In melting cathode copper, if the cathodes can be kept from taking up oxygen or sulphur during the preheat, so that the electric furnace received material which on melting will have the proper composition to give a high-conductivity product, the scheme ought to help the electric furnace in its difficult task of competition with the present huge reverberatory furnaces.

This scheme of preheating the charge is advocated for steel melting by the Industrial Furnace Co., makers of the Snyder furnace, in which the whole roof is removed for charging so that a hot charge could be dumped in readily. No details on the operation of the plant in practice are yet available, but apparatus for such operation is being installed at the Standard Steel Castings Co., Cleveland. A considerable decrease in power consumption and electrode consumption per ton and a corresponding increase in output is expected, test run on 2½-ton charges of steel that required 600 kw. h. per ton with metal cold, taking but 360 kw. h. per ton when the metal is preheated to a cherry red.

If the same ratio holds in brass melting the power consumption of an arc furnace which melts red brass at 325 kw. h. per ton without preheating and produces, say, 6 tons in 9 hours, would melt at less than 200 kw. h. per ton and would produce about 10 tons if all the charge were preheated.

An induction furnace, melting yellow at 200 kw. h. without preheating and producing some 7 tons in 24 hours, would melt at 120 kw. h. per ton and produce over 11 tons. Clark⁷³ has designed for the Bridgeport Brass Co. a method of preheating for use with the induction furnace, but so far⁷⁴ the ratio between fuel prices and power prices has been such that in the very efficient induction furnace the complications added to the furnace by preheating have outweighed

⁷³ Clark, W. R., U. S. Patent 1,328,713, Jan. 20, 1920.

⁷⁴ Webster, W. R., Personal communication, Jan. 31, 1920.

the possible saving in cost of power plus fuel over that of power alone, so that the scheme has not yet had experimental test. The less efficient the furnace and the higher the cost of power, the greater the incentive to try out preheating will be.

FUTURE DEVELOPMENT OF ELECTRIC BRASS FURNACES.

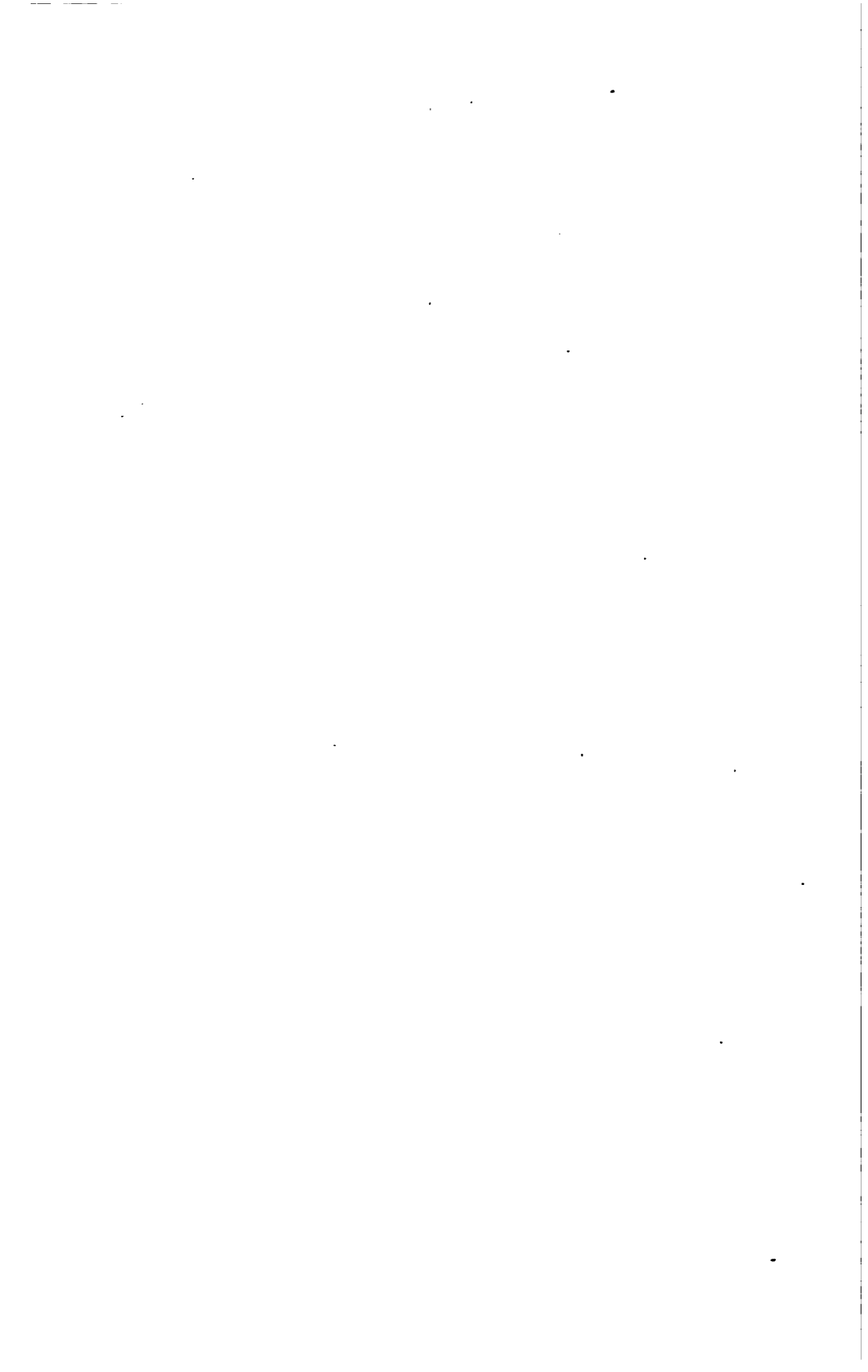
There are some points on which the path of progress of electric brass furnaces is fairly clear. Mechanical refinements, use of automatic electrode control in batteries of arc furnace, and a slow but general improvement in life of refractories may be looked for as experience is gained. There are, however, many points of interest on which there is not yet enough collected experience to allow prediction.

Whether the tendency in the use of existing types of electric brass furnaces will be toward the use of small furnaces, pouring 500 pounds per heat or less, or toward the 1-ton or larger furnaces, can not yet be decided. Practice will doubtless differ according to conditions, but just what conditions make one or the other alternative the better are not yet thoroughly clear.

Whether the types of furnaces that are so far doing the bulk of the nonferrous melting will continue to do so, or whether furnaces which so far have not had wide commercial use, or some of the others which are still in the experimental stage, or some brand-new entrant into the field will find large use has still to be seen. Much progress is still to be expected and the furnaces of to-day are doubtless crude compared to those of the future.

Nevertheless, enough progress has already been made so that there are electric furnaces suitable for and commercially successful in melting all sorts of nonferrous alloys. No one type of furnace is "best" for all sorts of nonferrous alloys, but the field is thoroughly covered if the different types are used on that portion of the field each is best fitted for.

The electric furnace has not yet displaced the fuel-fired furnace from the field, nor will it do so completely in the near future, if ever. But it does seem certain that in the great majority of cases the electric melting of brass is both the best and the cheapest method, and that electric furnaces, by their sheer superiority of service, must be ultimately destined for a dominant place in brass melting.



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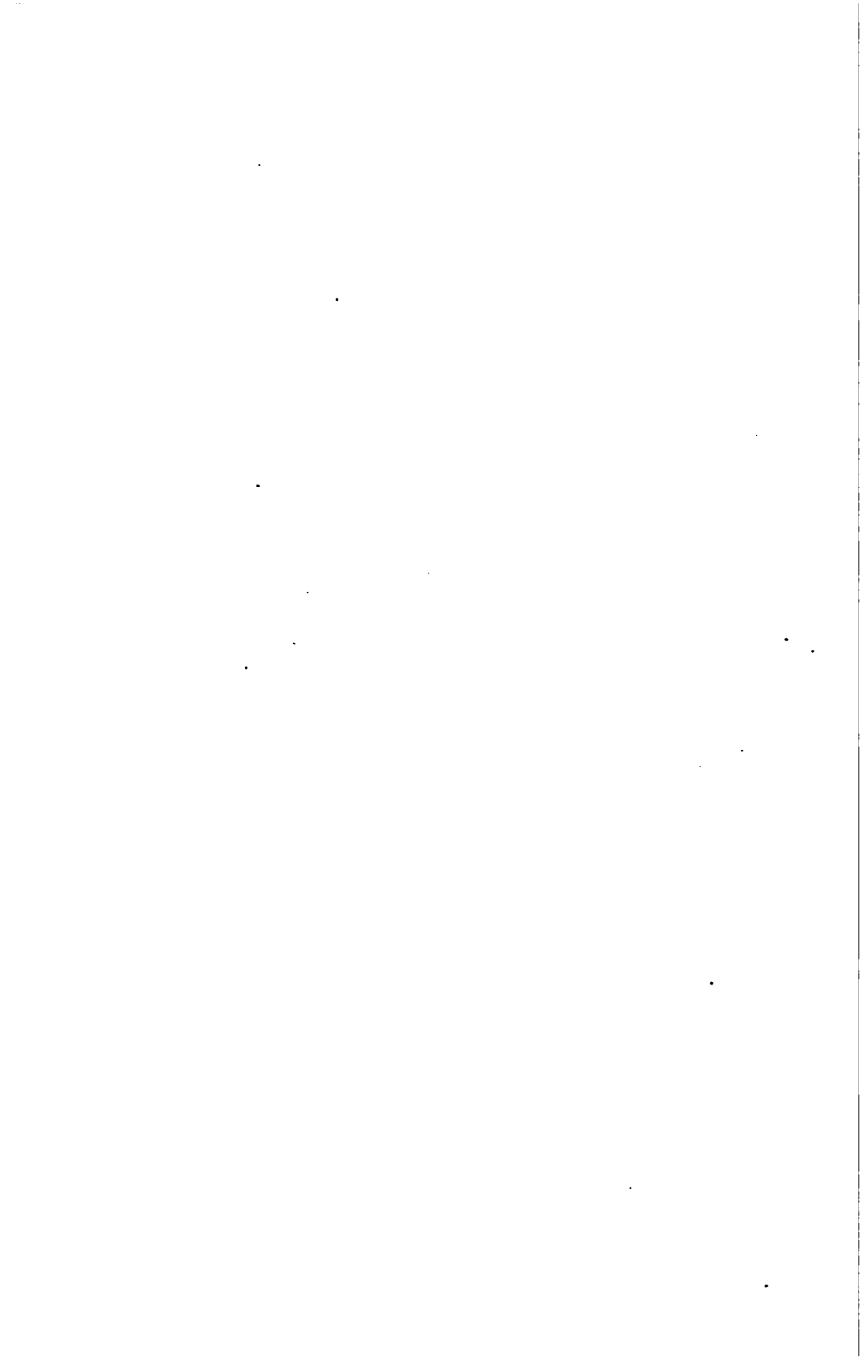
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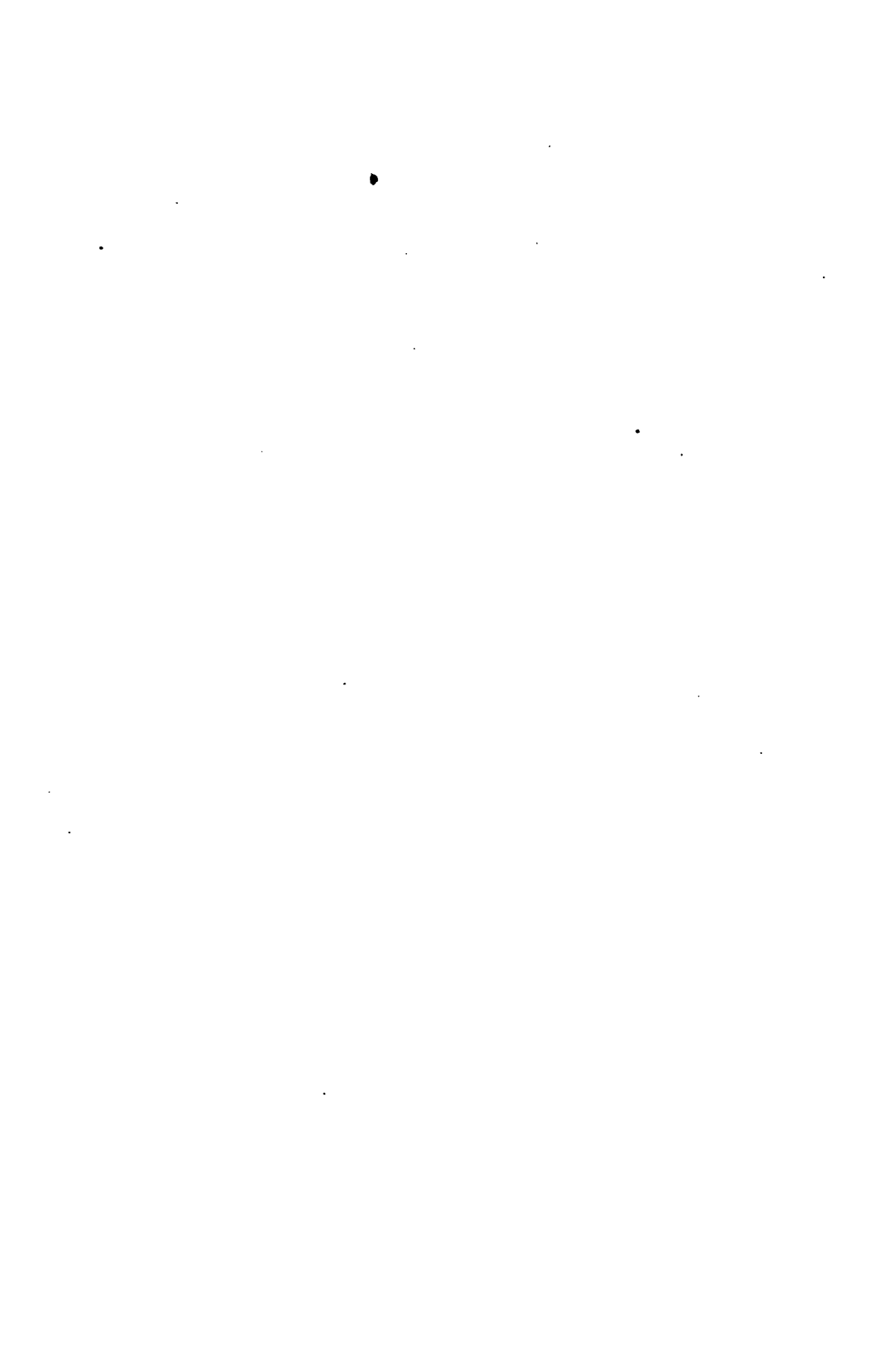
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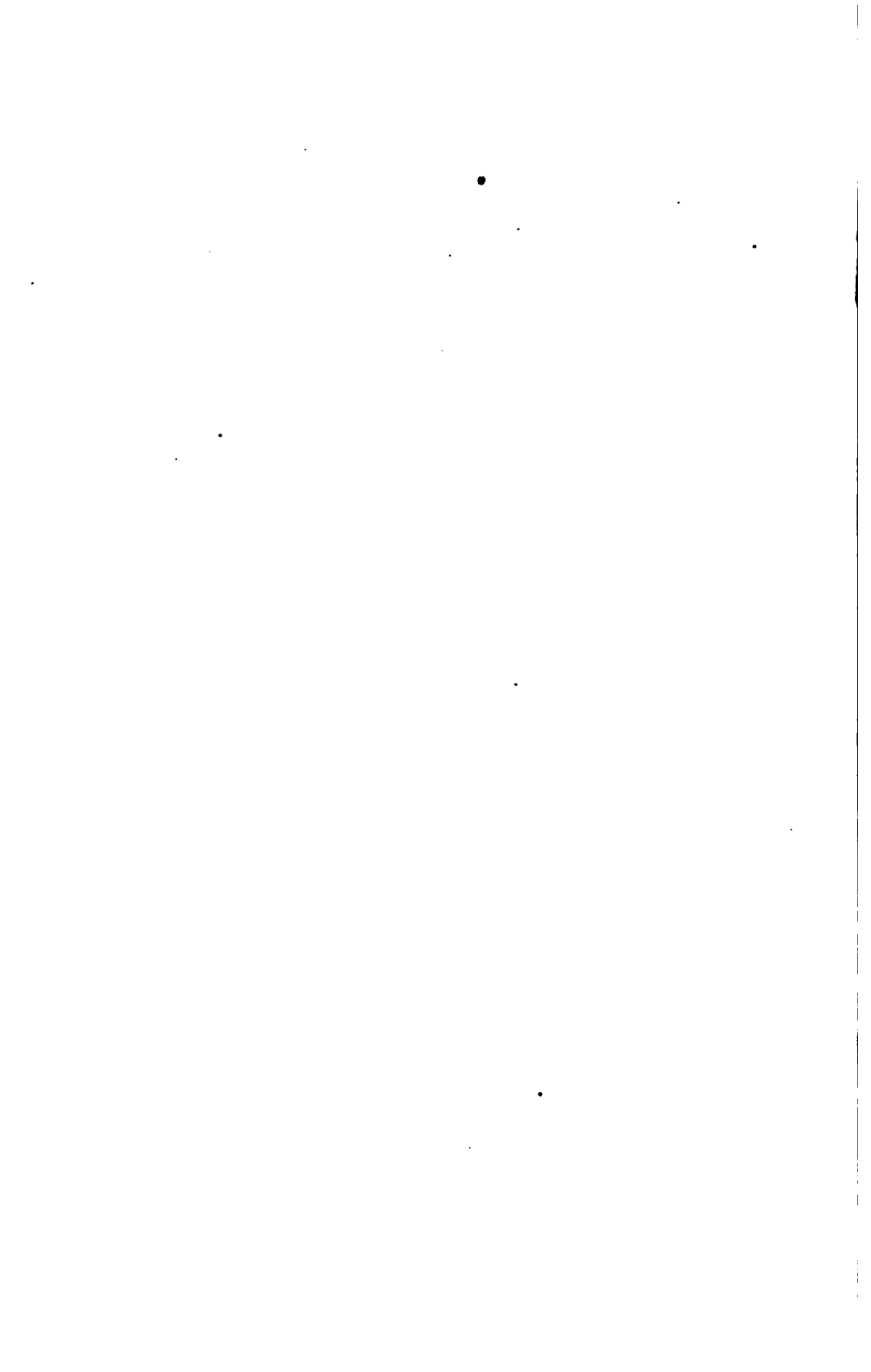
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DEPARTMENT OF THE INTERIOR

HUBERT WORK, SECRETARY

BUREAU OF MINES

H. FOSTER BAIN, DIRECTOR

**CENTRAL DISTRICT BITUMINOUS COALS
AS WATER-GAS GENERATOR FUEL**

BY

W. W. ODELL and W. A. DUNKLEY

**THIS BULLETIN REPRESENTS WORK DONE UNDER A COOPERATIVE AGREEMENT
BETWEEN THE BUREAU OF MINES, DEPARTMENT OF THE INTERIOR,
THE STATE GEOLOGICAL SURVEY DIVISION OF THE STATE
OF ILLINOIS, AND THE ENGINEERING EXPERIMENT
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CENTRAL DISTRICT BITUMINOUS COALS AS WATER-GAS GENERATOR FUEL.

By W. W. ODELL and W. A. DUNKLEY.

INTRODUCTION.

PURPOSE OF INVESTIGATION.

About two-thirds of the manufactured gas supplied to the public by the gas plants in the Illinois district is carbureted water gas. The leading generator fuel is coke, made in by-product coke ovens or in retorts, chiefly from eastern gas coals. During the fall and winter of 1917-18, when the demand for coke by war industries and the restrictions of the Fuel Administration in connection with the transportation of coke from eastern States made it almost impossible for many gas companies to obtain their coke requirements, several water-gas plants had to use bituminous coal as generator fuel. Many operating difficulties were encountered, some of which were solved more or less satisfactorily in different plants. A number of plants attacked the problem rather blindly because facilities for making tests were lacking and because technical men were too busy with labor and supply problems to study thoroughly the conditions involved or to ascertain what other plants had done with this fuel. It was therefore with the hope of assisting the gas industry during this critical period, and of promoting the use of Central District coals for gas making, that the Illinois Mining Investigations, a joint agency of the U. S. Bureau of Mines, the University of Illinois, and the Illinois Geological Survey Division undertook the tests discussed in this paper.

This report was compiled under the auspices of a technical committee appointed by Gov. Frank O. Lowden to promote the use of Central District coals in gas manufacture as a war measure. This committee included the following representatives of the various agencies concerned: U. S. Bureau of Mines, E. A. Holbrook and W. W. Odell; Illinois Gas Association, C. W. Bradley and R. B. Harper; University of Illinois, Dean C. R. Richards, Prof. S. W. Parr, and Prof. H. H. Stoek; State Geological Survey Division, F. W. De Wolf, G. H. Cady, and W. A. Dunkley. F. W. De Wolf was chairman of the committee and E. A. Holbrook was secretary.

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SCOPE OF INVESTIGATION.

The field of investigation was naturally divided into the following parts: (1) Study of the practice already developed and the difficulties encountered; (2) development, if possible, of operating methods that would obviate these difficulties; (3) testing various coals of the Central District to determine which might be considered for this purpose; and (4) study of some of the technical conditions involved in the manufacture of water gas from bituminous coals.

The first part of the investigation comprised collecting and summarizing the operating data by correspondence and by visits to several plants. The information obtained, which was published by the Illinois Geological Survey Division in Bulletin 22 of the cooperative mining series, formed a basis for the second part of the investigation, the development of practical methods to obviate the difficulties that had been experienced. It was decided that this work could not be done successfully except on a commercial scale. The Public Service Co. of Northern Illinois offered the facilities of its Streator gas plant for these experiments. The work at Streator extended over a period of several months. A preliminary report, dealing chiefly with the operating methods worked out at Streator, has been published as Bulletin 24 of the cooperative mining series by the Illinois Geological Survey Division. The purpose of the present paper is to discuss, in some detail, the principles involved in water-gas manufacture as they apply to the use of bituminous generator fuel; and to discuss in more detail the results obtained in the Streator tests and the application in other plants of the operating methods developed.

PREVIOUS WORK.

Study of the literature of gas manufacture brings to light few, if any, records indicating the successful use of bituminous coal in water-gas manufacture, but conversations with gas manufacturers justify a belief that considerable unreported work has been done in individual plants from time to time. The few experiments that had been made probably convinced operators that numerous difficulties were to be expected when bituminous coal was used, and as long as coke and anthracite were available at fairly reasonable prices there was little incentive for them to experiment with bituminous coal. However, as the war developed increasingly critical conditions in the fuel industry, several operators turned their attention to the possibilities of using bituminous coal in water-gas plants, and when the work reported in this paper was undertaken, several such plants had been operating two or more months with coal fuel.

LIMITATIONS AND ADVANTAGES OF BITUMINOUS GENERATOR FUEL.

There are limitations to the use of a high-volatile fuel in water-gas generators, inasmuch as water-gas sets are designed for low-volatile, high-carbon fuels. When coal is used *under the same operating conditions as for coke fuel*, the daily production capacity is considerably reduced; hence the use of bituminous coal for this purpose has been limited to such plants as did not need to be operated at full capacity. The difference in the cost of coal and coke has been very great in some parts of the country, and gas manufacturers in these regions, with plants not operating at full capacity, have found it decidedly advantageous to use coal fuel.

GENERAL PRINCIPLES OF WATER-GAS MANUFACTURE.

The manufacture of carbureted water gas is essentially a chemical process based upon chemical reactions, some of them simple and easily understood, others complex and not so easy to understand. The process may be considered logically in two parts: First, the formation of the uncarbureted water gas, so-called blue gas; and, second, the enrichment or carbureting of this gas to the quality desired for illuminating purposes.

BLUE GAS.

Uncarbureted water gas (blue gas) consists chiefly of hydrogen (H_2), carbon monoxide (CO), and carbon dioxide (CO_2) when high-carbon, low-volatile fuels, such as coke or anthracite coal, are used. It is formed by passing steam (H_2O) through an incandescent bed of fuel. The amount and composition of gas formed per unit of time depend upon the nature of the fuel, the rate at which the steam passes through the fuel, and the temperature of the fuel at the time the process is taking place. The chemical reactions occurring under varying conditions have been studied by a number of investigators, and the results of these studies will be summarized here to assist those readers that may not have these principles clearly in mind.

The following chemical equations show the principal reactions taking place in a water-gas generator during the steaming period or "run." After each equation, the heating value per cubic foot (corrected to standard conditions) of the total gas formed in the reaction is given:

| | B. t. u. per cubic foot. |
|------------------------------------|--------------------------|
| 1. $C + 2H_2O = CO_2 + 2H_2$ | 217 |
| 2. $C + H_2O = CO + H_2$ | 325 |
| 3. $CO_2 + C = 2CO$ | 324 |

Two of the three gases produced by these reactions— CO and H_2 —are combustible, and their heating values per cubic foot are nearly alike. The other gas, CO_2 , is not combustible and its presence

in water gas is undesirable. The quantity of CO_2 present in water gas can be controlled when the generator reactions and their practical application are well understood.

Consideration of the heat of combustion of the gas produced by each of these reactions indicates that equation 1 is an undesirable reaction. Equation 3, however, represents a secondary reaction that takes place when the CO_2 formed in reaction 1 has an opportunity to react further on hot carbon, the time of contact, the concentration of the reacting gases, and the temperature of the fuel bed being the factors that control the extent to which this reaction takes place. When all the CO_2 formed according to equation 1 is converted into CO as in equation 3, the resulting product (considering equations 1 and 3 together) is a mixture of equal volumes of H_2 and CO , or the equivalent of reaction 2. It is desirable in the practical operation of water-gas generators to produce the conditions most favorable for reaction 2 or its equivalent, and at the same time the least favorable conditions for reaction 1. The favorable conditions are: Intimate contact of the steam with the incandescent fuel, a high temperature in the fuel bed, and a desirable time of contact corresponding to this temperature. The reverse of these conditions: Imperfect contact of short duration and low temperatures in the fuel bed favor reaction 1.

All of these reactions may be, and usually are, occurring simultaneously in different parts of the fuel bed in practical operation. They have been more or less thoroughly studied under controlled conditions in the chemical laboratory.

TABLE 1.—*Experiments with water vapor and charcoal.*^a

| Temperature. | | Gas velocity. | $\frac{1}{\text{vol.}}$ | Constituents. | | | | Composition of dry gas. | | |
|--------------|-------|-------------------------|-------------------------|------------------|------------------|------------------|------------------------|-------------------------|------------------|------------------|
| °C. | °F. | | | CO_2 . | CO . | H_2 . | H_2O . | CO_2 . | CO . | H_2 . |
| | | <i>Cu. ft. per sec.</i> | | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| 674 | 1,248 | 0.0818 | 31.44 | 3.84 | 0.63 | 8.41 | 87.12 | 20.8 | 4.9 | 65.3 |
| 758 | 1,396 | .0636 | 15.72 | 9.23 | 2.67 | 22.28 | 65.82 | 27.0 | 7.8 | 65.2 |
| 838 | 1,540 | .1168 | 8.84 | 12.11 | 7.96 | 32.77 | 47.16 | 22.9 | 15.1 | 62.0 |
| 861 | 1,582 | .1870 | 5.35 | 13.33 | 11.01 | 36.48 | 39.18 | 21.9 | 18.1 | 60.0 |
| 964 | 1,749 | .2224 | 4.50 | 5.66 | 32.70 | 44.43 | 17.21 | 6.8 | 30.5 | 53.7 |
| 1,010 | 1,850 | .2171 | 4.60 | 1.45 | 48.20 | 47.30 | 3.02 | 1.5 | 49.7 | 48.8 |
| 1,080 | 1,946 | .2460 | 2.89 | 1.25 | 46.31 | 48.84 | 3.66 | 1.3 | 48.1 | 50.6 |
| 1,125 | 2,057 | .3090 | 2.51 | .60 | 48.34 | 50.73 | .303 | .6 | 48.5 | 50.9 |

^a Columns 2, 4, 9, 10 and 11 are computed by the authors.

Table 1¹ by Harries shows the effect of the temperature on the decomposition of water vapor in the presence of carbon (charcoal). Water vapor was passed over heated charcoal and the composition of the resulting gas was determined. The experiment was on a laboratory scale. It will be noted that as the temperature is increased more water vapor is decomposed. Simultaneously there is

¹ Harries, [Decomposition of water vapor in the presence of carbon]: Jour. Gasbel., Jahrg. 37, 1894, p. 82.

an increasing percentage of CO and a decreasing percentage of CO₂, in the gas produced, in spite of the fact that the time of contact at the higher temperature was much shorter. Unfortunately Harries does not state the capacity of his apparatus, hence it is impossible to calculate the absolute time of contact; however, the relative contact time is indicated in column 4 as the reciprocal of the velocity.

The above table represents results obtained with charcoal as fuel. Other forms of fuel, such as coke and coal, decompose steam with different reaction velocities. Although charcoal decomposes water vapor more rapidly than does coke or some other fuels, the table is characteristic of the effects of temperature on the production of water gas and shows the importance of maintaining a high temperature in the fuel bed of a water-gas generator during the steaming period.

Table 2² shows the effect of the time of contact on the decomposition of steam and on the composition of the gases formed when the temperature remains constant. The results were obtained by passing steam over heated willow charcoal.

TABLE 2.—*Decomposition of steam by charcoal at 1,100° C.*

| Time of contact in seconds, <i>t</i> . | $\frac{1}{t}$. | Composition of gas (per cent). | | | | | | Composition of dry gas (per cent). | | | |
|--|-----------------|--------------------------------|------|------------------|-------------------|-------------------|--------------------|------------------------------------|-------|------------------|-------------------|
| | | CO ₂ . | CO. | H ₂ . | CH ₄ . | H ₂ O. | Total fixed gases. | CO ₂ . | CO. | H ₂ . | CH ₄ . |
| 6.92..... | 0.1444 | 0.0 | 50.5 | 47.3 | 1.3 | 0.9 | 99.1 | 0.0 | 50.95 | 47.75 | 1.31 |
| 5.62..... | .1778 | .1 | 50.1 | 48.1 | .8 | .9 | 99.1 | .1 | 50.60 | 48.50 | .80 |
| 3.37..... | .2968 | .3 | 43.3 | 43.4 | .7 | 12.3 | 87.7 | .34 | 49.40 | 49.46 | .80 |
| 1.73..... | .5630 | .4 | 39.6 | 39.0 | .2 | 20.8 | 79.2 | .50 | 50.00 | 49.25 | .25 |

It will be noted that as the time of contact of the steam with the heated carbon (charcoal in this case) decreases, the per cent of CO₂ increases and the per cent of CO decreases. In other words, as the time of contact decreases, the reaction of equation 1 ($C + 2H_2O = CO_2 + 2H_2$) becomes increasingly prominent. The amount of steam decomposed increases as the time of contact increases, the decomposition being almost complete with five seconds contact.

EFFECT OF TIME OF CONTACT AND TEMPERATURE.

The effect of the time of contact and the temperature on the relative percentages of CO₂ and CO is made more evident by a further consideration of the reactions affecting these two constituents. It has been pointed out that when CO₂ formed by the reaction of equation 1 has further opportunity to react with glowing carbon the reaction of equation 3 ($CO_2 + C = 2CO$) takes place if the time of contact and

² Clement, J. K., Adams, L. H., and Haskins, C. N., Essential factors in the formation of producer gas: Bull. 7, Bureau of Mines, 1911, p. 42.

temperature are favorable. Therefore the reactions of equations 1 and 3 together are equivalent to the reaction of equation 2 ($C + H_2O = CO + H_2$), as stated. Thus, when the gases from the water-gas generator are analyzed, it is impossible to say how much of the carbon monoxide present was formed by the reaction of equation 2 or by the combined reactions 1 and 3; however, the conditions affecting the final result are much the same as shown in Table 3.³

TABLE 3.—*Composition of a mixture of CO and CO₂ when in equilibrium with coke under a pressure of one atmosphere.*

| Temperature. | | Per cent
CO. | Per cent
CO ₂ . |
|--------------|-------|-----------------|-------------------------------|
| °C. | °F. | | |
| 900 | 1,652 | 83.2 | 16.8 |
| 1,000 | 1,832 | 94.5 | 5.5 |
| 1,100 | 2,012 | 98.1 | 1.9 |
| 1,200 | 2,192 | 99.4 | .6 |
| 1,300 | 2,372 | 99.7 | .3 |

This table shows the effect of temperature on the percentages of CO₂ and CO which can exist in equilibrium with hot coke. It will be noted that as the temperature is increased, more of the CO₂ is reduced by the coke to CO, until, in the neighborhood of 1,300° C., practically all of the CO₂ has been reduced to CO. It should be borne in mind that the time of contact of the gases with the hot carbon has not been considered here. All the time needed to effect an equilibrium has been allowed. The percentages of CO are the highest possible at the given temperatures, regardless of the length of time allowed. Experiments have shown that at the lower temperatures the reduction of CO₂ to CO according to equation 3 ($CO_2 + C = 2CO$) proceeds very slowly. For example, when CO₂ was passed through coke heated to 900° C. only 13 per cent of CO was formed with 80 seconds contact, while 142 seconds contact were required to produce 27.6 per cent of CO. Table 4 shows the effects of the time of contact on this reaction at various temperatures.

Time is an important factor in all chemical reactions, and its relation to the reactions in a water-gas generator should be well understood. In Table 4 are shown the effects of changes in time of contact and in temperature.

TABLE 4.—*Rate of formation of CO from CO₂ and coke.*

AT A TEMPERATURE OF 900° C. (1,652° F.).

| Time of
contact,
seconds. | Per cent
of CO
observed. |
|---------------------------------|--------------------------------|
| 142.0 | 27.6 |
| 80.2 | 13.1 |
| 43.9 | 9.4 |
| 16.1 | 4.9 |
| 9.6 | 2.6 |
| 3.7 | .8 |

³Clement, J. K., Adams, L. H., and Haskins, C. N., work cited, p. 50.

AT A TEMPERATURE OF 1,000° C. (1,832° F.).

| | |
|-------------|-------|
| 123. 2..... | 78. 4 |
| 80. 25..... | 64. 4 |
| 33. 25..... | 52. 9 |
| 18. 72..... | 32. 0 |
| 6. 37..... | 13. 9 |
| 4. 1..... | 11. 5 |
| 3. 07..... | 9. 2 |
| 1. 98..... | 6. 3 |

AT A TEMPERATURE OF 1,100° C. (2,012° F.).

| | |
|-------------|-------|
| 90. 0..... | 97. 1 |
| 29. 92..... | 85. 4 |
| 13. 2..... | 66. 1 |
| 6. 765..... | 55. 6 |
| 3. 198..... | 31. 7 |
| 1. 784..... | 30. 4 |
| 1. 66..... | 24. 0 |
| 1. 59..... | 22. 1 |
| 1. 462..... | 21. 4 |
| 0. 962..... | 13. 3 |

AT A TEMPERATURE OF 1,200° C. (2,192° F.).

| | |
|-------------|-------|
| 18. 92..... | 98. 9 |
| 12. 7..... | 97. 8 |
| 8. 25..... | 95. 3 |
| 2. 402..... | 68. 5 |
| 1. 582..... | 43. 9 |
| 1. 08..... | 33. 5 |

AT A TEMPERATURE OF 1,300° C. (2,372° F.).

| | |
|-------------|-------|
| 8. 86..... | 99. 9 |
| 4. 149..... | 97. 9 |
| 2. 1..... | 93. 2 |
| 1. 13..... | 83. 4 |

Note.—The percentages of CO given in Table 2 do not represent the percentage of CO in the finished gas. Reaction 3 is considered by itself in this table and the CO shown is the amount produced when CO₂ is passed through incandescent coke with the time of contact and at the temperatures given.

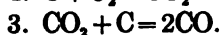
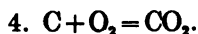
In the practical operation of a water-gas machine, obviously only a few seconds at most can be allowed for the reactions to take place in the generator. Since at the lower temperatures a prolonged time of contact is necessary and at higher temperatures the reactions take place almost instantaneously, the necessity for maintaining high temperatures in the generator is evident.

When reactions 1 and 3 are considered together and a high fuel temperature is assumed, the resulting products of the reactions are chiefly CO and H₂, as expressed by reaction 2. The amount of CO, present in water gas then depends on the time the gases are in contact with the glowing coke, as well as upon the temperature of the coke. In other words, at high temperatures and with a fuel bed of

sufficient depth, reaction 2 or its equivalent takes place, while at lower temperatures or with a short time of contact (exemplified by very low fuel bed and excessive ash accumulations in the generator) more of reaction 1 takes place simultaneously with reaction 2. During the steam run the temperature of the fuel in the generator drops considerably, the reaction velocity, or rate at which the steam and carbon combine to form gas, being decreased simultaneously. Expressed in other terms, more steam passes through the fuel undecomposed as the temperature in the generator decreases. This fact must be given due consideration when an operating cycle is worked out.

BLASTING FUEL WITH AIR.

After the steam run is made, it is necessary to bring the temperature in the generator up to the desired temperature before another run can be made. This is done, in present-day processes, by blasting the fuel with air. Air is a mixture of 20.9 per cent of oxygen (O_2) by volume and 79.1 per cent of nitrogen (N_2). Nitrogen, being an inert gas, does not enter into the reaction. The principal chemical reactions that take place during blasting are as follows:



In blasting fuel with air, the oxygen of the air combines with the C to form CO_2 , as expressed in reaction 4. All the oxygen of the blasted air combines with carbon (at the usual prevailing temperatures) before it has gone very far through the fuel bed; thus the zone of complete combustion is close to the grates and represents but a small percentage of the fuel bed. As the CO_2 that is first formed passes up through the incandescent fuel, it combines further with carbon to form CO, as in reaction 3 already mentioned. When the blast is started, directly after the steam run, the fuel is not hot enough for the completion of reaction 3 with the usual time of contact that prevails during blasting.

The curves of Figure 1, plotted from analyses made during actual operation, show the effect of changes in temperature of the fuel bed on the CO_2 content of the gases made during the blow and during the run. It is noted that during the blow the fuel temperature increases and the CO_2 content decreases, while during the run the fuel temperature decreases and the CO_2 content of the gas increases. CO is the chief combustible constituent of the blast gases that are burned in the carburetor and superheater by the admission of secondary air. It may be expected from the foregoing that a change from coke to a high-volatile fuel may upset the heat balance in a water-gas set by increasing the amount of combustible gas to be burned by the secondary air in the carburetor and superheater. It is not intended to rep-

resent these reactions as the only ones occurring in a water-gas generator. Other reactions take place to a limited degree in addition to those in the volatile matter that may be present in the original fuel, but they will not be mentioned here, since the prime factors have been given.

Each of the four reactions that have been discussed above is accompanied by a transfer of heat. When reaction 4 takes place it evolves heat—is exothermic. All the other reactions proceed with an absorption of heat and are therefore endothermic.

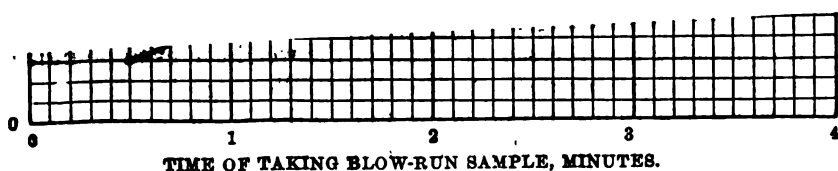
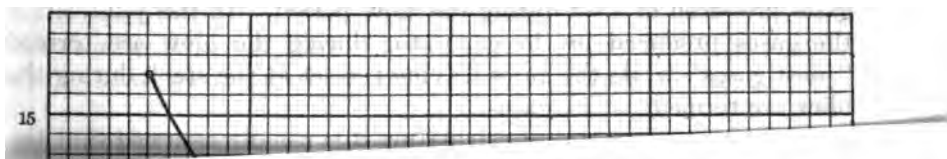


FIGURE 1.—Carbon dioxide (CO_2) in gases sampled at generator lid during blow-run period.

According to these reactions, a definite amount of carbon is consumed in the production of 1,000 cubic feet of blue gas. In practical operation, the amount of fuel required per 1,000 feet is considerably more than that indicated by the equations, since heat losses inevitably occur. These heat losses include—

1. Losses as sensible heat in the blue and blast gases which leave the generator at high temperature.
2. Losses due to unconsumed fuel and sensible heat in the ashes removed from the generator.
3. Losses due to the latent heat of vaporization of water in the fuel.
4. Losses due to radiation and convection from the generator.

sufficient depth, reaction 2 or its equivalent takes place, while at lower temperatures or with a short time of contact (exemplified by very low fuel bed and excessive ash accumulations in the generator) more of reaction 1 takes place simultaneously with reaction 2. During the steam run the temperature of the fuel in the generator drops considerably, the reaction velocity, or rate at which the steam and carbon combine to form gas, being decreased simultaneously. Expressed in other terms, more steam passes through the fuel undecomposed as the temperature in the generator decreases. This fact must be given due consideration when an operating cycle is worked out.

BLASTING FUEL WITH AIR.

After the steam run is made, it is necessary to bring the temperature ~~to the temperature before another run~~

ERRATA.

PAGE 9. Figure 1, bottom lettering, "Time of taking blow-run sample, minutes" should read "Time of taking blow and run samples, minutes."

Title to Figure 1, "Carbon dioxide (CO_2) in gases sampled at generator lid during blow-run period" should read "Carbon dioxide (CO_2) in gases sampled at generator lid during blow and run periods."

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up through — — —
form CO , as in reaction 3 already mentioned. When the blast is started, directly after the steam run, the fuel is not hot enough for the completion of reaction 3 with the usual time of contact that prevails during blasting.

The curves of Figure 1, plotted from analyses made during actual operation, show the effect of changes in temperature of the fuel bed on the CO_2 content of the gases made during the blow and during the run. It is noted that during the blow the fuel temperature increases and the CO_2 content decreases, while during the run the fuel temperature decreases and the CO_2 content of the gas increases. CO is the chief combustible constituent of the blast gases that are burned in the carburetor and superheater by the admission of secondary air. It may be expected from the foregoing that a change from coke to a high-volatile fuel may upset the heat balance in a water-gas set by increasing the amount of combustible gas to be burned by the secondary air in the carburetor and superheater. It is not intended to rep-

resent these reactions as the only ones occurring in a water-gas generator. Other reactions take place to a limited degree in addition to those in the volatile matter that may be present in the original fuel, but they will not be mentioned here, since the prime factors have been given.

Each of the four reactions that have been discussed above is accompanied by a transfer of heat. When reaction 4 takes place it evolves heat—is exothermic. All the other reactions proceed with an absorption of heat and are therefore endothermic.

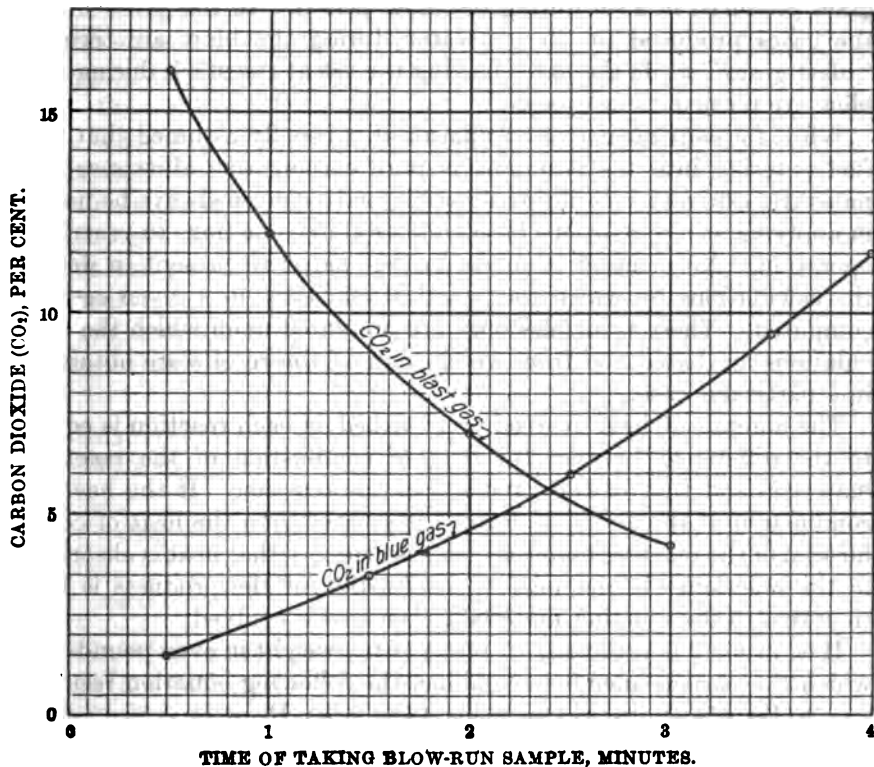


FIGURE 1.—Carbon dioxide (CO₂) in gases sampled at generator lid during blow-run period.

According to these reactions, a definite amount of carbon is consumed in the production of 1,000 cubic feet of blue gas. In practical operation, the amount of fuel required per 1,000 feet is considerably more than that indicated by the equations, since heat losses inevitably occur. These heat losses include—

1. Losses as sensible heat in the blue and blast gases which leave the generator at high temperature.
2. Losses due to unconsumed fuel and sensible heat in the ashes removed from the generator.
3. Losses due to the latent heat of vaporization of water in the fuel.
4. Losses due to radiation and convection from the generator.

5. Losses of heat during lay-over periods when no gas is being made.
6. Losses due to combustion of fuel during fire-cleaning periods.
7. Losses due to the use of excess steam during the steam run.

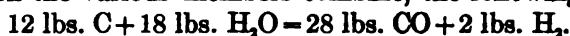
These losses must inevitably be different in various plants where the kind and size of sets, daily operating time, weather conditions, and fuels are different. A study of the thermal properties of these reactions serves to point out the maximum fuel efficiencies that can be realized in practical operation.

There is some difference in nomenclature in the designation of the gases produced in a set during the blow period. In this publication the gases produced in the generator during the blow are termed "blast gases" while the gases leaving the set at the stack during the blow are termed "stack gases."

While for convenience of calculation it is usually assumed that the fuel entering into the reactions is pure carbon, due allowance for moisture, ash, and volatile combustible matter must always be made in applying any conclusions based on the above reactions to practical operation. No attempt is made in this paper to present in detail the calculations by which the practical efficiency in a given case is computed. Those principles only are presented upon which the calculations are based and from which some of the results are obtained in a particular case.

The amount of heat evolved or absorbed in each reaction is equal to the difference between the heat of combustion of the reacting materials and that of the products of the reaction. If the heat of combustion of the products formed is greater than the heat of combustion of the reacting materials, this is evidence that heat is absorbed in the reaction. If the heat of combustion of the products is less than that of the reacting materials, then heat is evolved.

If equation 2 is expressed in terms of the weight in even pounds by which the various members combine, the following equation results:



From this amount of fuel 755 cubic feet of gas are produced, consisting of equal volumes of H_2 and CO.

The heat of combustion of carbon in the formation of CO is 4,350 B. t. u. per pound of carbon; that of H_2 in the formation of water is 61,523 B. t. u. per pound of H_2 . Knowing these values, the amount of heat to be supplied by the generator for reaction 2 can be calculated approximately as follows:

| | B. t. u. |
|--|----------|
| Heat of combustion of 2 lbs. of $\text{H}_2 = 2 \times 61,523$ | 123, 046 |
| Heat furnished by the formation of 28 lbs. of CO = $12 \times 4,350$ | 52, 200 |
| Heat absorbed..... | 70, 846 |
| Assuming that the water was steam at 100 lbs. pressure it has already received 18×1153.9 B. t. u..... | 20, 770 |
| The total heat absorbed in the formation of 755 cu. ft. of gas produced according to equation 2 is..... | 50, 076 |

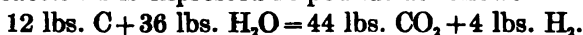
Approximately 100 per cent more steam is actually used in water-gas generation than is shown by the theoretical chemical reaction. This steam, in passing through the fuel bed unchanged, absorbs heat and must therefore be considered when arriving at an absolute heat balance. Likewise, there must be considered the sensible heat of the gases.

The total heat must be supplied by the combustion of fuel in the generator. The amount of heat necessary for the production of 1,000 feet of this gas is:

$$(1,000 \div 755) \times (50,076) = 66,300 \text{ B. t. u.}$$

It is estimated that reaction 1 ($C + 2H_2O = CO_2 + 2H_2$) takes place in practice only to the extent of about 5 per cent and reaction 2 ($C + H_2O = CO + H_2$) to the extent of approximately 95 per cent during the run. This is considered, of course, in calculating the fuel consumed during the run.

Reaction 1 is expressed in pounds as follows:

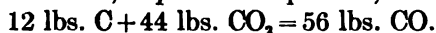


In this instance, the combination of 12 pounds of carbon with H_2O produces 1,133 cubic feet of gas, of which one-third by volume is CO_2 and two-thirds is H_2 . The heat balance of this equation, developed after the method employed for equation 2, is as follows:

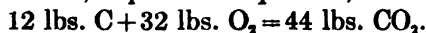
| | B. t. u. |
|--|----------|
| Heat furnished by combustion of 12 lbs. of carbon burning to CO_2 ($12 \times 14,544$) | 174, 528 |
| Heat furnished by the steam (36×1153.9) | 41, 540 |
| Total | 216, 068 |
| Heat of combustion of H_2 ($4 \times 61,528$) | 246, 092 |
| Total heat to be furnished by combustion of fuel in generator for this reaction | 30, 024 |

A heat balance can be similarly developed for the reactions expressed by equations 3 ($CO_2 + C = 2CO$) and 4 ($C + O_2 = CO_2$).

Reaction 3, expressed in pounds, is:



Reaction 4, expressed in pounds, is as follows:



This is the primary reaction taking place in the zone of complete combustion of the generator during the blast. The heat generated by this reaction is:

$$12 \times 14,544 = 174,528 \text{ B. t. u.}$$

Reaction 3 is a secondary reaction and the extent to which it takes place can be determined only by analyses of the gas. The net result of blasting the generator can only be determined when it is known to what extent reaction 3 takes place. Since reaction 4 is the heat-producing reaction, it is evident that the extent to which it must take place during the blow period to supply the require-

ments of the other reactions and the heat losses which inevitably occur in practical operation will depend upon a number of variables, including the extent to which each of the heat-absorbing reactions occurs. By assuming that each of these reactions occurs in definite amounts, that the fuel contains a definite amount of inert material, that the gas and blast products leave the generator at a given temperature, and that a certain amount of heat is lost by radiation and convection, and in other ways, it is possible to compute what the fuel consumption will be during each phase of operation.

The gases usually leave the generator at or near a temperature of 1,400° F. and when the net result of blasting the generator or the effect of making a steam run is to be estimated, the loss due to the sensible heat of the gases must be considered. Computations based upon certain assumptions and specified conditions have been made by a number of engineers. The coke consumed per 1,000 cubic feet of blue gas made, as computed by Norton H. Humphreys ⁴ from conditions assumed by him, may be summarized as follows:

| | Pounds. |
|--|---------|
| Coke used to form CO and CO ₂ during run..... | 17.2 |
| To supply heat carried off as sensible heat..... | 11.0 |
| Consumed during blow..... | 12.6 |
| Total..... | 40.8 |

In determining the heat losses, especially those due to convection, radiation, and the sensible heat in the gases leaving the generator, some assumptions that are based upon imperfectly worked out data must be made. For example, in computing the heat loss as the sensible heat of the gases, the heat-carrying capacity of these gases at high temperatures must be known. Although various values are given for the specific heats of gases at high temperatures, consideration of the available data leads the authors of this paper to believe that the specific heats taken by Mr. Humphreys may be a little high but are valuable to illustrate the problem in hand.

REACTIONS IN CARBURETOR AND SUPERHEATER.

When gas oil is sprayed into the carburetor on the hot checker brick, it is decomposed or cracked. The final products of decomposition depend on a number of varying conditions. Some of the more important are the following:

1. Kind of oil used.
2. Temperature of checker brick.
3. Spacing and condition of checker brick.
4. Quality and quantity of blue gas in the atmosphere where the oil is being cracked.
5. Quantity of oil cracked per unit of time.

⁴ Humphreys, N. H., *Thermal problems for gas engineers: Jour. Gas Lighting*, vol. 135, pp. 393-394.

6. Pressure.

7. Efficiency of the oil-spray nozzle.

When some of the conditions affecting the efficiency of oil cracking are changed, some undesirable results are occasionally noted. A study of all the results obtained under varying conditions has led to some interesting deductions and to a better understanding of the reactions occurring in the carburetor and superheater. The chemical and thermal reactions taking place when oil is cracked are more numerous and complicated than the simple reactions taking place during the production of blue gas in the generator. In addition to the reactions of the oil-decomposition products among themselves, there are, under some conditions, reactions between these products and the gases from the generator (blue gas).

When oil is cracked in the usual water-gas process, there are formed gas and varying amounts of carbon, tar, naphthalene, and condensable hydrocarbons. Since it is the object to produce the maximum amount of gas per gallon of oil—consistent, of course, with the quality of gas desired—with a minimum amount of carbon, naphthalene, and tar, the conditions governing the formation of each are of as much importance for the successful operation of a water-gas set as the factors affecting the various constituents of the gas. Some of the latter are discussed here in the order given above.

KIND OF OIL.

The kind of oil used has considerable bearing on the results obtained; hence the specifications on which gas oil is purchased are of interest. It is not uncommon to find oil purchased according to the specific gravity (the Baumé reading), or according to the viscosity and the Baumé test. Some companies fractionate the oil and collect fractions every 100° F. up to 700° F., noting the condition and quantity of the fractions and of the residue. This residue may be all carbon or coke and may represent a small per cent of the original sample, or it may be a heavy oil and represent either a large or a small per cent of the sample. Some specifications limit the amount of residue; others demand a certain uniformity of the quantity fractionating at the different temperatures.

Since the crude oils from different fields are not alike, their distillates are likewise dissimilar. Furthermore, a mixture of a heavy oil and a light oil will produce an oil with an intermediate specific gravity that could not be distinguished by a Baumé hydrometer from an oil of the same gravity with a constant boiling point.

Sherman and Kropp^a report that the heating value of an oil varies with its specific gravity. They state that the formula:

$$\text{B. t. u. per pound} = 18,650 + 40 (\text{B.} - 10)^{.66}$$

^a Journal of Gas Lighting and Water Supply, Meeting of Dutch Managers.—Report of committee on testing of gas oils: Vol. 116, 1911, p. 643.

⁶⁶ B. is the degree Baumé.

gave results within 1 per cent of the actual on more than 60 oils tested by them. Tests made since by others have shown a similar relation. Attempts have been made to make use of this in grading the various oils, but the B. t. u. efficiency

B. t. u. in the gas per gallon of oil used

B. t. u. in the oil per gallon

varies to such an extent that it is impossible to grade gas oils this way.

When consideration is given to the characteristics of the oils from the different fields, it is evident that there is no simple quick test which can be made the basis for assigning the correct value to a particular oil. Some of the known properties of oils, however, are helpful in making a specification or in choosing between two oils of different quality and price.

An oil of high viscosity may cause some trouble in cold weather during unloading and pumping, requiring the use of steam coils. No relation appears to exist, however, between the viscosity and the cracking efficiency of an oil.

In general the fractions resulting from distillation become more valuable for water-gas manufacture as the temperature at which they distil over becomes higher. This is more particularly true when the fractions, which are taken off for every 100 degrees up to 700° F., increase in volume up to that temperature. However, with some oils whose maximum fraction comes off at a considerably lower temperature than 700° F., the volume of the succeeding fractions decreasing rapidly, the high-boiling fractions have been shown to be inferior. The specific gravities of these fractions help to assign a value to an oil. The value of any fraction increases—for a given boiling point—as the specific gravity decreases. When a gas oil of good quality is distilled to a carbon residue, the coke should not exceed 2 per cent.

It is usually considered that the following classes of hydrocarbons in gas oils are valuable for enriching in the order named: (1) Paraffins; (2) olefines; (3) naphthenes; (4) asphaltics; (5) aromatics.

Since the paraffins are the best gas-making oils, such tests as sulphonation or bromination are of value as a means of distinguishing between the amount of paraffins and unsaturated hydrocarbons and aromatics present. A low bromination value indicates a high percentage of paraffins.

A very ingenious method of comparing gas oils was devised in the Peoples' Gas Light Co. laboratories and described in the proceedings of Illinois Gas Association, 1916. By this method, oil is cracked in the laboratory under conditions approximating those in actual practice. By making all the oil tests under the same conditions, and allowing the same time of contact at the same temperatures, com-

parative results can be obtained. In this article are given the results of tests made on more than a dozen oils, including such tests as specific gravity, degrees Baumé, viscosity, flash point, burning point, distillation results (giving fractions for each 100° F. up to 700° F.), bromine number, B. t. u. per pound, per cent sulphur, and cracking results, which include B. t. u. per gallon, per cent of tar, per cent of gas, per cent of carbon, and B. t. u. efficiency. In addition, a study was made of the effect of changes of temperature on the per cent of carbon, tar, and gas formed, and curves are plotted for the results. Table 12 (see p. 67) shows the effect of cracking oil in different atmospheres, such as nitrogen, carbon monoxide, and hydrogen. The results obtained by the use of this apparatus coincide with the conclusions of other investigators, who have also pointed out the superiority of the paraffin hydrocarbons for carbureting water gas.

The sulphur in the oil varies with the different oils and may be as low as 0.1 per cent or as high as 2 per cent. The value to be put on the sulphur test will naturally depend on the equipment of the plant where the oil is to be used for removing sulphur from the gas. The importance of this test will have to be judged by the plant superintendents, who know their own conditions best. The amount of sulphur in the unpurified carbureted gas is, of course, not directly proportional to the sulphur in the oil, although an increase in the sulphur content of the oil increases the per cent of sulphur in the gas, other things being equal.

TEMPERATURE OF CHECKER BRICK.

Regulation of the temperature of the checkerwork in the carburetor and superheater is essential for efficient oil cracking. The effect of the temperature on the cracking of oils has been studied by several investigators and a number of significant facts have been pointed out. The variety of conditions under which the various experiments were made and the differences in the oils tested often make absolute quantitative comparisons difficult, if not impossible. In general, it may be said that, other conditions remaining the same, the effects of temperatures in the checker chambers on the results obtained are as follows:

1. The candles per gallon reach a maximum between 1,300 and 1,350° F.
2. The percentage of illuminants reaches a maximum between 1,300 and 1,400° F.
3. The per cents by volume of methane and hydrogen increase with increasing temperature.
4. The volume of oil gas increases with the temperature.
5. The per cent of carbon formed from the cracking of the oil increases with the temperature.

6. The per cent of tar formed decreases with increasing temperature.

7. The B. t. u. per gallon of oil increases with the temperatures and quite often the maximum is not reached under 1,500° F.

The above conclusions apply, in general, to temperatures up to 1,600° F., which is about the upper limit of the experiments made, so far as data are available. In plants making gas to a heating-value standard, it would seem, on consideration of the above facts, that the higher the temperature carried the more favorable would be the results. However, the formation of an increasing amount of carbon from the oil, as temperatures increase, and an increasing formation of naphthalene preclude the possibility of operating with temperatures much in excess of 1,400° F. With excessive temperatures, carbon rapidly clogs the interstices between the checker bricks and the offtake connections of the gas set. The naphthalene is carried forward some distance with the gas and, when cooled, is deposited upon parts of the works connections and of the distribution system that are difficult to reach, causing serious stoppages.

It is therefore generally desirable to operate with as high a temperature in the checker chambers as good practice has shown to be consistent with the above facts. Temperatures between 1,300 and 1,400 °F. seem most favorable when the ordinary gas oils are used.

There are many gaseous reactions that take place, or may take place in the checker chambers, between the various products of oil cracking. The reaction velocity is different for the various reactions and is affected by the temperature and other variables. Many of the primary gaseous reactions have been studied by investigators. W. F. Rittman^{*} discusses some of these very completely and gives the equilibrium constants for a number of these reactions at various temperatures.

SPACING AND CONDITION OF CHECKER BRICK.

The spacing of checker brick in the carburetor and the superheater is an operating detail that has considerable bearing on the oil cracking efficiency. The bricks absorb the heat produced during the combustion of the blast gas in the checker chambers and give up the stored heat to the oil, oil gas, and blue gas during the run. Increasing the space between the checker brick produces the following results:

1. Decreased number of brick in a shell of given size.
2. Decreased surface exposed for contact with the oil vapor and gas.
3. Decreased quantity of heat stored at any given temperature.

^{*} Rittman, W. F., *Thermal reactions in carbureting water gas*. New York, 1914; *Thermal reactions of petroleum hydrocarbons in the vapor phase*: Jour. Ind. Eng. Chem., vol. 7, 1915, pp. 945-953; *Utilisation of aromatic hydrocarbons derived from cracked petroleum*: Jour. Ind. Eng. Chem., vol. 7, 1915, pp. 1014-1019.

4. Increased per cent of voids or free space. Other things being the same, further results of this change are:

- a.* Decreased intimacy of contact of the oil vapor and bricks.
- b.* Increased time of contact.
- c.* Possible decreased temperature of the gases being fixed in the carburetor.
- d.* Possible increased temperature of the waste gases at the stack.
- e.* Decreased internal resistance in the set.
- f.* Decreased linear velocity of the gas.
- g.* Decreased intimacy of the mixture of blast gases with secondary air.

All of these variables affect the cracking of the oil. They are closely interrelated, and since in practice it is very difficult so to fix the conditions that one variable at a time can be studied, it follows that present knowledge of the effects to be attributed to each of these variables is very incomplete. In discussing them, therefore, tendencies only can be indicated, and it is quite impossible at the present time to fix the magnitude of the effect to be attributed to each variable.

INTIMACY OF CONTACT.

By the intimacy of contact is meant the thoroughness with which the gas comes in contact with the surface of the bricks. An increase in the spacing decreases this intimacy of contact, and vice versa. Although the extent to which the gas comes in contact with the bricks is of great importance in the practical operation of a water-gas set, yet it is difficult to assign a definite value to this variable for a given set of conditions, for the reason that a change in the spacing of the bricks not only affects the intimacy of contact but produces other changes already mentioned.

The surface of the brick has an effect, be it catalytic or simply a matter of heat transference, or both, on the cracking of the oil, and changes with a change in intimacy of contact. There are many ways of checkering carburetors and superheaters, and with some of these methods the intimacy of contact can be changed without changing the time of contact.

Figure 2 illustrates the usual "staggered" checkering which compels the gas to take a sinuous course in passing through it. With this method of checkering, a change in the spacing of the bricks directly influences the time of contact and other variables.

Figure 3 illustrates checkering in what is known as flue formation. The bricks are not staggered but are laid in such a way as to form numerous flues through which the gas passes. As with the staggered formation, a change in the spacing changes the time of contact;

however, a change from the system shown in Figure 2 to that of Figure 3, using the same number of bricks and the same distance between rows, alters the intimacy of contact without affecting the time of contact of the gas. In changing thus from the arrangement of Figure 2 to that of Figure 3, the intimacy of contact is decreased. This means that during the blasting period the blast gases pass from the stack at a higher temperature and more blasting is required to bring the temperature in the checker chambers up to the desired degree, which results in an increased fuel consumption.

Intimacy of contact can be changed further without affecting the time of contact, gas velocity, and other variables, by changing the

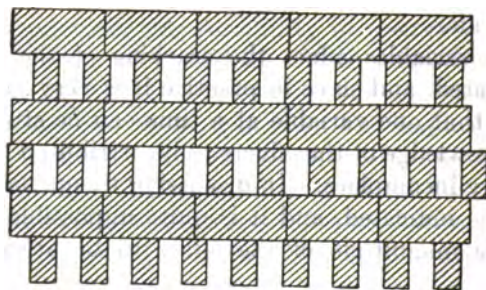


FIGURE 2.—Staggered checkering.

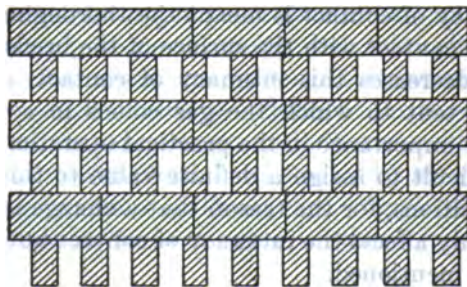


FIGURE 3.—Flue checkering.

size of the bricks. For example, when $2\frac{1}{2}$ -inch bricks spaced 4 inches apart are used, the intimacy of contact is not nearly so great as when $1\frac{1}{2}$ -inch bricks spaced 2 inches apart are used, although the time of contact is practically the same in each case.

TIME OF CONTACT.

The time of contact designated in this paper is the time required for any given molecule of gas to pass through the checker chambers, and may be computed as follows:

I is the time of contact in seconds; S is the total free space in the checker chambers (when checkered) in cubic feet; and V is the volume of the gas in cubic feet flowing through the checker chambers per second.

$$\text{Then, } I = \frac{S}{V}.$$

The time of contact calculated from this formula represents the time the gas is in the checker chambers, but not the time the gas is in actual contact with the bricks.

Available information on this subject seems to be meager, due perhaps to the fact that so many variables, about which little is known, must be considered at the same time. Another reason may be that the time of contact is continually varying during the gas-making period. The make of blue gas during the first half minute of the run may be four times as great as that during the last half minute. Under these circumstances, the time of contact is increasing from the beginning to the end of the make period, while the velocity is decreasing.

Obviously, as the spacing between the bricks is widened the volume of actual free space within the checkerwork increases, and if it be assumed that the volume of gas flowing through the checkerwork in a given time remains the same, it is evident that the time of contact computed as stated above will increase and the gases will remain a longer time within the checkerwork. Time is an important factor affecting the completeness of oil-cracking reactions; therefore, *other conditions remaining the same, the longer the time of contact, the more complete the oil cracking.*

When oil is being cracked in a water-gas carburetor, *intimate contact is desired, with a time of contact sufficiently great to gasify the oil.* Too great a time of contact may result in the production of naphthalene or of lamp black.

EFFECT OF CHECKERING ON TEMPERATURE OF GASES BEING FIXED.

A change in the spacing of the checkerwork changes the total volume of brick and consequently the total heat capacity of the checkerwork at any given temperature. Since the heat for fixing the oil gas is derived from the brick, it is evident that with a smaller or a greater amount of stored heat available for this purpose any given volume of gas will be heated correspondingly to a lower or a higher temperature. It has already been pointed out that as the spacing of the checkerwork is widened the intimacy of contact of the gas with the brick surfaces is diminished, and consequently there is less opportunity for the heat of the brickwork to be transmitted to the gas by surface contact. This alone tends to decrease the temperature of the gas, if the quantity of the gas in both cases is the same.

EFFECT OF CHECKERING ON THE TEMPERATURE OF STACK GASES.

The bricks of the checker chambers are heated by the combustion of the generator blast gases when the secondary air is admitted to the carburetor.

The blast gases transmit a portion of their heat to the brick during the blow period. It is evident that any increase in the spacing of the checker brick decreases the number of bricks that can be placed in the space allotted for checkerwork. Consequently, with wider spacing the total volume of brick decreases and with it the total heat storage capacity of the brickwork decreases for any given temperature. Since the brickwork can not absorb so much heat from the blast gases per unit of time, the temperature of the gases leaving the machine will be higher and the loss of sensible heat in the stack gases will be greater, due to the higher temperature now prevailing.

EFFECT ON INTERNAL RESISTANCE OF SET.

When the checkerwork is more widely spaced, the percentage of free space increases, as has been noted, and the internal resistance of the checker chambers to the passage of gas decreases. With given air pressures, available during the blast, more air can be forced through the fire as the internal resistance of the set decreases. This of itself is an advantage if the wider spacing does not adversely affect the other conditions mentioned above.

DECREASED VELOCITY OF GAS.

With increased free space and consequent increased time of contact, the linear velocity of the gases passing through the checkerwork necessarily decreases, since the total volume of gas passing per unit of time is assumed to remain constant.

This change in velocity affects the deposit of dust-like material carried mechanically by the gases passing through the checkerwork. Its further effects on temperature, deposit of carbon, and efficiency of cracking are intimately connected with the time and intimacy of contact, under which heading they have been discussed.

EFFECT ON INTIMACY OF MIXTURE OF BLAST GAS WITH SECONDARY AIR.

In order to obtain complete combustion of the generator blast gases in the checkerwork, it is essential to have an intimate mixture of the gas and the secondary air. This intimacy of mixture is facilitated by the many impingements made by the mixture on the brickwork, and the extent of this impingement decreases as the spaces widen.

QUALITY AND QUANTITY OF BLUE GAS.

The quality of the blue gas, in which atmosphere the oil is cracked, affects the final products in the finished gas. Oil has been cracked in various gas atmospheres on a laboratory scale and the results have been studied. Downing and Pohlman⁷ state that at the temperatures

⁷ Downing, R. C., and Pohlman, E. F., Cracking of gas oils in various atmospheres: Gas Jour., vol. 137, 1917, pp. 24-26.

usually prevailing in the checker-work, the B. t. u. content per gallon of oil used is greater when the oil is cracked in blue gas of good quality than when the oil is cracked in CO_2 , N_2 , or CH_4 alone. An increased amount of CO_2 in the blue gas lowers the B. t. u. efficiency of oil cracking. However, it was pointed out that the B. t. u. content of the gas, per gallon of oil cracked, is greatest when there is 20 per cent steam in the blue gas. It seems to be true that the oil cracks better in blue gas of good quality than in that of poor quality, which gives additional incentive to produce the best possible grade of blue gas in the generator. When cracking oil in blue gas of good quality at the usual temperatures—1,300 to 1,400° F.—the quantity of gas and tar is greater and the per cent of carbon formed is less than when the cracking takes place in blue gas of poor quality. During the make period the quantity as well as the quality of blue gas is continually decreasing, while the per cent of excess steam is usually increasing. The efficiency of oil cracking is affected by these factors in opposite directions. As the volume of blue gas decreases during the run, the concentration of the oil gas increases (with the constant input of oil), and the time of contact is correspondingly longer, which tends toward the formation of an increasing amount of carbon. Simultaneously the per cent of steam in the blue gas is increasing, and with it the oil-cracking efficiency is increased, with a diminished tendency to deposit carbon. The degree to which these two opposing factors offset each other depends on a number of other variables; hence, it is difficult to assign definite values to them.

The quantity of oil admitted to the carburetor per unit of time affects the concentration of the gas in the carburetor, as well as the time and intimacy of contact of the oil gas with the checker brick. This, of course, directly affects the yield of gas, tar, and carbon. If oil is added so rapidly that the surface of the top bricks in the carburetor is cooled, owing to the conductivity of the brick being too low to transmit its heat to the surface quickly enough, there is a tendency for distillation, and not cracking, to take place, and the effect is a deposition of a coke-like carbon on these bricks. This tendency is greater with some oils than with others. In fact, the quality of the oil is often a more important factor than the rate of oil input in governing the quantity of tar, carbon, and similar substances formed.

PRESSURE.

All gaseous reactions are affected in some degree by pressure. Rittman has shown⁸ that when in the cracking of oils pressure is increased to higher than atmospheric the following results occur: First,

⁸ Rittman, W. F., *Thermal reactions in carbureting water gas*. New York, 1914.

an increased amount of carbon is formed; second, an increased volume of methane; and third, a decreased volume of illuminants.

The temperature of cracking and the extent of increase of pressure affects the results just given. Although slight changes in pressure in the checker chambers of a water-gas set do not alter the results to any extent, the tendencies should be noted.

EFFICIENCY OF THE OIL SPRAY.

The efficiency of oil spraying is of great importance in water-gas manufacture. The spray nozzle usually needs more attention than it gets. When the carburetor is checkered, great care is usually taken to have the last course (top course) of checker bricks at such a height that the nozzle can distribute oil uniformly over the entire top course. However, the nozzles themselves are not always tested to see that they are performing as desired. When a spray spreads the oil too heavily in certain spots, the result is that the surfaces of the bricks receiving the most oil become cooler than the adjacent bricks. Distillation takes place to a considerable degree around these cooler bricks, and a deposit of carbon forms that increases with time. While this part of the checkering is being choked with carbon, the adjacent bricks may become so hot that they fuse and run together. When nozzles of proved efficiency are not in use, a change of nozzle from day to day will help somewhat in overcoming this undesirable result.

APPLICATION OF PRINCIPLES TO PRACTICAL OPERATION WITH LOW-VOLATILE FUELS.

In the foregoing section, the operating principles discussed have been general, and although their interrelations have been dealt with to a certain extent, it remains to be shown how they are affected by the changes and conditions of practical operation.

Most of the operating variables of a water-gas plant are interdependent; in other words, a change in one affects many others. That this is true has been realized by most operators, some of whom attempted to study the effects of various changes, considered the problem bewildering, and stopped discouraged. R. P. Harper has listed 289 variables and indicated their interrelations in his article in the *American Gas Engineering Journal*.⁹ Some of these variables are listed below, although no attempt is made to enumerate all of them.

⁹ Harper, R. P., Gas machine factors involved in the manufacture of carbureted water gas: *Am. Gas Eng. Jour.*, vol. 116, 1919, pp. 312-316, 320-324.

- a. Blast.
 - Amount of air blown in total blasting time.
 - Quality—Dry or moist.
- b. Steam.
 - Quality—Dry or wet.
 - Quantity per unit of time—Dependent on variable steam pressure as well as intentional change in valves.
 - Total length of steam run.
- c. Fuel.
 - Kind—Physical and chemical properties.
 - Size—
- d. Depth of fuel bed.
 - Amount charged at a time.
 - Frequency of charging.
- e. Cycle.
 - Duration of blow.
 - Duration of steam run.
- f. Proportion of up and down runs.
- g. Clinker and ash in generator.
- h. Kind of oil used and time of admission into carburetor.
- i. Condition and spacing of checker bricks.
- j. Efficiency of oil spray.
- k. Temperatures maintained in carburetor and superheater.
- l. Quality and quantity of blue gas made in generator during run.
- m. Combinations of the above variables.

In attempting to control and regulate the operating variables, it is essential from a practical viewpoint to achieve the following:

1. Make the maximum amount of standard quality gas possible per day or running hour.
2. Use the least amount of generator fuel possible per 1,000 cubic feet of gas made.
3. Use the least amount of steam possible per 1,000 cubic feet of gas made.
4. Use the minimum amount of oil for carbureting per 1,000 cubic feet of gas made.
5. Accomplish these four purposes with the lowest possible operating and maintenance costs.

The accomplishment of these results depends, among other things, upon the maintenance of suitable temperatures in the generator and checker chambers. The various processes must be so arranged that all the chambers attain their operating temperatures at the same time. A temperature balance between the various parts of the set must be created and preserved within reasonable limits at all times during operation.

MAINTENANCE OF A HOT GENERATOR.

The maintenance of a hot generator is one of the first problems to be solved in applying theory to practical operation. Tables 3 and 4 (see pp. 5 and 6) show the necessity of maintaining high temperatures in the fuel bed for efficient operation.

When endeavoring to preserve the desirable high temperature in the fuel bed, the variables affecting this temperature must be given due consideration. Some of the prime factors on which it depends are as follows:

1. Air blast.
 - Duration of blasting period.
 - Quantity of air per minute.
 - Condition of air—dry or moist.
2. Steam run.
 - Duration of steam run.
 - Quantity of steam per minute.
 - Dry or wet steam.
3. Condition and depth of fuel bed.
 - Frequency of charging.

AIR BLAST.

Since air blasting accomplishes the double purpose of heating the fuel in the generator and heating the checker brick in the carburetor and the superheater, it is well to keep in mind the conditions governing reactions 4 ($C + O_2 = CO_2$) and 3 ($CO_2 + C = 2CO$) given on page 8.

The length of the blasting time depends, of course, on the amount of fuel in the generator and the quantity of air per minute that can be forced through the bed with the available blast. The zone of complete combustion is, normally, that part of the fuel bed directly over the grates, possibly not more than a foot in depth. The thermal reactions taking place above this zone are endothermic. (See reaction 3 ($CO_2 + C = 2CO$) on page 3.) Since the fuel in the upper part of the generator is not heated by direct combustion of the fuel but chiefly by convection, it is evident that, other things being equal, the rate of the blast has considerable influence upon the regulation of temperature in the fuel bed. A prolonged blast, with a low blast pressure, is usually not so desirable as a high blast pressure with a short blasting period. This is true not only on account of the time saved with the shorter blasting period, but also on account of the better distribution of heat through the fuel bed.

Moisture in the air blast also has a marked effect on the temperatures in the generator. Study of the endothermic reactions 1 and 2, on page 3, clarifies this point and emphasizes the desirability of dry air; hence great care is usually taken that the exhaust steam from the turbine is not drawn into the fan or the blower.

STEAM RUN.

With a properly timed cycle, as much heat is abstracted from the generator fuel bed during the run as was stored in it during the blow. The actual duration of the steam run is governed by the quantity of steam admitted per unit of time. It is desirable to use steam at such a rate that the length of the run is little if any greater than the

length of the blast period, when average blast pressure is used. Decreasing the quantity of steam and increasing the steaming time increases the time of contact both in the generator and fixing chambers, which may or may not be advantageous, according to existing conditions.

The steam should be dry or as free from moisture as possible. Water that condenses in the main steam-supply lines can be removed by a steam separator. Every pound of water entering the generator as such absorbs approximately 971 B. t. u. in changing from water at 212° F. to steam at the same temperature.

In order to maintain the desired fuel temperature in the generator, the quantity of air and steam must be correctly proportioned. This is best accomplished by closely studying the ill-effects of an excess either of air or of steam. Consider, for example, a fresh fire operating with a 3-minute blast and a 3-minute steam run, with insufficient steam. It is apparent that under these conditions, in which not enough steam is added during the run to absorb the heat stored in the generator during the blast, the generator fuel will become hotter as the day proceeds. This naturally results in the generation of more combustible gas (reaction 3 being more complete at high temperatures) during the blast, which, if burned in the carburetor and superheater, increases their temperatures to an undesirable degree. Even if the temperature in the checker chambers is reduced by using an excess of carburetor air blast, the waste of heat in the blast gases is excessive and involves the consumption of additional generator fuel per 1,000 cubic feet of gas made. Moreover, the excessive temperature in the generator brings about a more complete fusion of the ash and occasionally a very hard clinker may be formed.

Operating with the same cycle but using an excess of steam, or—what amounts to it—an insufficient amount of air blast, the opposite effects are noted. The generator temperature becomes so low that the CO_2 formed in the zone of complete combustion during the blow does not combine further with carbon to form CO as in reaction 3 on page 8. Insufficient combustible gas is therefore produced during the blast period to furnish the necessary heat for the carburetor and superheater and the temperatures in these chambers gradually decrease. This results in improper cracking of the oil as evidenced by oil showing at the seal pot and by an increased oil consumption per 1,000 cubic feet of gas made. If the generator becomes cooler as the day proceeds, the clinker is not usually hard; in some cases there is no clinker, but an accumulation of ash. The blue gas is not so rich when this excess of steam is used. Tables 1, 3, and 4 show that, at the reduced temperatures now prevailing in the generator, more CO_2 , less CO , and an increased amount of undecomposed steam will be present in the blue gas. This fact is utilized in some plants to

determine the efficiency of operation. To do so, the finished gas is tested for CO_2 . An increase over a fixed standard, which is considered to be a satisfactory one, is usually taken to denote the use of an excessive amount of steam during the run.

CONDITION AND DEPTH OF FUEL BED.

The condition of the fire in the generator not only affects the temperature in the generator, but the output or "make" as well. The accumulation of ashes leaves a decreasing space for incandescent fuels, hence the last runs before cleaning the fire (under usual operating conditions) are made with less incandescent fuel than earlier in the day. Since the same cycle is usually employed throughout the day, it is obvious that the quality and quantity of the gas produced are affected as already stated. This is especially the case when an inferior fuel containing a high per cent of ash is used.

When too many runs are taken off a charge, that is, when the charging interval is too long, the fuel in the generator is too low for a considerable part of the day. This decreases the volume of incandescent fuel in the generator, and, like the accumulation of ash in the generator, results in a decreased quality and quantity of blue gas being made during the run. When the charging interval is long and the ash has accumulated rather extensively, the results are: A marked decrease in the make per run, an increased steam consumption per 1,000 cubic feet of gas made, and an increased per cent of CO_2 in the finished gas.

To summarize, it may be said that since a high temperature in the generator fuel bed is necessary for efficient operation, the following facts should be kept in mind:

1. As deep a fuel bed as possible should be maintained in the generator to permit a greater quantity of steam to be decomposed and to allow a longer time of contact of the gases with the incandescent fuel.

2. Charges should be made often enough to prevent the fuel in the generator from becoming excessively low, for the same reason as in 1.

3. An excessive amount of ashes or clinker should not be allowed to accumulate in the generator.

4. The air blast should be as free from moisture as possible.

5. The maximum amount of available air-blast pressure should be used at the generator when maximum capacity is desired, thereby obtaining the maximum introduction of air to the fire per unit of time.

6. The use of an excessive amount of steam should be carefully avoided during steam runs. This precaution not only aids in maintaining the proper temperature in the generator, but helps to keep the fuel consumption per 1,000 feet of gas down to a minimum.

7. When there is a tendency for steam to condense in the steam lines, the use of a steam separator is advisable.

8. The steam used during the run should be as dry as possible.

9. Leaky steam valves permitting up steam to enter the generator when a down run is being made, or vice versa, should be watched for and corrected when found. Leaky hot valves that permit steam to by-pass the fire should also be avoided.

10. Dirt and carbon should not be allowed to accumulate under the gates of the hot valves.

OPERATING CYCLE.

The selection of the best operating cycle is an individual problem for each operator. Since the quantity of air and steam available per unit of time varies to a considerable degree in different plants, it is obvious that a fixed standard, such as a 3-minute blast and a 4-minute run, or a 4-minute blast and a 3-minute run, can not be specified for all cases. Since steam produces better gas at the higher temperatures, a very short operating cycle would be desirable were it not for the fact that operating the valves requires an appreciable amount of time. Figure 4 shows graphically the relative amount of time consumed in this operation. As short a cycle as possible consistent with maximum make, is desirable. In selecting a typical cycle, take as an example a 3-minute blast using all the available air. Good practice has shown that for every 1,000 feet of air used, approximately $21\frac{1}{2}$ pounds of steam are required. This ratio of steam and air applies only to low volatile fuels and varies somewhat with different fuels of this class. However, it is to be noted that the steam used per 1,000 cubic feet of gas made in practical operation is nearly double the amount theoretically required.

Therefore if x = the air per minute in cubic feet,

$3x$ = the total air during the 3-minute blast,

$\frac{3x}{1,000} \times (21.5 \text{ pounds}) = \text{total steam required per run, or}$

for a 3-minute run = $\frac{(3x) \times (21.5)}{(1,000) \times (3)}$ pounds steam per minute.

If the steam supply is not great enough to give the desired amount of steam per minute, a longer run than 3 minutes will be required for the same 3-minute blow. When a 3-minute blow and a 3-minute run are employed, it may be asked, "Why not use a 2-minute blow and a 2-minute run or a 4-minute blow and a 4-minute run?" The time required to operate the valves is one of the limiting factors in settling this point. The more prolonged the cycle, the smaller is the percentage of time required for changing valves. The average quality of gas made during a prolonged run is not so good, as has already been mentioned. As the cycle is shortened, the percentage

of time required for valve operation increases as shown in Figure 4, but a greater portion of the run is made at a higher average generator temperature, resulting in a better quality and greater quantity per

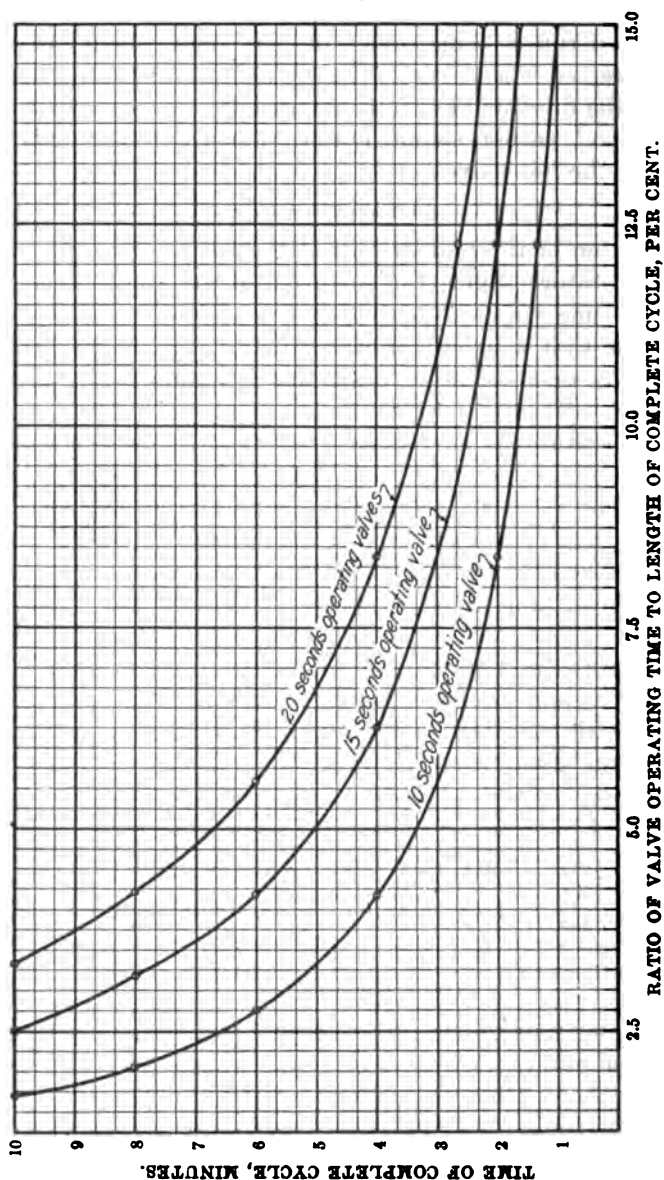


FIGURE 4.—Loss of time due to operation of valves.

minute of blue gas. The shortest cycle that will permit the maximum capacity is desirable. Under average conditions the time of the complete cycle may vary from 4 to 9 minutes when good results are being obtained.

UP RUNS AND DOWN RUNS.

In all modern water-gas sets the direction of steam through the fire during the run is reversed at regular intervals. This reversal has been found to be advantageous in many respects. The methods of making up and down runs vary in different plants. In some the steam is reversed during every run, making so-called "split runs." In others, the down runs are made at regular intervals, all other runs being up runs.

Among the advantages realized from these reversals are the following:

1. The clinker can be caused to form near the grates where it can be removed more easily.
2. The upper hot valve is cooled by down runs, preventing overheating during the up run and blow periods.
3. The generator fuel is consumed more completely, less unburned fuel being removed with the ashes and clinker.

When the number of up and down runs is correctly proportioned, the fuel bed is in a better gas-making condition. An excessive number of down runs is to be avoided, since these would result in the top of the fuel bed being unduly cooled, and the grates would have a tendency to become overheated.

CLEANING THE FIRE.

The accumulation of ash and clinker in the generator reduces the make per run, hence efficiency of operation requires that the fire be cleaned at as frequent intervals as will permit a maximum daily make with the least consumption of time for cleaning. How often the generator should be clinkered must be decided by the individual operators and depends upon the fuel used, the total running time, and other factors. If a set is operated 18 hours a day, clinkering is usually done after a 9-hour running period. If 14 hours is the total day's run, the clinkering is done after 7 hours running. When a fuel with a very fusible ash or a high per cent of ash is encountered, it is often an advantage to clinker more frequently. Under some conditions, three cleans a day can be made in the same time required for two, owing to the fact that much more time and care are required for clinkering when the clinker becomes very thick and is high in the generator. A clean fire is essential for a good make and for the generation of blue gas of good quality. The gas maker is usually aware of the condition of his fire even when the figures for the make per hour are not available to him, since the increased resistance of the fire to the passage of air through it causes an increased air pressure under the grates, other conditions remaining the same, as shown by the water gage, and at the same time the air passing through the fire per minute is decreased, as shown by the air meter.

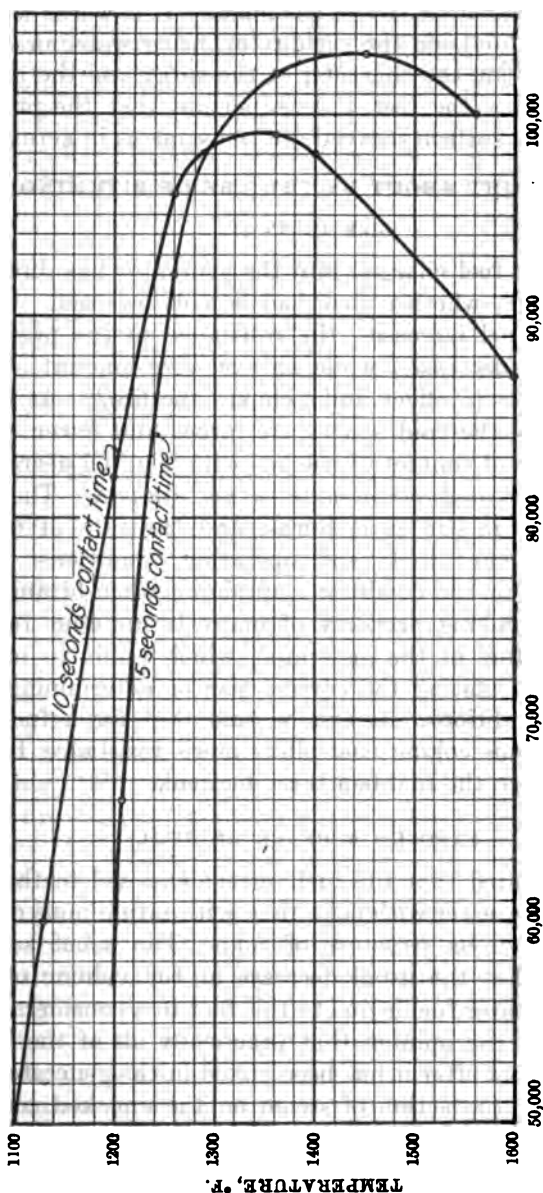
CARBURETION OF GAS.

The time allowed for spraying a definite amount of oil into the carburetor directly affects the time of contact and influences the amount of gas, carbon, and tar formed. If the checkerwork of a 6-foot set occupies 150 cubic feet out of 380 cubic feet total interior volume of the carburetor and superheater, the volume of the remaining free space will be 230 cubic feet. If an average make of 1,000 cubic feet of gas per minute or 16 cubic feet per second at ordinary temperatures is assumed, this will be equivalent to 57.3 cubic feet of gas at 1,400° F. The time of contact will be equal to the volume of free space in the checker chambers divided by the volume of gas passing per second, or $\frac{230}{57.3} = 4.0$ seconds. Toward the end of the run the make falls off or decreases considerably. It may be as low as 600 cubic feet of gas per minute, equivalent to 35.8 cubic feet per second at 1,400° F. The time of contact under such circumstances would be $\frac{230}{35.8} = 6.4$ seconds. Increasing the quantity of oil added per minute increases the volume of oil gas and consequently the total volume of gas passing through the checkerwork per unit of time, decreases the time of contact of the gas with the checker brick, and increases the concentration of the oil gas formed.

When an excessive amount of oil is used, the time of contact is unduly decreased, resulting in incomplete cracking of the oil and the production of an excessive amount of tar. When this occurs, some undecomposed oil passes through the checker chambers and some appears on the surface of the water in the seal pot. Decreasing the quantity of oil added per unit of time increases the time of contact and is more favorable for the production of carbon or naphthalene. A more consistent product and a better regulation of reactions would be obtained if the production of blue gas were more uniform. However, a desirable feature of this changeable condition is that the time of contact, in ordinary operation, is shortest at the beginning of the run, when the temperature in the checkerwork is the highest. In average practice, the time of contact, figured from the total make over the total run period, is usually 4 to 6 seconds. The most desirable time of contact for any given oil under particular conditions can best be ascertained by actual test.

It is often stated in literature pertaining to water-gas manufacture that 2½ or 3 inches is a "nice" spacing for checker brick. Checkering is often installed without the slightest thought of what the consequences of increasing or diminishing the spacing might be. The general effect of changing the spacing of the checker brick on intimacy and time of contact has been mentioned on page 17. In Figure

5¹⁰ is shown the effect of changed contact time on the make at different temperatures. While these curves would probably be different for the various oils used, the relation of the curves for any particular oil, used for different contact periods would probably be the same.



B. T. U. PER GALLON OF OIL CRACKED.

FIGURE 5.—Effect of time of contact and temperature on oil-cracking efficiency.

A consideration of this effect and a study of the results and makes obtained with a particular oil, may show the desirability of a change in the spacing of the checker brick.

¹⁰ Downing, R. C., Some fundamentals affecting the utilisation of gas oil in carbureted water-gas manufacture: Proc. Ill. Gas Asso., March, 1916.

The harmful effects of poor oil distribution in the carburetor have been mentioned in a previous section. In practical operation, the continued use of an inefficient oil spray occasions more frequent cleaning or rechecking of the carburetor and increases the consumption of oil per 1,000 cubic feet of gas made. It is worth while to test the "oil sprays" outside the carburetor, using water or oil, for the purpose of observing the character of the spray and the area that it covers. In making the test, it is important that the pressure and distance from the surface sprayed are the same as in actual use.

CHANGES BROUGHT ABOUT BY THE USE OF BITUMINOUS FUELS.

SIZE OF COAL.

The size of the fuel charged into the generator has direct bearing on gas production, whether that fuel be coke or coal. Usually, as the size of the fuel increases, the resistance to the passage of air through the fuel decreases, while an excessive amount of very fine fuel has the opposite effect and "chokes the fire." As the size of the fuel increases, the void spaces are larger and fewer in number, and the intimacy of contact of the gas with the fuel decreases, *when the volume of air used per minute is not changed*. The fuel constantly decreases in size as it burns, and the rate of combustion varies with different fuels and operating conditions. Attention here is confined to the conditions obtaining when bituminous fuels are used. The coking property of many bituminous fuels causes matting of the fuel in the generator, which tends to increase the difficulties of calculating the correct size or determining one best size for given conditions. It can be said that for a fuel of given size the bituminous coking coal offers more resistance to the passage of air through the fuel bed than does coke.

EXPOSED SURFACE OF FUEL.

It has been stated that the fuel surface exposed to the action of air and steam is greater with coke than with bituminous coal, due to the shape and porous structure of coke. The belief seems to be quite common that the usual decrease in the volume of gas from bituminous generator fuel is due to this fact to a considerable extent. The author is of the opinion that practically all of the blue gas is made from the coal after it has been coked in the generator and that little is made by the action of steam on the uncoked coal. It has been shown by Boudouard¹¹ that at 650° C. (1,202° F.) it takes 12 hours contact time for equilibrium to be reached between CO₂ and carbon, and at the end of this time only 39 per cent of CO is formed from 100 per cent of CO₂. It is quite evident that practically no

¹¹ Boudouard, O., Sur la décomposition de l'acide carbonique en présence de charbon: *Compt. rend.*, t. 128, 1899, pp. 824-826.

blue gas is made at or below this temperature. Furthermore, coal is coked, at least on the surface, at this temperature. Evidently the surface differences between coal and coke do not, of themselves, cause a great difference in the capacity of a given set.

CHARACTERISTIC PROPERTIES OF COAL AND COKE.

The properties of bituminous coal are quite different from those of coke, and it is not surprising that these differences require special consideration when the one is substituted for the other as generator fuel. Some of these differences are the following:

| | Coal. | Coke. |
|---|------------------|------------------|
| Volatile matter.....per cent | 30 to 40 | 3 to 8 |
| Moisture.....do.... | 6 to 12 | a 5 |
| Ash: | | |
| From Illinois coal.....do.... | 10 | 16 |
| From eastern coking coal.....do.... | 6 | 9 |
| B. t. u. per pound..... | 11,500 to 13,000 | 12,000 to 13,500 |
| Caking tendency shown when heated in a generator..... | Variable. | None. |

a Unless freshly quenched or standing in rain.

The per cent of ash given for coke is greater than that for coal. When coke is compared with the coal from which it is made, this relation is correct. However, coke made from eastern coal may have less ash than some Central District bituminous coals.

EFFECT OF COKING PROPERTIES.

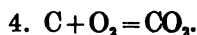
The caking or coking property of bituminous coal determines, to some extent, the operation and performance of a generator. It affects, among other things, the quantity of air per minute passing through the fire with a given blast pressure, the size of the fuel charge, the frequency of charging, the size of fuel that can be used, and the temperature in the generator.

When a coking coal is being blasted volatile matter and moisture are expelled from the coal as the temperature is raised, and the outer surface, which is the first to become hot, begins to coke. When this coking takes place there is a natural tendency for the fuel to mat. This mat offers more resistance to the passage of air than does loose fuel, and if means are not found to prevent caking, the resistance to air may become so great as to reduce the air per minute of blast, at a given pressure, as much as 20 per cent. Large fuel charges at long intervals or charges of small fuel accentuate the matting tendency, while with smaller and more frequent charges of large fuel the matting decreases. The matting of the fuel reduces the temperature in the generator, due to the decreased amount of air that can pass through the fuel bed per minute at a given blast pressure.

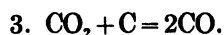
QUALITY OF BLUE GAS AND BLAST GAS.

When the usual operating methods applied to coke are employed with bituminous coal, the fuel in the upper part of the generator is ordinarily not heated to so high a temperature as when coke fuel is used. This top fuel is being distilled during the run, and the distillation products alter the composition of the gas leaving the generator. The gases produced during the run and the blow with coal as generator fuel are therefore different from the gas produced by coke fuel. Analyses of these gases have been made at various stages of the operation and are given on page 35.

When fuel is blasted in a water-gas generator, combustible gas is produced which is burned by the addition of secondary air in the checker chambers. This gas is produced as a result of two chemical reactions, which have been mentioned previously and are repeated here for clearness. The first reaction to take place is represented by equation 4:



In this reaction there is complete combustion of the carbon to CO_2 , which is not combustible. The heat of combustion of the carbon is, to a great extent, taken up or absorbed by the fuel in the generator. As this fuel becomes hotter, a point is reached where the CO_2 combines further with carbon and produces CO as in equation 3:



The CO thus generated is the combustible gas that may be burned in the checker chambers. As the fuel in the generator becomes hotter, reaction 3 takes place to a greater extent until practically all of the CO_2 produced is converted into CO before it leaves the generator. Since there is only 20.9 per cent by volume of oxygen in the air, only 20.9 per cent of carbon dioxide could be obtained in reaction 4, while, according to equation 3, 41.8 volumes of CO can be produced from 20.9 volumes of CO_2 .

When a fuel containing a high per cent of volatile matter is blasted, the quantity of combustible blast gas produced is increased by the amount of volatile matter driven off during the blow if other conditions remain the same. This volatile matter usually has a greater heating value per cubic foot than CO has; hence the heating value, as well as the volume of the blast gas, is increased. The heating value and volume obtained depend upon a number of variables, including the following:

1. Kind and size of fuel.
2. Volatile content of the fuel.
3. Cycle.

4. Temperature attained in the generator.

5. Rate and volume of generator air blast.

The following average analyses are typical of blast gases sampled at the generator lid with coke and bituminous coal as fuel:

TABLE 5.—Average composition of blast gas from coke and bituminous coal.

| Fuel. | CO ₂ . | O ₂ . | III. | CO. | CH ₄ . | H ₂ . | N ₂ . | Total. | B. t. u. per cubic foot. |
|------------------------------------|-------------------|------------------|------------------|------------------|-------------------|------------------|------------------|------------------|--------------------------|
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | |
| Coke..... | 10.0 | 0.4 | 0.0 | 21.0 | 1.2 | 0.0 | 67.4 | 100 | 90 |
| High volatile bituminous coal..... | 7.8 | .0 | .3 | 18.7 | 3.2 | 6.7 | 63.3 | 100 | 123 |

Different fuels when-blasted will give gases of different composition. The above table merely indicates the differences in composition and heating value that may be expected when similar fuels are used.

Similarly, with-high volatile fuel the blue gas produced during the steam run is different in quality and quantity from that when coke is used. The quality is improved by the volatile matter from the coal, the extent of improvement depending on the amount and kind of volatile matter in the fuel, on the variables just mentioned pertaining to blast gases, and on the quality and quantity of steam used during the run. The following average analyses are typical of blue-gas samples taken at the generator lid with bituminous high-volatile coal and coke fuels:

TABLE 6.—Analyses of blue gas sampled at generator lid.

| Kind of fuel. | CO ₂ . | O ₂ . | III. | CO. | CH ₄ . | H ₂ . | N ₂ . | Total. | B. t. u. per cubic foot. |
|------------------------------------|-------------------|------------------|------------------|------------------|-------------------|------------------|------------------|------------------|--------------------------|
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | |
| Coke..... | 6.7 | 0 | 0.0 | 35.7 | 0.3 | 53.0 | 4.3 | 100 | 291.5 |
| High volatile bituminous coal..... | 8.0 | 0 | .5 | 30.5 | 3.6 | 50.0 | 7.4 | 100 | 314 |

As shown in this table, the quality of blue gas from bituminous coal is higher than that from coke, on account of the increased per cent of hydrocarbons present. This has a marked effect on the amount of oil required for carburetion (see p. 71).

Other conditions remaining the same, the volume of blue gas produced per run or per unit of time is usually less when a coking bituminous coal is used in the generator. There are a number of causes for this decreased output, of which the following are probably the most important:

1. Increased resistance to the air blast offered by the coking coal in the generator on account of the tendency to mat.

2. Decreased volume of air per unit of time, with the same blasting pressure, causing a decreased average temperature of the fuel in the generator.

3. Decreased volume of high-temperature fuel.

In addition to the above a number of other factors doubtless influence the production of blue gas. Among these are the following:

- a. Difference in friability of coal and coke.
- b. Difference in conductivity of coal and coke.
- c. Difference in rate of combustion.
- d. Difference in clinker conditions.

The relative importance of these factors varies with different fuels and definite values can not be assigned to any particular factor.

EFFECT ON CARBURETOR AND SUPERHEATER.

HEAT BALANCE.

Since the blast gas is richer when high-volatile fuel is used, a shorter blast is required to produce a definite amount of heat in the checker chambers by the combustion of the blast gas. This means, in the final analysis, either that the checker chambers become too hot during the course of the day's operating, or that the generator becomes too cold. The heat balance is disturbed, and it is apparent that a different cycle or a different method of operation must be employed to obtain the best results with this fuel.

DEPOSITS IN CHECKER CHAMBERS.

Occasionally powdery or granular deposits of carbon have been observed in the checker chambers, more particularly in the superheater. The quantity of carbon deposited, whether it is a powder or the solid deposit often found in the carburetor, depends on a number of variables, including the kind of oil used, the time in which oil is admitted, the spacing of the checker brick, the temperature in the checker chambers, the temperature in the generator, the kind of generator fuel, and the method of admitting secondary air to the carburetor.

EFFECT ON OIL CRACKING.

The variable conditions affecting the cracking of oil have been discussed in a previous section (see p. 16). The use of coal as generator fuel changes many of these conditions, with a consequent effect on the efficiency of oil cracking. The chief changes brought about are in the quantity and quality of the blue gas and the temperature in the cracking chambers.

SMOKE PRODUCTION.

The production of smoke is a characteristic feature of bituminous coal as fuel. Since the surface fuel in the generator is relatively cool, the fuel emits a tarry vapor when blasted. If the average tempera-

ture of the generator fuel is particularly low, the gases first generated on blasting are chiefly CO_2 and N_2 . Since this mixture is incombustible it is difficult to burn the tarry vapors produced with these gases. The temperature of the fuel in the generator, then, has considerable to do with the production and possibility of elimination of smoke.

ASH AND CLINKER FORMATION.

With the lower generator temperature that ordinarily prevails when bituminous fuel is used, there is less tendency for clinker to form and a greater tendency for ashes to accumulate. These ashes may occupy a greater volume, weight for weight, than the clinker, and are detrimental to the quantity production of good water gas. However, coal will produce a very hard clinker under some conditions, and the fusibility of the ash of the various coals differs greatly.

EFFECT ON PURIFICATION.

If the gas is thoroughly cleaned before it reaches the purifying boxes, there is no reason to believe that the efficiency of the purifying process will be affected in any way by the use of coal fuel unless it contains an excessive amount of sulphur. On account of the larger amount of tar produced with coal the gas is not thoroughly cleaned in some plants, and the purifying material has been known to be ruined by lack of care in this respect.

USUAL METHODS OF OPERATING WITH BITUMINOUS COAL FUEL.

HEAT BALANCE AND CYCLE.

With the changed composition of the blast and blue gases, it is natural to suppose that the heat balance throughout the set would be disturbed by the changes brought about in substituting coal for coke in the generator, and such is actually the case. In the usual operation with coal fuel, the checkering tends to become too hot, frequently rising as high as $1,600^\circ \text{F.}$, while the generator fuel is not hot enough. This is due principally to the greater heating value of the blast gas, which in turn can be explained by the presence of considerable volatile matter from the coal. The greatest obstacle to the substitution of coal for coke as generator fuel is the maintenance of high temperatures properly proportioned throughout the different parts of the set. Since 75 to 80 per cent of the total gas made is produced in the generator, the importance of maintaining the correct temperature there is evident.

In the usual operation of a set, with bituminous coals as generator fuel, the tendency for unequal distribution of heat in the set must be checked by one of the three following methods: (1) Burning some of

the blast gas outside the set at the stack; (2) neutralizing some of the excess heat produced in checker chambers, by admitting steam or excess air into the carburetor during the blast; or (3) using an excess of steam during the run without prolonging the blasting time.

Method 1 provides a means for maintaining the desired temperatures throughout the water-gas set, but does so wastefully, since the heat evolved by burning the gas outside the set is seldom utilized. With this method, the loss due to the heat wasted at the stack may be expected to increase as the per cent of volatile matter in the coal increases.

Method 2 is not commonly practiced but offers a possible means of accomplishing the same results as method 1, with similar inefficiency. However, when sufficient air pressure is available to use an excess of air to control the temperatures in the checker chambers, one possible advantage is gained. The carbon deposited in these chambers may be thus burned out or the accumulation of carbon prevented. The cause of excessive deposits of carbon has been mentioned on page 22. See also page 72.

Method 3, which is commonly employed in Illinois, frequently in combination with method 1, results in a high fuel and steam consumption and a considerably decreased output per unit of time. Such an excess of steam consumption during the steam run decreases the fuel temperature unduly and this in turn means that, with the same blast conditions, the blast gas produced will contain more CO_2 and less combustible gases. Thus, on blasting, the complete combustion reaction, $\text{C} + \text{O}_2 = \text{CO}_2$, takes place, but the reaction, $\text{CO}_2 + \text{C} = 2\text{CO}$, discussed on page 8, does not take place to an appreciable extent until later in the blast, when the temperature in the fuel bed is sufficiently high. Method 3 is probably more efficient than operating with normal steam and reduced blasting time, although the desired results have not been obtained from it.

Figure 6 shows the effect on the composition of the blast gases of an excess of steam during the steam run or, what amounts to practically the same thing, a deficiency in the quantity of blast. The first curve shows the change in per cent of CO_2 as the blast progresses, when 40 pounds of steam per minute is used on each steam run. The third curve shows similarly the relative amount of CO_2 in the blast gas at different stages of the blast, operating with the same cycle and using approximately the same amount of air as in the operation represented by the first curve, but with less steam during the run. The per cent of CO present in the blast gas under both of these conditions is shown by the second and fourth curves. The slope of these curves will vary according to the height of the fuel, the condition of the fire, the kind of fuel, the amount of green fuel present, and the way the carburetor is blasted. When opened too wide, the carburetor blast

may rob the generator of needed air blast; then the CO_2 curve would not drop so rapidly. These curves are sufficiently representative to show that an excess of steam during the run cools the generator to such a degree that a smaller amount of combustible gas and more CO_2 are produced during the blast.

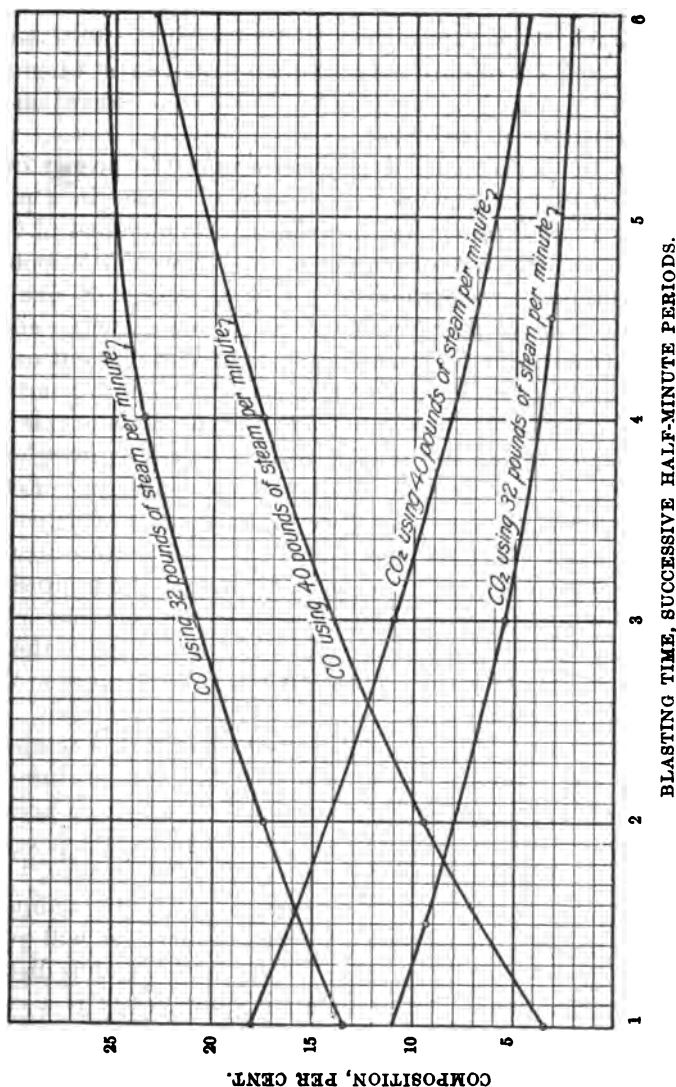


FIGURE 6.—Effect of change in quantity of steam on composition of blast gases. Cycle—3-minute blast and 4-minute run.

It is apparent that by operating according to method 3 a cycle can be found that will result in checker chambers being heated to the desired temperatures only. A combination of methods 3 and 1, wherein some of the blast gas is burned at the stack, permits the same results to be accomplished with a higher temperature in the fuel bed of the generator.

This change in operation may require a change in the operating cycle. Although it is possible to increase the quantity of steam per run in many plants by merely opening the steam valve wider, it may be found necessary to increase the length of the steam run in some plants if the steam supply is insufficient. For example, a plant employing a 3-minute blast and a 3-minute run with coke fuel may find it necessary to operate with a 2-minute blast and a 3-minute run, or even with a 3-minute blast and a 4-minute run when changing to coal. This can be determined only by a consideration of the conditions prevailing in each case.

The factors to be considered in determining the proper cycle of operation with high-carbon fuel have been discussed in a previous section, when it was stated that to make 1,000 cubic feet of finished gas about 1,500 cubic feet of air during the blow and 32 pounds of steam during the run were required. The advantage of making the blow and the run as short as possible consistent with adequate time of contact between the gases and the fuel and without using too large a proportion of the time for changing valves, was pointed out. In changing to bituminous fuel, a readjustment of the cycle is necessary in order to maintain the temperature balance throughout the set, on account of the increased value of the blast gas. Let it be assumed that for a given set the most favorable operating condition with coke fuel is a 4-minute blow followed by a 4-minute steam run, and that the same cycle with the same blast and steam pressures is continued with a high-volatile coal fuel. If the coal has a coking tendency, one of the first results noted will be a matting of the fuel in the top part of the generator, causing increased resistance in the fuel bed and resulting in the passage of less air through the fire. Under these circumstances, the coal is not heated to so high a temperature at the end of the 4-minute blow, although the temperature in the carburetor and superheater will remain fairly constant on account of the richer quality of the blast gas. The make of gas per unit of time during the run will decrease by perhaps 20 to 30 per cent. Should the blast pressure be increased in an attempt to raise the temperature of the generator fuel, the increased volume of blast gas will overheat the checker chambers if all the gas is burned in them, thus disturbing oil-cracking conditions. The usual method employed is a compromise. It is desirable to reach a condition where the volume of generator blast gases is reduced sufficiently to prevent the carburetor and superheater from becoming overheated, and at the same time to prevent undue lowering of the average temperature in the generator fuel. To do this, it is customary to increase the quantity of steam used per minute during the run, or when this can

not be done, to lengthen the steam run. At the same time, the blasting period and the air pressure are unchanged. The result is that the generator fuel does not have so high a temperature at the end of the run. On blasting, the fuel will not now generate so much CO, since complete combustion takes place to a greater extent in the generator with the formation of CO₂. More heat is being stored in the generator per minute of blast in this case, a fact that is readily understood when the heat reactions of carbon and oxygen are considered.

One pound of carbon burning to CO, liberates 14,544 B.t.u., while 1 pound of carbon burning to CO liberates only 4,350 B.t.u. However, the same average temperature of generator fuel is not reached at the end of the blow because the initial temperature was considerably lower. This results in the production of less blue gas during the steam run, containing a higher per cent of CO₂. Since with this method less CO for heating the checker chambers is formed during the blast, these chambers are not usually overheated. Should the temperature still be a little high, and it is not desirable to reduce it by cooling the generator further with a greater excess of steam, it may be lowered by overblasting the carburetor (using an excess of secondary air).

DETERMINATION OF OPERATING CYCLE.

Conditions in different plants will affect the selection of a favorable cycle for coal fuel. A method of operating has been devised by the authors making use of the so-termed "blow run," which involves other considerations in determining the operating cycle and which is discussed later in this bulletin on page 62. The following suggestions are, however, offered for guidance in determining the cycle to be employed when operating a water-gas set with the usual methods.

1. It is usually advantageous to use as much air as possible per minute of blow.

2. When a run and a blow of about equal length are used, enough steam should be used during the run, in order that no appreciable amount of generator blast gas in excess of what is required to heat the checkerwork properly will be produced during the blow.

3. When sufficient steam is not available to accomplish this, with the steam run the same length as the blow, the duration of the steam run should be increased until the desired result is obtained.

4. The use of steam in excess of what is required to accomplish the above should be avoided.

5. The CO₂ content in the blue gas should be reduced as much as possible, consistent with burning all of the generator blast gas in the checker chambers.

COAL FUEL IN THE GENERATOR.

ARCHING AND CAKING.

Coke in the generator presents a porous loose mass to the passage of air and gases through the fire, while on the other hand most freshly charged bituminous coals tend to cake and run together. This caking, if excessive, is evidenced by increased blast pressure under the grate, other conditions remaining the same; and, if the set is equipped with an air meter in the generator blast line, a decided decrease in the amount of air going through the fire will be indicated soon after a fresh coaling is made. This is especially true when excessively heavy charges are made. It is frequently customary in plants operating with coke to make one or more fuel charges two or three times the size of the normal running charge at the beginning of the day's run. With ordinary bituminous coal this procedure would almost certainly choke the generator. In general, it may be said that large egg-size coal, even with the caking controlled, offers more resistance to the passage of air than does coke, and the addition of a large charge of green coal causes an excessive increase in this resistance. When the fuel cakes, it forms a nearly impenetrable arch over the fire, which hinders the passage of air, slows up the combustion during the blast period, decreases the temperature of the fuel bed, and results in a great decrease in the amount of gas produced during the run. This arched mass of fuel frequently remains in place where formed, even after the already coked fuel beneath it has burned away, leaving a hollow space between. The arch may be so firmly established that it is necessary to break it up with bars before another coal charge can be added. This, of course, is additional labor, even if it were not an indication of an inefficient operating condition. Methods of overcoming these difficulties are discussed later under the Streator experiments (see p. 47).

It was formerly believed by some operators that when bituminous coal was used as generator fuel, practically all of the volatile matter would be driven off from a fresh charge during the first blast after coaling. On the contrary, it is found that the surface of the fuel bed remains fairly cool from charge to charge. The surface of a charge has not been entirely converted into coke when the next charge is added. The rate of conversion into coke will of course depend upon the temperature in the generator, which in turn is dependent upon the amount of air passed through the fire during the blast period and the amount of steam used during the run. On one occasion, a 6-inch lump of coal removed from the fire just before a fresh charge was made showed about three-fourths inch of coke on those faces perpendicular to the laminations of the coal, while the other two faces had coked to a depth of about one-fourth inch.

The fact that the surface of the fuel usually remains relatively cool may delay the ignition of the generator gases when the lid is opened for charging. These gases are usually combustible and a continuously burning pilot light or some form of torch kept ready to ignite them will avoid any unnecessary delay caused by their failure to ignite when the lid is opened. As with coke fuel, care should be taken that the gases are ignited before attempting to charge.

CONTROL OF CLINKER.

The difficulty experienced from clinker in plants using bituminous coal varies with the kind of coal and with the operating conditions employed. In plants where comparatively low generator temperatures are carried, the fusing point of the ash may not always be reached, with the result that no clinker is formed but instead a bed of loose ashes. In other plants where more air blast is used in the effort to maintain a higher fuel temperature, a very hard clinker may be formed.

In general, the ash of Central District coals fuses at a lower temperature than that from some of the better eastern cokes; therefore special attention must usually be given to operating methods when these coals are used.

Owing to the greater friability of some of these coals, considerable fine material is almost inevitably introduced into the generator when charging. If no fuel spreader is used or the one in use is inefficient, most of the fine coal is deposited in the center of the fuel bed, increasing the thickness and resistance to draft of that part of the fire. This, with the coking tendency of the fuel, causes most of the air blast to pass up around the generator wall, giving a more intense combustion there than in the middle of the fire and resulting in a fusion of the ash and the formation of clinker or "edgings" on the wall. If this clinker is high up on the wall, barring down from above may be necessary, and in any case the deterioration of the lining bricks is hastened.

The difficulty can be largely overcome by the use of an efficient spreader which will deflect most of the coal to the outer edge of the fuel bed and by split runs or a greater percentage of down runs, which will usually result in keeping the clinker formation nearer the grate, where it can be more easily handled. Figure 7 illustrates a type of fuel spreader that is very efficient when properly designed for the generator with which it is to be used. By cleaning the fire at specified intervals the accumulation of clinker can be regulated to such a thickness that it can be readily removed.

PREVENTION OF STICKING OF VALVES BY TAR DEPOSITS.

The gases leaving the generator when bituminous coal is used consist not only of carbon monoxide, hydrogen, and carbon dioxide, the usual constituents of water gas, but of varying amounts of hydrocarbons. Some of these are permanent gases, but tarry vapors are also present that are partly distilled in their passage through the set

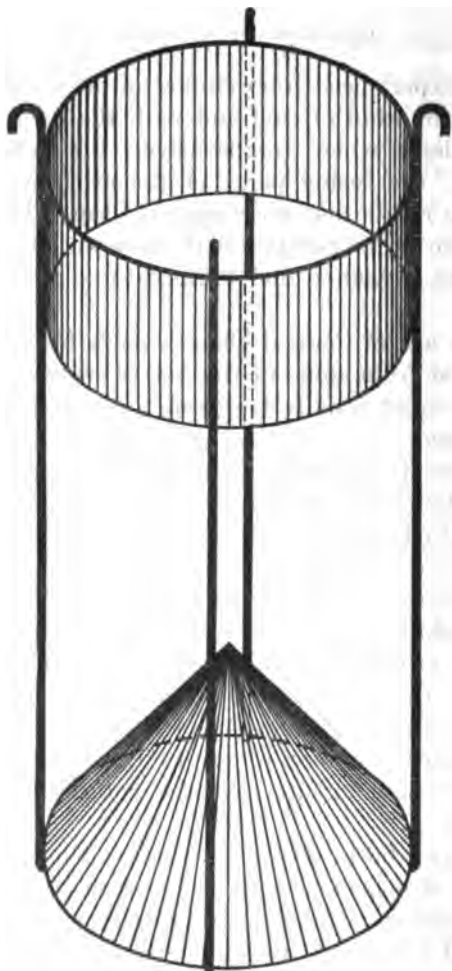


FIGURE 7.—Fuel spreader.

and if condensed deposit as a sticky viscous pitch. This pitch is in evidence at any part of the machine where there is a slight leak, as, for example, at the hot valve and the carburetor and superheater blast valves, around the oil-spray stuffing box, and occasionally at the generator lid. The pitch is fairly fluid when hot, but when cool is tough and viscous and is likely to cause valves to stick very tightly.

This need give little trouble if preventive measures are taken. Usually a fairly fluid mixture of lubricating oil and flake graphite, smeared occasionally on the valve stems, will prevent the sticking. In general, troubles of this sort are reduced to a minimum when proper temperatures are maintained throughout the set and the weight of charges is not excessive. In plants operating with but short lay-overs no trouble is experienced.

REDUCED CAPACITY WITH COAL FUEL, WITH THE USUAL OPERATING METHODS.

The chief difficulty encountered by gas companies, especially the large ones, when they use bituminous coal instead of coke for generator fuel is decreased capacity. Various operators have reported a decrease of 25 or 30 per cent in the possible output of the plant when coal is substituted for coke. One reason for this decrease seems to be the difference in resistance of the two fuels to the passage of air under the same blast conditions. The rate of combustion of the fuel, and therefore the rate at which the fuel acquires a gas-making temperature, seems to be roughly proportional to the amount of air passing through the fire in a given time. If the volume of air passing is augmented by increasing the air pressure, the combustion is more rapid, and the fuel bed is brought more quickly to the proper temperature for decomposing steam. In a majority of gas plants the blowing capacity is limited, and the initial pressure at the base of the generator is conditioned by the type of blower used. Consequently, since the coal fuel bed offers more resistance to the passage of air through the fire than does coke, more air is transmitted at the pressure available with a coke fire than with coal, the combustion is more rapid, and during the run the gas production is greater with coke. If it were possible to force the same volume of air through the coal fire in a given time, it seems likely that the volume of gas produced during a given time would more nearly approach that with coke.

Under usual operating conditions, therefore, a blow period takes longer in proportion to the length of steam run to bring the fuel up to the same working temperature with coal than with coke. On account of the greater amount of volatile matter in the coal, the amount of combustible gas given off during the relatively longer blow period is more than sufficient to maintain the requisite temperatures in the carburetor and superheater; consequently a blast of such length as will heat the generator suitably overheats the other chambers (if the gas is entirely burned in them) or necessitates the burning of considerable gas at the stack. Either condition is undesirable and wasteful. It may be found impossible to burn this excessive production of combustible generator blast gases in the set

even with the carburetor blast valve wide open, unless the volume of these gases is decreased by partly closing the generator blast valve, or the fuel bed is unduly cooled by excessive use of steam during the steam run. In either case, however, the generator will not attain the desired temperature so quickly. In some plants where there is an excess blowing capacity it may be advisable to cool the carburetor and superheater by overblowing them rather than by underblowing and oversteaming the generator. With any of these methods, however, capacity can only be obtained by sacrificing a considerable amount of combustible gas with a resultant waste of fuel.

FUEL AND OIL ECONOMIES.

Potential fuel and oil economies will vary in different plants, not only with the operating conditions employed, but also with the composition and quality of the coal and oil used, the daily operating time, and the prevailing weather conditions; hence, operating results can only be compared when operating conditions are known. Table 7 is a summary of results obtained with various fuels and oils in a few plants. The operating conditions are given when known. The data given for plant E in this table were obtained by the authors at Streator, Ill., with coal fuel, by methods similar to those employed in the other plants. These results agree fairly well with others reported, although small differences are due to seasonal variations, the Streator data being obtained in the late fall, the remainder in summer. It will be noted that the oil used is from one-fifth to one-half gallon less per 1,000 cubic feet of gas when made with coal fuel than when made with coke. The generator fuel is increased about 30 per cent when coal is used, and the per cent of ash and moisture in the coal is not excessive. However, the lower cost per ton of coal as compared with coke usually much more than offsets the difference in fuel consumption per 1,000 cubic feet of gas.

TABLE 7.—Practice at five water-gas plants.

| Details of practice. | Equipment. | | | | | |
|--|--|--|--|---|--|--|
| | 2 6-ft. United Gas Improvement Co. sets. Generator, 4 ft. internal diameter. | 1 8-ft. Western Gas Construction Co. set. Generator, 6 ft. internal diameter. ^a | 2 8 ft. 6 in. United Gas Improvement Co. set. Generator, 6 ft. 6 in. internal diameter. ^a | 1 8 ft. 6 in. Gas Machinery Co. set. Generator, 6 ft. 6 in. internal diameter. ^a | 1 6-ft. United Gas Improvement Co. set. Generator, 4 ft. internal diameter. ^a | |
| | Plant A. | | Plant B. | Plant C. | Plant D. | Plant E. |
| Kind of fuel..... | Indianapolis egg-size oven coke. | Franklin County, Ill., lump coal. | No. 4 seam Indiana lump coal. | Franklin County, Ill., egg coal. | Jackson County, Ill., lump coal. | Williamson County, Ill., large egg coal. |
| Generator fuel per charge.....lb. | 500..... | 750..... | 1,000-1,100... | 1,100..... | 1,600-1,800... | 659. |
| Frequency of charging generator..... | 30 min..... | 35 min..... | Every 5 runs | Every 5 or 6 runs. | Every 8 or 9 runs. | Every 5 runs. |
| Air per minute.....cu. ft. | 1,800..... | 1,027..... | 3,200 (?)..... | No meter... | No record... | 1,200. |
| Steam per minute: | | | | | | |
| Up run.....lbs. | 22..... | 28..... | 55..... | 40..... | 60..... | 34. |
| Down run.....do. | 27..... | 33..... | | | 45..... | 32. |
| Running time.....hr. per day.. | 14.6..... | 18.3..... | 14.4..... | 22..... | 18-20..... | 7.33. |
| Cycle: | | | | | | |
| Blow.....min. | 3..... | 3..... | 2..... | 3..... | 2..... | 3. |
| Steam run.....min. | 5..... | 5..... | 3..... | 4..... | 3..... | 4. |
| Split runs.....do..... | 2 up, 2 down, 1 up. | 2 up, 2 down, 1 up. | $\frac{1}{2}$ min. up, $\frac{1}{2}$ min. down. | No split runs | No split runs | 2 up runs to 1 down run. |
| Kind of gas-oil used. | 32 to 36° B.. | 32 to 36° B.. | | 28 to 30° B.. | 28 to 30° B.. | 28 to 30° B. Texas. |
| Gas made per run.....cu. ft. | 3,633..... | 2,900..... | 3,800..... | 4,550..... | 4,400..... | 2,831. |
| Gas made per hour.....cu. ft. | 27,245..... | 21,750..... | 41,800..... | 38,700..... | 52,800..... | 23,468. |
| Generator fuel per 1,000 cu. ft. of gas.....lbs. | 32.75..... | 44.5..... | 42.5..... | 40..... | 42..... | 46. |
| Oil per 1,000 cu. ft. gas.....gal. | 3.54..... | 3.13..... | 2.68..... | 3.2..... | 3..... | 2.97. |
| Heating value of gas.....B.t.u. | 600..... | 600..... | 575..... | 565..... | 588..... | 578. |
| Temperature bottom superheater.....°F. | 1,300 to 1,400. | 1,300 to 1,400. | 1,240 to 1,280. | 1,300..... | 1,300 to 1,400. | 1,280. |
| Cleaning time..... | 2 hrs. per day. | 2 hrs. per day. | 1 hr. for 2 cleans. | 2 hrs. per day. | 2 to 2 $\frac{1}{2}$ hrs. per day. | $\frac{1}{2}$ hr. per day. |
| Gas per min. per sq. ft. of grate area.....cu. ft. | 58..... | 46..... | 45..... | 40..... | 44.4..... | 56.5. |
| Steam per 1,000 cu. ft. gas.....lbs. | 33.7..... | 52.6..... | 43..... | 30.2..... | 36..... | 42.3. |
| Air per 1,000 cu. ft. gas.....cu. ft. | 1,480..... | 1,060..... | 1,680 (?)..... | No meter... | | 1,272. |
| Blast pressure.....in. of water.. | 18 to 20..... | | 16..... | 16 to 18..... | 20 to 22..... | 17 to 18. |

^a Data on practice with coke fuel not available for publication.

SELECTION OF STREATOR PLANT FOR EXPERIMENTS.

Since water-gas operation involves so many variable conditions, it was decided that satisfactory experiments with bituminous coal could be conducted only on a commercial scale. With the realization that some experiments might decrease production capacity considerably before an efficient operating method was evolved, it

was deemed inadvisable to select for experimental work a plant that was working at capacity, since operation for a few hours under a trial set of conditions might jeopardize the gas supply to the community being served without proving conclusively the source of the difficulties.

The Public Service Co. of Northern Illinois offered the facilities of its Streator plant for tests; this offer was accepted because the plant is well equipped and has a generating capacity several times as great as any demand likely to be placed upon it. Moreover, it is relatively small, and experiments with various coals, some of which might prove to be entirely unsuitable for water-gas manufacture, would not involve the purchase of so much worthless material as would be required at a larger plant.

EQUIPMENT OF PLANT.

The generating equipment of the Streator plant consists of two 6-foot water-gas sets, one of United Gas Improvement Co. construction, the other built by the Western Gas Construction Co. These sets are used alternately, one being operated while the other is down for rechecking or repairs. The sets stand side by side in the same generator house, and the steam and blast supply lines to the sets, as well as the collecting mains from them, are interconnected. The blast is supplied by one of two No. 5 turbine-driven blowers connected by elbows into the ends of a sheet-iron T. The side outlet from this T passes through a brick wall to the generator house where the branches of a Y connection carry the blast to the two sets. A sheet-iron gate in the branch to the Western Gas Construction Co. set makes possible the shutting off of this branch when the other set is in use. There is no such provision, however, in the other branch.

Each set has the usual pressure gages, wash box, and scrubber, and one water-cooled primary condenser is used in common by both sets. In addition, the United Gas Improvement Co. set is equipped with steam and air flow meters. There is also an indicating pyrometer which can be used with either set, consisting of an indicator and two fire ends, one inserted near the middle of the carburetor and the other near the the top of the superheater. No means are available for recirculating wash water from the tar separators. Hot water from the condenser is used in the Western Gas set when it is in operation, but, owing to a difference in the elevation of the wash boxes and the lack of a suitable pump, cold fresh water only can be used in the other set. The gas, after leaving the primary condenser, passes to two relief holders, one of 30,000 cubic feet, the other of 25,000 cubic feet capacity. These holders are usually employed together to diminish the back pressure on the set. The

outlets of these holders lead to an exhauster. A by-pass also permits connection of the exhauster with the inlet of the relief holders. On account of the small size of the holder connections, this by-pass is used but a part of the time, restricting the fullest utilization of the holders as condensers. The gas from the exhauster passes through a water-cooled secondary condensor, thence to an outdoor rectangular shavings scrubber, from which it passes through two cylindrical outdoor purifiers connected in series by a center seal. These purifiers are so arranged that the gas can enter at the middle and leave by the top and bottom, or vice versa. After the purified gas is measured by a 6-foot station meter, it passes into a 100,000 cubic foot storage holder, then through a station governor to the city. The gas is tested at the gas office, about 1 mile east of the plant.

The normal daily output of the plant during the experiments was about 165,000 cubic feet. Since the capacity of either set was several times greater than the daily output, it was seldom necessary to operate more than 6 or 7 hours a day, and local conditions made even this short operation intermittent. Therefore, while the experiments served to point out operating methods and to solve some of the problems arising from the use of coal fuel, as well as to indicate the relative value of various available coals, they could not show conclusively what fuel and oil efficiencies and production capacities might be realized from continuous operation in larger plants. Gas engineers generally admit that better fuel efficiencies can be obtained with continuous than with intermittent operation. Since a considerable portion of the operating day elapsed before the carburetor and superheater of the set were in condition to fix the oil properly, it would seem that better oil efficiencies could be realized at a plant operating full time. The application of the Streator methods to conditions in other plants will be discussed in a later section of this report. The Streator experiments point the way to additional investigations and suggest solutions to some troublesome problems.

PREPARATION FOR TESTING BY CHECKING APPARATUS.

Before the experiments were begun, the apparatus to be used in determining the operating conditions was tested and calibrated in so far as facilities permitted. The gages of the steam meters were calibrated by checking the reference points of each against a standard pressure gage. The zero point was determined when the hands were so set as to show correct readings in the working part of the scale. The pyrometers were checked by comparison with a calibrated pyrometer. The fuel was forked from the storage pile into tared wheelbarrows and weighed on platform scales. The oil used was measured by a standard oil meter, which was calibrated against oil-tank measurements over a period of several weeks. It was found

that on account of variations in temperature and the difficulty of taking sufficiently accurate measurements, the daily oil consumption could not be determined accurately from daily tank measurements. Meter registrations with the factor determined were therefore used. When gas measurements were being made, it was assumed that the station meter, which had been overhauled and tested within the past two or three years, was substantially correct. Care was taken that the water was kept at the proper operating level.

GAS-ANALYSIS APPARATUS AND METHODS.

Throughout the experiments, frequent analyses of blast, blue, and finished gases were made to guide experimental procedure and to determine the effect of various changes in operation. The gas-analysis apparatus was of a modified Orsat type, consisting of a water-jacketed burette with a three-way cock connected to a glass manifold, from which branches led to the various pipettes through glass stopcocks. Carbon dioxide, oxygen, illuminants, and carbon monoxide were absorbed in solutions of potassium hydroxide, potassium pyrogallate, bromine, and acid cuprous chloride, respectively. Two pipettes were used for the absorption of carbon monoxide, one being so arranged that the gas could be bubbled through the solution. The other absorption pipettes contained glass tubes to give more absorption surface. Absorption was facilitated by passing the gas back and forth from the burette to the pipette several times until the volume ceased to contract. The copper chloride solutions, frequently renewed, were kept in the cuprous condition by pieces of copper gauze. The methane and hydrogen in the gases analyzed were determined by combustion of the residual gas from the absorptions with pure oxygen in a special pipette over gas-saturated water. The usual procedure was to pass a measured volume of oxygen into the pipette, then to pass a measured volume of gas into the pipette very slowly over a spiral of platinum wire heated by an electric current. Usually the entire gas sample remaining after the absorptions was burned. The current was then interrupted and the burned gas allowed to cool and to return to the burette for measurement. The methane and hydrogen were computed in the usual manner from the contraction and subsequent absorption in potash. Even the leanest blast gases could be analyzed without difficulty by this method.

Hydrogen sulphide in the unpurified gas was determined at frequent intervals by the Tutweiler apparatus, using an iodine solution so made that 1 cubic centimeter was equivalent to 100 grains of H_2S per cubic foot of gas. Daily stain tests were made with lead-acetate paper at the office where the heating value was taken. Total sulphur determinations were made with a Drehschmidt total-sulphur apparatus, in which the products of combustion from a metered volume of gas

were bubbled through a solution of sodium carbonate to which a few drops of bromine had been added. The solution was divided between two wash bottles of a type that gives very intimate contact of gas with solutions. The gas was burned at the rate of about one-half cubic foot per hour. After about 2 cubic feet of gas (corrected) had been burned, the solution was washed from the apparatus into a beaker, acidulated with hydrochloric acid, and boiled to expel carbon dioxide. The sulphur was precipitated by barium chloride in the usual way and the precipitate of barium sulphate was removed on an ashless filter, ignited, weighed, and the amount of sulphur calculated in grains per 100 cubic feet (corrected) of gas burned.

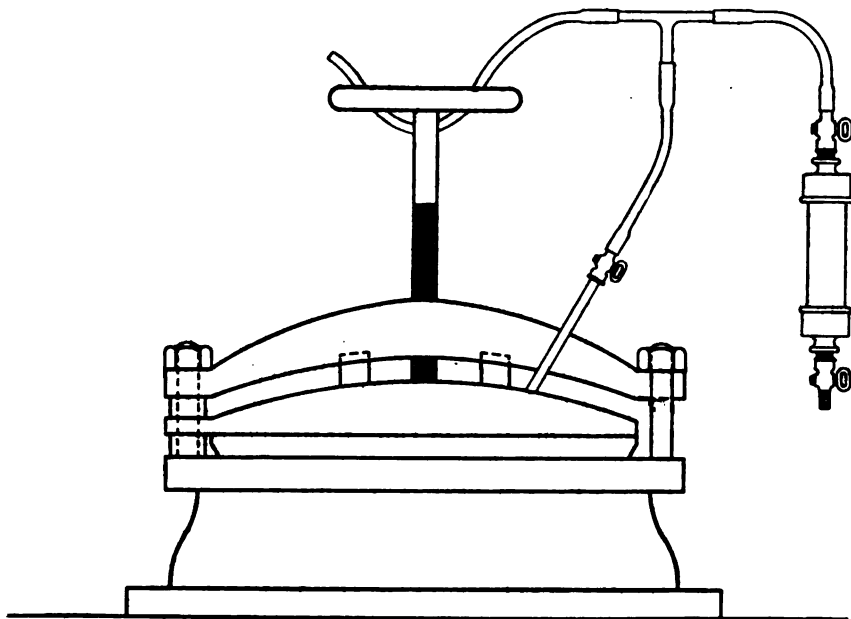


FIGURE 8.—Apparatus for sampling gas at generator lid.

SAMPLING.

Samples of gas were collected for analysis in sampling tubes made from 12-inch sections of 2-inch pipe fitted at each end with reducing couplings and ordinary gas hose cocks. Before a sample was taken, the sampling tube was filled with water which had been previously saturated with the gas to be analyzed. In taking samples at the stack or generator lid over a very short time, as, for example, during a fraction of a minute, the arrangement shown in Figure 8 was employed. Since the gas was under pressure, a continuous purging of the connecting tube through the tee and the side outlet connection was assured, and the uniformity of the sample collected was obtained by making the outflow of water from the sampling tube as uniform as

possible. By having several such tubes ready it was possible to collect more than one sample during a single run or blow, and from the analyses to plot curves showing the changing composition of the gases.

The coal was sampled by laying aside at regular intervals a forkful of the coal that was being loaded into the charging buggy. This composite sample of forked coal was prepared for analysis by crushing and quartering it repeatedly according to the standard Bureau of Mines method.

EXPERIMENTAL PROCEDURE.

In beginning the Streator experiments the plan adopted was to operate the plant in a commercial way; that is, to produce the needed amount of gas of the requisite quality, and yet to manipulate operating conditions so that the effects of some of the more important variables on the problems involved might be studied. The primary object of the experiments was to work out a satisfactory operating method which would improve the economy and capacity of the set and decrease the operating difficulties, rather than to make a more academic study of the principles involved. At the same time care was exercised not to upset the routine of the plant operators any more than was absolutely necessary.

At first it was considered inadvisable to make any radical changes in the operating conditions. Accordingly the experiments began with coke fuel. Later coal and coke mixtures were used, the per cent of coal being increased from time to time until 100 per cent coal was used. Changes in operating conditions were made one by one. The investigation was not limited to the solution of certain defined problems; on the contrary, an effort was made to ascertain the combination of operating methods that would give the best general results. The following specific problems, most of which were closely inter-related, will be discussed: Obtaining the cooperation of gas makers; the use of coal and coke mixtures; selection of coal; control of the arching and caking of coal in the generator; reduction of capacity with coal fuel, employing the usual operating methods; the air purge; smoke prevention; and control of clinker.

The solution of these problems, with the exception of the first, which is psychological, necessitates the study of the effects of the several variables. Only those that seemed to exert the greatest effect and that could be studied without greatly disturbing the routine of operation or the service being rendered by the gas company to the community were studied with a view to determining their influence. Among the more important phases of operation studied were: Coal and coke mixtures, depth of the fuel bed, weight and frequency of charges, rate of blasting and length of blasting period,

rate of steaming and length of steaming time, proportioning of up and down runs, and behavior of various coals.

The procedure followed was not exactly in the order here outlined. For example, in the study of various mixtures of fuel several days were spent on a mixture of certain proportions, and the effects of various blasting and steaming conditions, the weight of the charges, and other conditions were studied for the mixture.

The same procedure was repeated with various mixtures. The variables studied with a given mixture were those that seemed to be of the greatest importance in a given case. The procedure from day to day was governed by the results obtained with a given set of conditions, the analyses of gases produced in various stages of the operation, and other observations. After experimental determination of the conditions that seemed to give the best results with a given mixture of fuels, a run of several days was usually made with the conditions held as constant as possible to ascertain whether the results were maintained or were disturbed by other factors not under control. Throughout the experiments it was necessary to contend with certain deficiencies in plant equipment and with incidents in operation which introduced changes; therefore the only way to draw conclusions is to consider several days' results in perspective and to make allowances for known occurrences which affected the results on certain days.

Although the actual operating results largely determined the operating procedure from day to day, gas analyses gave considerable assistance in studying the effects of changes in conditions. Arrangements were so made that samples of gas could be collected at the generator lid, from the down-run connection between the base of the generator and the carburetor, and at the stack. Frequent analyses were made of gases collected at these points. It was possible to observe the variation in composition of the generator blast gas or the blue gas from minute to minute during a given blow or run, or to compare the average composition of the gases of successive blows or runs. In this way, the effects of changing temperature in the fuel bed during a given part of the cycle or the effect of changes in depth of fuel over several runs could be studied. Moreover, the completeness of combustion of blast gases in the set and the amount of excess secondary air could be ascertained from analyses of gas samples taken at the stack. Analyses of the finished gas gave additional information relative to blue-gas and oil-gas production and suggested changes in operating conditions.

COOPERATION OF GAS MAKERS.

Securing the cooperation of the gas makers presented little difficulty in the Streater work, since the plant had been operating with

coal for several weeks prior to the tests and the force had some familiarity with operating conditions for this fuel. In some plants, however, the opposition of the working force has been one of the chief difficulties encountered.

The uncertainty of the results to be obtained with an unfamiliar fuel and the lack of a money incentive for testing new and untried operating methods that may lead to difficulties or inconveniences, are sufficient to prejudice some gas makers, even before a trial is made.

The superintendent who is about to try a new fuel, especially one that is so different from ordinary practice as bituminous coal, must fit his method of introducing the innovation to the temperament of his operators, realizing that for good results the cooperation of the gas makers must be obtained. An experienced gas maker senses every change in the behavior of his machine and this peculiar understanding can be utilized to advantage if the more visible conditions are borne in mind at the same time. This paper does not propose to tell the superintendent how to handle his men, but merely suggests how essential to success is their cooperation.

USE OF COAL AND COKE MIXTURES.

When the Streater experiments were begun, it was thought best to start with a fuel of known properties and work gradually to the unknown fuel; therefore at the start a run of about one week was made with retort coke of the type that had been used in the Streater plant for some months, the operating conditions being fairly well established. The coke that happened to be on hand was not of good quality; it was rather soft and gave an obstinate clinker unless an excess of steam was employed. The results obtained from this coke were not as good as those to be expected from a better grade and consequently are not given here as typical of coke fuel.

After the performance with plain coke had been studied for about one week, operation was commenced with a mixture of coal and coke. The coal first used was mined in Perry County, Ill.; its proximate analysis, sulphur content, and heating value are given in Table 8.

TABLE 8.—*Water-gas operation with mixtures of coal and coke as generator fuel.*

| | Set A. | | | | | Set B. | | Set C. |
|---|--------------------|-------------------|-------------------|-------------------|--------------------|-------------------|-------------------|--------------------|
| | 100 per cent coke. | 60 per cent coal. | 70 per cent coal. | 75 per cent coal. | 100 per cent coal. | 75 per cent coal. | 80 per cent coal. | 100 per cent coke. |
| Daily make..... | | | | | | | | |
|corrected cu. ft. | 184,000 | 194,000 | 197,000 | 181,000 | 174,000 | 195,000 | 172,000 | 279,500 |
| Make per hour.....do. | 23,000 | 21,500 | 23,200 | 22,200 | 21,200 | 26,000 | 23,400 | 25,400 |
| Make per run.....do. | 2,708 | 2,898 | 3,120 | 2,780 | 2,608 | 3,306 | 3,048 | 3,630 |
| Make per square foot grate area per running hour..... | 1,840 | 1,690 | 1,880 | 1,775 | 1,700 | 2,080 | 1,872 | 2,016 |
| B. t. u. of finished gas..... | 560 | 566 | 564 | 584 | 558 | 561 | 560 | 600 |
| Cycle: | | | | | | | | |
| Blow.....minutes.. | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 3 |
| Run.....do..... | 4 | 4 | 4 | 4 | 3½ | 4 | 4 | 5 |
| Blast pressure..... | | | | | | | | |
|inches of water. | 18.5 | 20.0 | 18.5 | 19.5 | 17.5 | 17.5 | 15 | 19 |
| Air per minute.....cu. ft.. | 1,750 | 1,650 | 1,550 | 1,560 | 1,170 | (?) | (?) | 1,890 |
| Air per 1,000 ft. of finished gas made.....cu. ft.. | 1,940 | 1,700 | 1,485 | 1,670 | 1,850 | (?) | (?) | 1,490 |
| Steam per minute: | | | | | | | | |
| Up runs.....lbs.. | 40 | 40 | 34 | 32 | 32 | (?) | (?) | 22.0 |
| Down runs.....do. | 36 | 38 | 37 | 35 | 34 | (?) | (?) | 27.1 |
| Steam per 1,000 ft. of gas made.....lbs.. | 42 | 55 | 45.3 | 48.2 | 41.1 | (?) | (?) | 33.0 |
| Generator fuel per 1,000 ft.....lbs.. | 44.5 | 49.0 | 45.6 | 47 | 52 | 47 | 46.5 | 36.7 |
| Oil per 1,000 ft..... | 2.94 | 2.78 | 2.55 | 2.76 | 2.79 | 2.51 | 2.66 | 3.58 |
| Up and down runs..... | 2 up.
1 down. | 2 up.
1 down. | 1 up.
1 down. | 1 up.
1 down. | 2 up.
1 down. | 1 up.
1 down. | 1 up.
1 down. | |
| Daily operating time.....hours | 8 | 9 | 8.5 | 8.15 | 8.12 | 7.5 | 7.35 | 11.0 |

NOTE.—Sets A and B were located in the Streater plant. Set B, not being equipped with steam and air meters, was operated experimentally only when set A was down for repairs. Set C, in another plant, was operated with coke fuel, and results from this set are given for comparison; the fuel used was oven coke. All these sets had the same internal diameter, 4 feet.

The fuel mixture first used consisted of 55 per cent coal and 45 per cent coke. The coal was mixed by loading the charging bucket with fuel from separate piles at the same time, account having been kept of the amount of each fuel weighed in during the day. Each day the percentage of coal was increased a trifle, and operating conditions were varied to give the best results obtainable with each mixture. After about six weeks of operation 100 per cent coal was used. Table 8 gives typical results obtained with various mixtures of coal and coke.

This table does not indicate any very marked differences which can definitely be attributed to the proportions of coal and coke in the various mixtures. Consideration of all of the results obtained with various mixtures under approximately uniform conditions indicates, however, that the percentage of coal in the mixture does affect the production capacity to a certain extent, and in a way somewhat different from what might be expected.

Figure 9 is a curve showing the gas production per square foot of grate area per hour, with various percentages of coal in the generator fuel mixture. It will be noted that mixtures containing over 55 per cent of coal attain the maximum rate of gas production with about 70 per cent coal, the make decreasing with higher or lower percentages of coal.

Adjacent points on the curve are not necessarily determined by data obtained on successive days. The fact that the percentages of coal were varied during this period, however, leads to the belief that there is a proportion most favorable for mixing fuels. The fuel and

oil efficiencies appear to be as favorable at this point as at any other, but the variation from day to day seems to be too much affected by other conditions to permit the plotting of a curve showing the effect of fuel proportions on these efficiencies.

CONTROL OF ARCHING AND CAKING IN THE GENERATOR.

Arching can be readily prevented by making charges of normal size with greater frequency during the early part of the day's run. At Streater it was the practice to make one run in the morning before coaling; this will be discussed later. A normal charge was added after the first run and another after the third run. This method was continued until the fire was up to its usual operating level—level with the bottom of the take-off connection—as with coke. This depth of the fire was maintained by charges of normal size 35 to 45 minutes apart. The weight of each charge was about the same as that of the normal charge when coke was used. The resistance to the passage of air through the fire was found to be not excessive with this method of charging, the fire burned down uniformly, and there was no arching; consequently barring down was

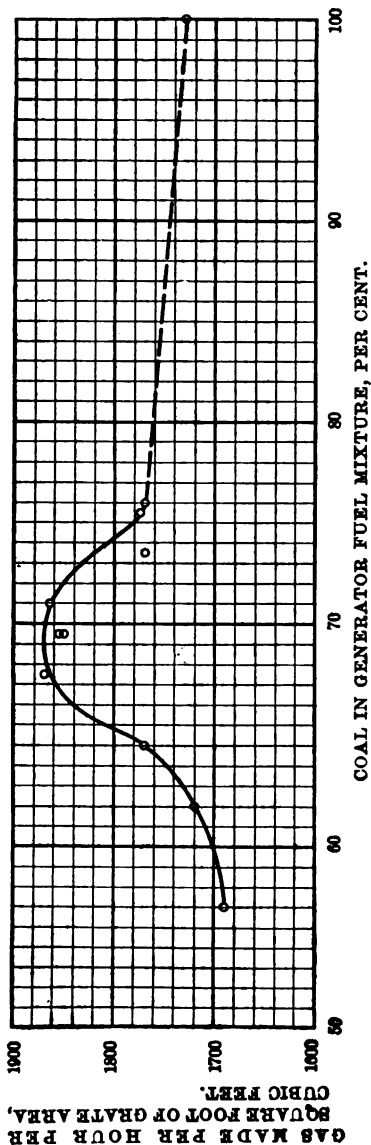


FIGURE 9.—Effect of percentage of coal in generator fuel mixture on capacity of set.

no longer necessary. When a coal with very strong coking properties is used and arching persists even under the above conditions, the difficulty can be remedied by making smaller but more frequent charges, although this necessitates opening the generator more fre-

quently and causes a certain loss of operating time. In many plants the loss of time during coaling is excessive and could be reduced considerably by standardizing the movements of the various men participating. It will usually be found, however, that the time required for coaling with Central District coals is no greater than with coke.

REDUCTION OF CAPACITY WITH COAL FUEL, WITH THE USUAL OPERATING METHODS.

Experiments with coal in the Streator plant during operating practice similar to that used with coke soon demonstrated that little increase in the production capacity was to be expected without a radical change in operating methods. The results obtained with coal at Streator, therefore, tended to confirm those from other plants, although the capacity reduction was not so great as had been reported by some of them. The average make per square foot of grate area in the Streator plant was 1,700 cubic feet per hour with coal fuel, whereas makes of from 2,000 to 2,200 cubic feet per hour could have been reasonably expected with coke of good quality. A reduction in capacity of 20 to 25 per cent was therefore indicated. To remedy the conditions that cause decreased capacity and disturb the heat balance in the set, the so-called "blow-run" method was devised. This method will be discussed on page 62.

AIR PURGE.

In most plants using coal fuel, the practice is to purge out the gas remaining in the set at the end of the steam run with blast gas. This is accomplished by opening the generator blast valve, after shutting off the steam, several seconds before the stack valve is opened. The generator blast gas formed displaces the residual gas in the set. That this procedure is sometimes advantageous is evident from a consideration of the composition of the gas remaining in the set at the end of the steam run. The following is a typical analysis of a generator gas produced with coal during the last minute of the steam run:

Analysis of generator gas, 319 B. t. u. per cubic foot.

| | Per cent. |
|-----------------------|-----------|
| CO ₂ | 12.2 |
| H ₂ | .2 |
| O ₂ | .0 |
| CO..... | 27.2 |
| CH ₄ | 5.2 |
| H ₂ | 53.2 |
| N ₂ | 2.0 |
| Total..... | 100.0 |

The gas leaving the set during the purge is usually richer than this, due to the presence of illuminants remaining from the cracking

of some residual oil that may not have been entirely decomposed during the earlier portion of the run.

This residual gas is worth saving if it can be purged out of the set by blast gas within a reasonable time and if care is taken that the purge is not so prolonged that not only the residual gas but some lean blast gas passes on to the holder. The importance of limiting the duration of the purge very carefully is evident from the composition of the gas formed in the generator at the beginning of the blow. The following is a typical analysis of blast gas 52.5 B. t. u. sampled at the generator lid at the beginning of the blow:

| <i>Analysis of generator gas, 52.5 B. t. u. per cubic foot.</i> | | Per cent. |
|---|--|-----------|
| CO ₂ | | 17.5 |
| O ₂ | | .5 |
| H ₂ | | .2 |
| CO..... | | 2.8 |
| H ₂ | | 4.0 |
| CH ₄ | | 2.5 |
| N ₂ | | 72.5 |
| Total..... | | 100.0 |

The mixture of only a small amount of this very lean gas with the finished gas in the holder would evidently offset any advantage that would be realized from saving the residual steam-run gas in the set. All operators do not understand this, as is evident from the fact that some are using air purges of such length that the amount of oil required per 1,000 cubic feet to enrich their blue gas is excessive. When the air purge is used, its duration should be carefully regulated so that the blast gas formed is just sufficient to purge out the good gas remaining in the set. The time needed for accomplishing this will vary in different plants according to the back pressure on the set and the blast pressure available. In fact, when the blast pressure is low and the back pressure is excessive, it may be impossible to purge the set by this method; if it is possible, it may take too much time to be advantageous. The correct length of air purge can usually be determined from the appearance of the flame at the stack when the stack valve is opened, the presence of yellow flame usually indicating that some fairly rich gas still remains in the set. At Streator an air purge of about 6 seconds was found to be satisfactory.

SMOKE PREVENTION.

The smoke given off by a gas works or other plant is naturally much more objectionable in some localities than in others. Some operators whose plants are situated in or near residence districts have feared the creation of a smoke nuisance from the use of bituminous coal as generator fuel more than any other objection raised to the use of such fuel. It is difficult to prevent the escape of smoke from

any gas plant at times regardless of the fuel used. The possibilities of smoke productions are somewhat greater with bituminous coal than with coke, but by arranging the operating conditions it is possible to reduce smoke to a minimum, if not to abolish it entirely.

In a water-gas set, as in boiler plants or other furnaces, smoke from coal is caused by incomplete combustion. If conditions are so adjusted that combustion is complete at all times no smoke will be produced. Smoke is most difficult to control when a set is put into operation after a long lay-over; it follows, therefore, that the amount of smoke to be controlled is likely to be greater in plants operating but a small portion of the day than in those running nearly full time. If a large amount of green coal is put in the generator and blasted after the fire is cleaned in the morning following a lay-over of several hours a large amount of gas is evolved. This gas would perhaps be combustible by itself, but it is diluted by a considerable amount of carbon dioxide formed during the early part of the blasting period by the combustion of the incandescent fuel below. Considerable steam is, moreover, produced by the decomposition of the green fuel and by the evaporation of the moisture in the coal. Thus, there exists the condition of a lean gas mixture, carrying tarry vapors, passing from the generator into the carburetor and the superheater, and thence up the stack. The generator gases are difficult to ignite at the very beginning of the blast even with coke fuel, but since they are colorless and carry no tarry vapors no smoke is produced.

The problem with coal fuel is, then, to bring the gases as soon as possible to such a requisite degree of richness that they will burn at the prevailing temperature in the checker chambers. This can be accomplished if the temperature of the generator fuel bed is maintained so high that a large part of the CO_2 formed in the zone of complete combustion is reduced to CO by the upper layers of the incandescent fuel. When a deep bed of incandescent coke is blasted, it soon produces combustible gases. Therefore the gas maker has two alternatives. He may start operation with coke, getting the fire hot and the generator gases lighted before charging any coal; or, if he desires to be entirely independent of coke, he can so arrange his operating cycle that considerable coked fuel from the previous day's run remains in the generator after the fire is cleaned. This latter necessitates retaining a sufficient fuel reserve from the preceding day on which to operate until the gases passing from the generator are rich enough to ignite.

The method adopted at Streater and found satisfactory was to take only one or two runs off the last coaling before a lay-over, and when no blow-run cycle was in use to blast the fire for 2 minutes before shutting down. No steam run followed the last blast in this case. By this method of operating, a bed of hot fuel was left beneath

the fresh charge, practically all of it being converted into coke by the next morning. The generator lid was left slightly ajar overnight, and sufficient air entered there to burn the volatile given off by the coal. This device helped to maintain a constant temperature in the carburetor and the superheater, but there was no overheating of these chambers. After the fire was cleaned in the morning, which usually did not take more than 20 to 40 minutes, a considerable bed of fuel remained for starting operation. This fuel was blasted about 10 minutes. Within 1 to 5 minutes the gases were combustible and could be lighted in the carburetor. After one run, a normal size charge of coal was made, and in the subsequent blast little or no difficulty was experienced in igniting the gases and burning all or nearly all of the smoke. After the start-up period, but little difficulty should be encountered in overcoming the smoke nuisance if operating conditions are normal; that is, (a) if the generator is kept hot enough to produce combustible gas during the blast; (b) if the checkerwork is hot enough to ignite the mixture of this gas with the secondary air, as supplied by carburetor blast; and (c) if the combustible gas and carburetor air are properly mixed and in correct proportions.

Some of these conditions, when lacking, may be attained by changing the operating procedure. For example, if the heats in the carburetor and the superheater are too low, an increased amount of generator blast or a decreased amount of steam used during the run are obvious remedies. When the mixing of air and blast gases in the carburetor and the superheater is poor, it sometimes may be remedied by a rearrangement of the checkerwork spacing, or by some device in the secondary air inlets to the carburetor and the superheater that will cause the incoming air to mix more intimately with the blast gas from the generator. Such cases must have individual attention, and no specific suggestion that will cover all conditions can be given.

CONTROL OF CLINKER.

In order to keep the temperature of the carburetor and the superheater within limits consistent with efficient practice, it was necessary to oversteam the generator during the Streator experiments when the set was operated with coal and the methods commonly used with coke fuel were employed. This tended to prevent the clinker from forming, or at least made it rather brittle. Very little difficulty was experienced from clinker during the experiments, particularly after a desirable operating cycle was formulated. In the early stages of the work some difficulty was encountered with side clinker—"edgings"—but this source of trouble was soon eliminated by increasing the per cent of down runs, whereupon the clinker fused and no longer clung to the wall, but collected on the grate, from which it was easily removed. Barring down clinkers through the charging door was no longer necessary.

After the blow-run method of operation was adopted, the clinker formed was somewhat harder but never gave as much trouble as that from the coke formerly used in the Streator plant.

SELECTION OF COAL.

When a Central District coal is selected as generator fuel, many of the requisites for other generator fuels hold. For example, the coal should not break up much on handling and it should be as free as possible from ash. The sulphur content is also a very important consideration. Analyses of the coals used in Streator indicate that at least one-fourth of the sulphur present in the coal passes into the blue gas. For example, a coal containing about 1.57 per cent S gave 155 grains of H_2S per 100 cubic feet of the blue gas. The Texas oil used in the carburetor increased this amount to 200 grains per 100 cubic feet in the unpurified carbureted gas. In general, it seems probable that any plant not equipped to purify raw gas containing more than 200 grains of H_2S per 100 cubic feet, at the average rate of purification, should use a coal with a sulphur content not exceeding 1.5 per cent in the delivered coal. Illinois State Geological Survey Bulletin 23 gives¹² a list of the mines of the Central District that could probably furnish coal complying with this condition.

The size of coal required for water-gas manufacture is also important, as with the other fuels. Since nearly all Central District coals show the coking property in varying degrees, however, it is not possible to prescribe one size for all coals.

The stronger the coking property of a coal the greater is the tendency for the lumps to mat in the fire, bringing about conditions similar to those existing when a coal containing considerable slack is used. In the experiments at Streator large egg size (4 to 6 inches) gave satisfactory results. Some coals broke up more readily than others, so that there was considerable slack. Although it is possible to use coal containing considerable slack under favorable conditions of blast pressure, such fuel tends to obstruct the passage of air through the fire. Careful forking and the avoidance of unnecessary breakage in handling are therefore recommended.

Certain Central District coals tend to disintegrate somewhat during storage, especially if exposed to the weather. Therefore it seems inadvisable to store coal for water-gas manufacture longer than is necessary to maintain an adequate stock. When circumstances necessitate the storage of such coal, the purchase of a larger size than otherwise required may be advisable to make due allowance for disintegration and breakage. The larger sizes are also less liable to spontaneous ignition.

¹² Cady, G. H., Mines producing low-sulphur coal: Bull. 23, Illinois State Geological Survey, Coop. Min. Ser., 1919, p. 9.

BLOW-RUN METHOD OF OPERATION.

Data obtained in tests showed that when coal is used as a fuel it is desirable to increase the capacity and efficiency of the generator by maintaining a higher temperature in the generator, by preventing excessive heating of the checker chambers by utilizing more of the heat stored in them, and by obtaining more of the volatile matter of the coal in the finished gas.

In order to accomplish these objects, the so-called blow-run method of operation was instituted. This was run as follows:

1. A blast was made of sufficient length (3 minutes, for example) to heat the carburetor and the superheater to the desired temperatures.

2. The superheater and carburetor blast valves were closed.

3. The stack valve was closed.

4. The generator blast was continued for a fraction of a minute (15 to 45 seconds), the combustible gas produced passing to the holder.

5. The generator blast valve was closed and a regular blast of suitable length (about $3\frac{1}{2}$ minutes) was made, oil being added to the carburetor in the customary manner.

6. After the completion of the run, the generator blast was stopped and a purge of suitable length (6 to 15 seconds) was made in the holder before opening the stack valve.

The adoption of this cycle presented no special difficulties in the generator plant. The gas maker had to work carefully at first to avoid operating the valves in the wrong sequence, but he soon became accustomed to the new procedure, and it was then performed as automatically as before. The effects of this operating method may well be discussed under the following heads: First, effect on generator conditions; second, effect on quantity and composition of finished gas; third, effect on carburetor and superheater conditions; fourth, effect on fuel and oil economy; fifth, effect on oil efficiency; and, sixth, effect on production capacity. Naturally these are to a great extent interrelated.

EFFECTS ON GENERATOR CONDITIONS.

The effects of the blow-run method on conditions in the generator are undoubtedly the most important, for although certain ultimate results in capacity, economy, and quality of production are sought for, the proper operation of the generator is the key to the entire situation. This is evident from the fact that from 75 to 80 per cent of the finished gas is made in the generator.

The addition of 30 seconds to the time of blasting the generator fuel affected results decidedly. So far as could be observed from

appearance and behavior of the set, the average temperature of the fuel bed was materially increased, resulting in a greater make of blue gas per minute during the actual steam run. The quality of the blue gas was also improved. The following analyses exemplify this change in composition.

TABLE 9.—*Typical analyses of blue gas produced with coal fuel, with and without the blow-run cycle.*

[B. t. u. per cubic foot (calculated): With blow run, 337; without blow run, 314.]

| | With blow
run. | Without
blow run. |
|--|-------------------|----------------------|
| Carbon dioxide (CO ₂)..... | 6.5 | 7.5 |
| Oxygen (O ₂)..... | .2 | .2 |
| Illuminants..... | .7 | .2 |
| Carbon monoxide (CO)..... | 33.1 | 32.3 |
| Methane (CH ₄)..... | 5.2 | 4.4 |
| Hydrogen (H ₂)..... | 48.9 | 49.0 |
| Nitrogen (N ₂)..... | 5.4 | 6.4 |
| Total..... | 100.0 | 100.0 |

The analysis of the blue gas made by the blow-run method does not include the gas actually made during the blow run, but illustrates the effect of this method on the quality of the gas produced during the steam run.

These two analyses represent typical differences in composition. The absolute analyses varied considerably according to conditions, including the temperature of the fuel bed, the amount of steam used, the length of the run, the length of blow run employed, and the quality of coal. Some analyses of the blue gas made during the first run after coaling gave calculated heating values as high as 354 B. t. u., while without the blow-run method 320 B. t. u. per cubic foot was the maximum obtained.

The analyses in Table 10 indicate the change in composition of the blue gas from run to run in the interval between two successive coalings.

TABLE 10.—*Analyses of blue gas produced during successive runs between coalings.*

| Constituents. | Up run—
First run
after
coaling
(B. t. u.—
353.4). | Down run—
Second run
after
coaling
(B. t. u.—
341.6). | Up run—
Third run
after
coaling
(B. t. u.—
336.2). | Down run—
Fourth run
after
coaling
(B. t. u.—
322.3). |
|--|---|--|---|--|
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Carbon dioxide (CO ₂)..... | 8.5 | 5.9 | 8.0 | 8.8 |
| Oxygen (O ₂)..... | .0 | .0 | .0 | .0 |
| Ill..... | 1.0 | .6 | .3 | .2 |
| Carbon monoxide (CO)..... | 32.0 | 32.2 | 30.2 | 28.3 |
| Methane (CH ₄)..... | 5.7 | (5.5) | 5.3 | 4.7 |
| Hydrogen (H ₂)..... | 51.0 | (53.0) | 53.4 | 54.6 |
| Nitrogen (N ₂)..... | 2.8 | (2.8) | 2.8 | 3.4 |
| Total..... | 100.0 | 100.0 | 100.0 | 100.0 |

Each sample was taken at a fairly uniform rate during the run, but since the rate of gas production was diminishing from the beginning to the end of each run, each analysis does not represent the absolute average of the gas made during the run.

The increased fuel-bed temperature attained altered materially the nature of the clinker. The alteration was greater with some coals than with others. The coals having less fusible ash formed very little clinker when the blow run was not used. Under these circumstances, much of the ash did not fuse, but remained soft. As a day's run progressed, this ash accumulated, with the result that the active fuel bed became constantly shallower and the rate of gas production decreased very materially toward the end of the running time. With the blow run, on the other hand, the temperature was sufficient to bring nearly all of the ash down to the grate as clinker and permitted the maintenance of a deep fuel bed. Despite the fact that the clinker might be expected to offer more resistance to the passage of air blast than unfused ashes, production was maintained at a more normal rate in the Streator plant with the blow-run method than without it.

The blow-run method also helped to counteract smoke production. As stated in a previous section, one of the sources of smoke is a cold generator that produces a lean blast gas and tarry vapors resulting from the distillation of the coal. As soon as the generator becomes hot, richer blast gas is produced, which burns when ignited in the checker chambers and assists in consuming the tarry vapors that would otherwise leave the set as smoke.

EFFECTS OF BLOW RUN ON COMPOSITION OF FINISHED GAS.

The volume of blow-run gas in the finished gas depends upon the length of the blow run, the blast pressure available, and the back pressure against which the gas produced has to be forced. In other words, it depends upon the volume of air blast that can be forced through the fire (for the production of blow-run gas) per 1,000 cubic feet of finished gas made. Obviously, if the back pressure of the collecting main from the set to the relief holder were the same as the gas pressure at the outlet of the set during the blow run, no gas would flow from the set to the holder. This is a condition that may actually exist in some plants, especially those larger ones in which several sets are being operated with but one collecting main.

With the blast pressure obtainable in the Streator plant it was possible to force about 1,260 cubic feet of air per minute through the fire during the blow. During the blow run with the same coal, the air meter showed only 1,000 cubic feet of air per minute passing through the fire, although the air pressure at the base of the generator had increased from 18 to 28 inches, due to this resistance. From the composition of the gas, it is estimated that blow-run gas

was produced from this volume of air at the rate of about 1,200 cubic feet per minute; therefore, with a 30-second blow run, about 600 cubic feet of gas, or approximately 18 per cent of the finished gas per run, was made during the blow-run period. It is, moreover, estimated, in lieu of any exact measurements, that the volume of the blow-run gas was equal to 25 to 30 per cent of that of the uncarbureted gas. The presence of this amount of blow-run gas in the finished gas, and the use of additional oil to carburet to the required standard, gives a gas of the composition shown in Table 11. An analysis of finished gas produced without the blow run is also shown for comparative purposes. These analyses are typical of the finished gas produced.

TABLE 11.—*Composition of finished gases.*

[B. t. u. (computed): With blow run, 574; without blow run, 560.]

| | Without
blow run. | With
blow run. |
|-----------------------|----------------------|-------------------|
| CO ₂ | 5.8 | 4.7 |
| O ₂ | .5 | .5 |
| Illuminants | 8.7 | 10.0 |
| CO..... | 27.3 | 25.5 |
| CH ₄ | 12.5 | 13.7 |
| H ₂ | 38.0 | 27.1 |
| N ₂ | 7.2 | 18.5 |
| Total..... | 100.0 | 100.0 |

The amount of oxygen is the same in both analyses (0.5 per cent). This oxygen was probably not in the gas originally, but was introduced into it when the sample was taken. Since one volume of oxygen is associated with about four volumes of nitrogen in the air it is evident that the actual amounts of nitrogen in the two gases were 5.2 and 16.5 per cent, respectively.

Analyses show that the blow run does not introduce air as such into the gas; accordingly, there is no danger of making an explosive mixture in the holder or mains.

Assuming that the heating value of the two gases is identical and that it is economically feasible to use the blow run in a given case, there appears to be no important reason, from the utilization standpoint, why the nitrogen content should be objectionable. Criticism of the presence of inert material in the gas is usually based upon the theoretical temperature usually called the "flame temperature" that could be obtained if all the heat from burning the gas were used to heat the products of combustion only. This flame temperature (as computed mathematically from the gas analysis, using the commonly accepted values for specific heats of the combustion products at high temperatures) is usually lower in the presence of appreciable amounts of inert substances, since it must be assumed that the latter are heated with the combustion products, although they do not con-

tribute heat. Such computations applied to the above analyses give "flame temperatures" of 3,560° F. for gas with the blow run and 3,600° F. for gas without the blow run, differing from each other by only about 1.7 per cent. When it is considered that most of the gas is burned in appliances and used where the heated material attains a temperature far below the calculated flame temperature, it is evident that the transfer of heat from the gases to the heated material would be little affected by such small differences in the theoretical flame temperature. This difference becomes still more negligible when it is considered that with average operation less than 50 per cent of the heat of combustion usually passes into the material heated. The investigation committee of the Institution of Gas Engineers (British)¹² reported that its investigations in the utilization of gases of various grades and compositions indicate that the effect of the presence of 20 to 25 per cent of inert substances on combustion of gases having a heating value of more than 500 B. t. u. is practically negligible.

As will be noted from the analyses, the percentage of illuminants is higher in the gas made with the blow-run cycle. This is to be expected, since more oil is required to maintain a given standard. No evidence, however, was gathered from the experiments at Streator that the gas was any more stable with one method than with another. The unseasonably warm weather during the course of the experiments made the determination of condensation losses difficult. It might be stated that while the experiments were in progress at Streator there was no report by the company of complaints arising from the quality of gas furnished to consumers.

EFFECTS OF BLOW RUN ON CARBURETOR AND SUPERHEATER.

An attempt to bring the generator to a good working temperature with coal fuel usually necessitates a length of blow period that overheats the carburetor or superheater, unless much of the combustible blast gas is burned at the stack, or heat is taken from these chambers by overblowing them. Increasing the blasting time by the introduction of a blow run, even though this blow run be but a fraction of a minute in duration, assists materially in heating the generator, but does not add any appreciable amount of heat to the carburetor and superheater, since the gas produced in the generator during the blow run is not burned in these chambers and the temperature of this gas as it leaves the generator is probably lower than that of the checkerwork.

In order to illustrate the changed conditions brought about by the use of the blow-run method and to point out the more prominent factors involved in the selection of a proper cycle, the two sample

¹² Gas World, vol. 69, Oct. 19, 1918, p. 1787.

tests presented as Table 12, embodying some of the results obtained during the experiments at Streator, are considered. Column A shows the operating figures without a blow run and column B the figures with a blow run.

TABLE 12.—Typical results showing some of the effects of the blow-run method of operating.

| | Test A (without
blow run). | Test B (with blow
run). |
|--|-------------------------------|---|
| Blue gas ^acu. ft. per 1,000 ft.. | 815 (314 B. t. u.) | 625 (337 B. t. u.) |
| Blow-run gas.....do. | | 180 (153 B. t. u.) |
| Oil gas ^ado.... | 185 (1,670 B.t.u.) | 195 (1,656 B.t.u.) |
| Total.....do.... | 1,000 | 1,000 |
| Make per run.....cubic feet.. | 2,700 | 3,500 |
| Oil per run.....gallons.. | 7.7 | 10.5 |
| Oil per thousand feet.....do.... | 2.85 | 3.05 |
| Per cent of blue gas in finished gas..... | 81.5 | 62.5 |
| Per cent of blow-run gas..... | | 18 |
| Gas burned at stack..... | Considerable. | Very little. |
| Cycle..... | 8-minute blast, 4-minute run. | 3-minute blast, ½-minute blow run, 3½ minute run. |
| B. t. u. of finished gas..... | 565..... | 565 |
| Total blue gas made per run..cubic feet.. | 2,200..... | 2,190 |
| Blue gas made per minute of steam run, allowing 15 seconds of running time for change of valves.....cubic feet.. | 566..... | 675 |
| Air per thousand feet..... | 1,400..... | 1,265 |
| Steam per thousand feet..... | 43.2..... | 35.3 |

It is evident from the figures in Table 12 that more generator gases are produced for carbureting per run in test B than in test A. Naturally, this requires that more oil be admitted to the carburetor during the run represented by test B. The increased amount of oil absorbs more heat from the checker chambers. This requires the use of more generator air blast (blast plus blow run) with the resulting hotter generator, and at the same time permits practically all of the combustible blast gas to be burned in the checker chambers.

It can be said, in general, that when the same amounts of air and steam are used, respectively, per minute in each case, the ratio of the total blasting time (including the blow run) to the length of the steam run must be greater with the blow-run method than without it. Although a blow run of almost any desired length may be used with a given air-blast condition, it must be remembered, in selecting a blow run of specified length, that as the duration of the blow run is increased, a greater quantity of gas is produced for carbureting. Hence, more oil is required per run for enrichment, correspondingly more heat is absorbed from the checker chambers, and therefore a longer blast or a shorter steam run is required in

^a The exact percentages of blue gas and oil gas are not known and the figures given here are estimates only.

order to maintain the heat balance throughout the set. When the blow run is being tried for the first time, there is sometimes a tendency to choose such a cycle that the checker chambers do not become hot enough to crack and fix the oil properly. This usually arises from failure to understand the nature of the changed conditions just discussed. The tendency when this condition occurs is to shorten the blow run, whereas greater capacity and a better heat balance could frequently be obtained by lengthening the blast period, increasing the blast pressure, or shortening the steam run.

EFFECTS OF BLOW RUN ON FUEL AND OIL ECONOMY.

In the discussion of the effect of the blow run on the composition and the quantity of blue gas produced, it was pointed out that a certain volume of blow-run gas (the richer blast gas produced at the end of the blasting period) is saved by passing it into the relief holder. In composition, this gas resembles that made in a producer especially designed for bituminous coal. One might expect that the composition of this gas would vary greatly from blow to blow, as the fresh fuel introduced during the charge is distilled and is gradually transformed into coke. This transformation is much more gradual than might be expected, and the following analyses of the blow-run gases for each successive blow between coalings illustrate the comparative uniformity of the gas produced over successive blow-run periods:

TABLE 13.—*Analyses of blow-run gas—successive runs.*

| Constituents. | First blow run after coaling. | Second blow run after coaling. | Third blow run after coaling. | Fourth blow run after coaling. | Average. |
|-----------------------|-------------------------------|--------------------------------|-------------------------------|--------------------------------|------------------|
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| CO ₂ | 2.1 | 2.7 | 1.9 | 2.4 | 2.28 |
| Illuminants..... | .6 | .4 | .3 | .1 | .35 |
| CO..... | 28.4 | 26.2 | 29.9 | 27.1 | 27.90 |
| CH ₄ | 3.0 | 3.3 | 3.0 | 2.9 | 3.30 |
| H ₂ | 5.5 | 7.5 | 5.1 | 6.2 | 6.07 |
| N ₂ | 60.4 | 59.9 | 59.8 | 60.3 | 60.10 |
| Total..... | 100.0 | 100.0 | 100.0 | 100.0 | 100.00 |

These analyses give calculated heating values for the successive blow runs of 157, 152, 153, and 149 B. t. u., respectively, the average being 153 B. t. u. The utilization of the blow-run gas of this average heating value means a saving of generator fuel. In some of the tests at Streator, the volume of blow-run gas amounted to approximately 18 per cent, or 180 cubic feet out of every 1,000 cubic feet of finished gas made. The heat in this amount of gas would be 27,500 B. t. u., or the equivalent of about 2.3 pounds of coal. Over a period of several days' operation, the generator fuel averaged about 5 pounds less

per 1,000 cubic feet of finished gas with the blow-run method than without the blow run, despite the fact that the increased capacity obtained in the former case reduced the daily operating time nearly 20 per cent, thereby making longer lay-over periods during which radiation and convection losses were taking place.

Part of the total economy obtained when operating with the blow-run method is due to a decreased amount of fuel consumed by the air blast, and part of it is due to a more efficient utilization of the steam during the run.

The amount of fuel consumed during the blasting of the generator is roughly proportional to the volume of air blast used. It is not always exactly proportional, since the two following reactions take place in greater or less degree, according to the initial temperature of the fuel, the rate of blast, and other factors:

1. $C + O_2 = CO_2$.
2. $2C + O_2 = 2CO$.

The carbon consumed according to reaction 1 is only half as great as that consumed by reaction 2, using the same amount of air.

The amount of generator air blast used per 1,000 cubic feet of finished gas is noticeably less when the blow-run method is used than when it is not. The amount of air used per 1,000 cubic feet in test B of Table 12 is 90 per cent of that used per 1,000 cubic feet in test A. If reactions 1 and 2 were taking place in the same proportions in tests A and B, the fuel consumed during the blast (not the total generator fuel per 1,000 feet) would be proportional to the air used per 1,000 cubic feet. In these examples, the indications are that this was approximately true and that the fuel consumed during the blast per 1,000 cubic feet of finished gas made in test B was 90 per cent of that consumed per 1,000 in test A. This is derived as follows:

$$\frac{\text{Air per 1,000 cubic feet in test B} = 1,265 \text{ cubic feet}}{\text{Air per 1,000 cubic feet in test A} = 1,400 \text{ cubic feet}} \times 100 \text{ per cent} = 90 \text{ per cent.}$$

The air used in the blow run was included in the air per 1,000 feet (1,265 cubic feet).

This factor (90 per cent) is emphatically not a constant, but would vary under different conditions. It does not pertain to the total generator fuel per 1,000 feet, but only to that part consumed during the blasting of the generator.

The amount of steam used per minute of run in both tests was approximately the same, but the steam used per 1,000 cubic feet of finished gas was decidedly less with the blow-run method than without it. The amount of blue gas made per minute in test B was much greater than that made in test A, although the total blue gas made per run was about the same in each case. Since the total steam used for the production of the same amount of blue gas was less in test B,

an evident saving is made by reducing the amount of fuel consumed by the steam during the run. Inasmuch as the same amount of gas is produced in test B as in test A, with 7 pounds less of steam per 1,000 feet, it may be assumed that the saving per 1,000 feet is equivalent at least to the amount of heat required to raise the 7 pounds of steam to the temperature of the effluent gas—approximately 1,350° F. This would amount to about one-third of a pound of generator fuel per 1,000 cubic feet of gas made.

The total heat absorbed by the 7 pounds of steam passing through the fire and being heated to 1,350° F. is approximately 3,641 B. t. u.

The hotter generator maintained with the blow-run method may occasionally result in a further saving of fuel, in that there is usually less likelihood of the fuel passing through the fire unconsumed.

Since the data in specimen tests A and B are taken from results obtained in a plant operating only 6 or 7 hours per day, it is impossible to predict accurately from the information available what the relative fuel economies with these two methods would be in plants operating full time. The indications are, however, that a material fuel saving could be expected with the blow-run method.

The oil economy is also affected by the use of the blow run, the extent of this effect depending upon the length of the blow-run period and the amount of gas produced during that period. The amount of oil required per 1,000 cubic feet of gas to maintain a certain standard in a given plant would be difficult, if not impossible, to predict with certainty unless many factors were known, such as the kind of oil, the quality and quantity of blow-run gas and blue gas made, and weather conditions. The economy possible when the blow run is used can be determined only by trial.

The mixture of blue gas and blow-run gas produced by the blow-run method is not as rich as the blue gas produced without the blow run; therefore, more oil is required for enrichment in making 1,000 cubic feet of finished gas in the former case. In general, it may be said that the oil required per 1,000 cubic feet of finished gas increases as the percentage of blow-run gas in the mixture increases. The increased amount of oil required, however, is not directly proportional to the percentage of blow-run gas, since, as has already been shown, the quality of blue gas produced is richer with the blow-run method.

The quality of the coal used also affects oil economy. The volatile matter of some coals has a higher heating value than that of others, and this affects the quality of both the blue gas and the blow-run gas. Under the conditions existing at Streator, a heating-value standard of 565 B. t. u. was maintained a mile from the plant with 3 to 3.05 gallons of Texas oil per 1,000 cubic feet of gas, using the blow run, against about 2.85 gallons per 1,000 feet, without the blow run.

The length of blow run used during the experiments varied from 15 to 45 seconds and the oil varied accordingly. It is noticeable that while the oil per 1,000 cubic feet using coal is greater with the blow-run method than without it, the amount of oil used per 1,000 cubic feet with coke fuel and without the blow run is still greater.

The oil economies obtainable show considerable seasonal variation. More of the condensable heat-producing constituents of the gas are chilled out in cold weather than in the summer months. This requires that a greater quantity of oil per 1,000 cubic feet of finished gas be used in winter. The figures given above were obtained in the late autumn.

COMPARATIVE EFFICIENCY OF OIL CRACKING WITH COKE AND WITH COAL FUEL.

The figures presented for oil consumption per 1,000 feet, operating with and without a blow run, are the results of more than one day's run and are not a selection of the best figures obtained in any series of tests. They have, in the main, been duplicated in other plants and are probably accurate enough for all practical purposes.

The cracking efficiency of the oil used, however, can not be stated with such accuracy. In attempting to derive a figure that will express oil-cracking efficiency, it is necessary to know the exact amount of oil gas present in 1,000 cubic feet of finished gas and to ascertain its heating value. In the Streater experiments, no attempt was made to determine this. However, the blue gas and the blow-run gas were analyzed and the volumes estimated from the available figures. The volume and heating value of the oil gas (gas from oil cracking) can be approximated by difference. Typical data are given in Table 14. The cracking efficiencies presented in this table are so nearly alike that without additional information it is impossible to say definitely that the oil cracks more completely into gas with coke fuel than with coal. However, at the present time it is the author's belief that these efficiency figures are representative of the results obtained. It is not improbable that the great excess of steam used when operating with coal and without the blow run had a beneficial effect on the oil cracking, resulting in the increased oil-cracking efficiency shown. Although in some instances a greater amount of carbon is deposited in the carburetor when coal fuel is used, there is little question that some of this results from the carbonization of the tarry matter distilled from the coal in the generator; therefore, this deposit can not be taken as evidence of a decreased efficiency of oil cracking.

TABLE 14.—*Oil efficiencies with coal and with coke fuel.*

| | Test A. | Test B. | Test C. |
|--|-------------------------------|---|-------------------------------------|
| Fuel used..... | Bituminous coal..... | Bituminous coal..... | Coke. |
| Blow-run cycle..... | None | Yes. | None. |
| Cycle..... | 3-minute blast, 4-minute run. | 3-minute blast, 4-minute blow run, 24-minute steam run. | 3-minute blast, 4-minute steam run. |
| Oil: | | | |
| Amount used per 1,000 cubic feet of finished gas, gallons. | 2.85..... | 3.05..... | 3.15. |
| Gravity, pounds per gallon. | 7.3..... | 7.3..... | 7.3. |
| Weight per 1,000 cubic feet, pounds. | 20.8..... | 22.265..... | 22.995. |
| Total heating value, B. t. u. | 416,000..... | 445,300..... | 459,900. |
| Gas: | | | |
| Blue gas made per 1,000 cubic feet, cubic feet. | 815 (314 B. t. u. per ft.). | 625 (337 B. t. u. per ft.). | 790 (292 B. t. u. per ft.). |
| Blow-run gas made per 1,000 cubic feet, cubic feet. | | 180 (153 B. t. u. per ft.). | |
| Oil gas made per 1,000 cubic feet, cubic feet. | 185 (1,670 B. t. u. per ft.). | 195 (1,656 B. t. u. per ft.). | 210 (1,592 B. t. u. per ft.). |
| Total gas made, cubic feet. | 1,000..... | 1,000..... | 1,000. |
| B. t. u. per foot of finished gas. | 565..... | 565..... | 565. |
| Total heat of combustion, per 1,000 cubic feet of finished gas, B. t. u. | 565,000..... | 565,000..... | 565,000. |
| B. t. u.: | | | |
| Furnished by the blue gas. | 255,600..... | 210,625..... | 230,680. |
| Furnished by the blow-run gas. | | 27,540..... | |
| Total in the oil used..... | 416,000..... | 445,300..... | 459,900. |
| To be supplied by the oil per 1,000 cubic feet. | 309,400..... | 326,835..... | 334,320. |
| Oil in carbon, tar, condensable products. | 106,600..... | 118,465..... | 125,580. |
| Efficiency of oil cracking, per cent. | 74.4..... | 73.4..... | 72.7. |

In the change from coke to coal fuel, the following variables, which affect oil cracking, are altered: Atmosphere in which the oil is cracked, time and intimacy of contact of the oil vapor with the hot checker bricks, volume of oil used per 1,000 feet of finished gas, volume of oil used per 1,000 feet of blue gas, make per unit of time, and heat absorbed from checker chambers during the run.

It is evident that the problem of checking cracking efficiency by theoretical considerations is therefore very complex.

EFFECT ON CAPACITY.

The blow-run method was first worked out with the hope of increasing the capacity with coal as fuel approximately to that attained with coke. The blow run is recognized to have practical limitations and should only be employed to an extent consistent with reasonable economy of fuel and oil. As stated, it was found that with a blow run of reasonable length the oil per 1,000 cubic feet of finished gas was less than that required when coke fuel was used. The amount of coal fuel per 1,000 cubic feet was less with the blow run than without it. At the same time, the capacity was increased until it approached that attained with coke fuel. Assuming that the blast and steam conditions are adequate and that pressure conditions are suitable, the blow run could be carried still further in an

emergency, but of course the use of it to an excessive degree would necessitate the use of more oil per 1,000 cubic feet to enrich the increased volume of lean gas. The heat required to crack the oil properly when such an excess is used might exceed the heat-storage capacity of the checkerwork. Under such conditions, part of the oil would inevitably pass through the set undecomposed. An excessive employment of the blow run, necessitating the use of a greatly increased amount of oil per run, might be very disadvantageous under some conditions, on account of an increased formation of carbon in the checkerwork, a fact more noticeable with some gas oils than with others. The standard of gas quality to be maintained might be another limiting condition at times. It would probably be impracticable to meet a high candlepower standard with this method. With the increasing tendency toward lower heating-value standards and the abolition of candlepower standards, the field for the application of this method is greatly widened.

The back-pressure conditions in the Streater plant did not make practicable the introduction of a portion of the oil during the blow run, since the blast pressure was not sufficient to produce as much blow-run gas when the pressure in the set was increased by the addition of oil. Had it been practicable to do so, it seems likely that an advantage would have been realized. The oil could have been admitted more slowly, the concentration of oil gas in the checker chambers would not have been so great, and the oil cracking efficiency might have been greater. Whether the advantages from decreased concentration would be offset by a change in the efficiency of oil cracking in an atmosphere of blow-run gas, can not be stated. The rate of make during the run, including the blow run, is shown in Figure 10. Relief-holder readings were taken every 15 seconds and the readings plotted. The first drop in the curve, at the end of 30 seconds, indicates the change from blow run to steam run.

USE OF COAL AND COKE MIXTURES WITH THE BLOW RUN.

If a sufficiently large make of gas per hour can not be obtained with 100 per cent coal and a blow run of reasonable length, it may be increased by using a fuel mixture (coal and coke) containing up to 25 per cent coke with a blow run of suitable length, depending upon the air blast available, the quality of gas to be maintained, and the hourly make desired. During a run of a few days at the Streater plant, with 25 per cent coke and a 30-second blow run, the hourly capacity was increased about 6 per cent over that attained during the days immediately preceding, when 100 per cent coal was used. The blast pressure was practically the same in both cases, but the air passing through the fire per minute during the blast was somewhat greater when the mixed fuel was used, probably the chief fac-

tor in causing the increased make. During the few days when this combination of conditions was tried, there was no percepti-

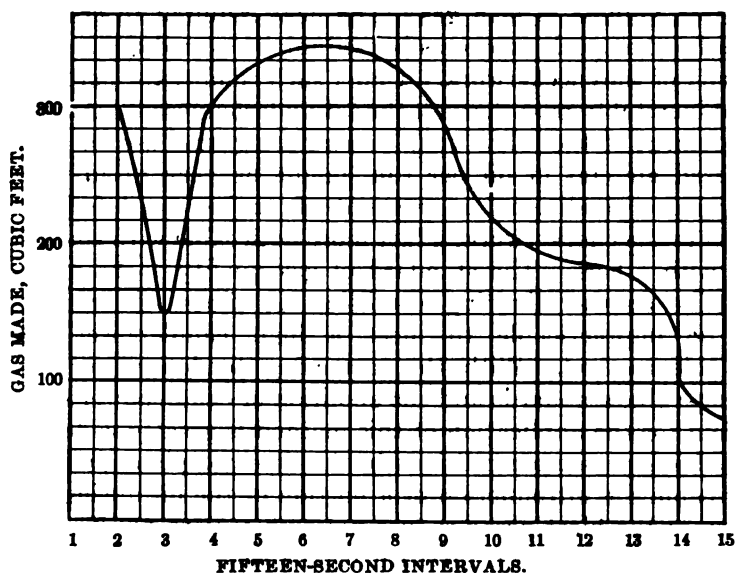


FIGURE 10.—Variation of gas made during run as determined from rise of relief holder during successive 15-second intervals.

ble increase in the amount of oil used per 1,000 cubic feet of gas to maintain practically the same heating value.

TABLE 15.—Conditions and operating results with various coals in the Streator plant.¹

| Coal used. | Perry County, Ill. | | Franklin County, Ill. | | Williamson County, Ill., A. | | Williamson County, Ill., B. | Retort coke. ² | Indianapolis oven coke. ³ |
|--|--------------------|-------------------|-----------------------|-------------------|-----------------------------|-------------------|-----------------------------|---------------------------|--------------------------------------|
| | With blow run. | Without blow run. | With blow run. | Without blow run. | With blow run. | Without blow run. | With blow run. | | |
| Daily make (corrected) | | | | | | | | | |
|cu. ft. | 165 | 155 | 180 | 162 | 163 | 172 | 177 | 184 | 270 |
| Make per hr. (corrected) | | | | | | | | | |
|cu. ft. | 26 | 21.7 | 28 | 23.2 | 25.6 | 23.4 | 24.8 | 23 | 25.4 |
| Make per hour per square foot grate. | 2,069 | 1,735 | 2,260 | 1,880 | 2,048 | 1,875 | 1,980 | 1,840 | 2,016 |
| Make per run (corrected) | | | | | | | | | |
|cu. ft. | 3,374 | 2,656 | 3,600 | 2,945 | 3,281 | 2,881 | 3,160 | 2,706 | 2,630 |
| Cycle: | | | | | | | | | |
| Blow.....minutes. | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 3 |
| Blow run.....seconds. | 30 | 0 | 30 | 0 | 25 | 0 | 30 | 0 | 0 |
| Steam run.....minutes. | 3.4 | 3.9 | 3.4 | 2.9 | 4 | 3.9 | 3.9 | 4 | 5 |
| Air purge.....seconds. | 6 | 6 | 6 | 6 | 6 | 6 | 6 | 6 | 0 |
| Up and down runs. | 1 to 1 | 2 to 1 | 1 to 1 | 1 to 1 | 1 to 1 | 1 to 1 | 1 to 1 | 2 to 1 | |
| Blast pressure during blow | | | | | | | | | |
|inches. | 17.5 | 18 | 18 | 18 | 18 | 18 | 18 | 16 | 19 |
| Air per minute during blow | | | | | | | | | |
|cu. ft. | 1,258 | 1,240 | 1,086 | 1,060 | 1,130 | 1,100 | 1,210 | 1,730 | 1,890 |
| Air per minute during blow run. | | | | | | | | | |
|cu. ft. | 1,016 | 0 | 975 | 0 | 980 | 0 | 800 | 0 | 0 |
| Air per 1,000 cubic feet finished gas. | | | | | | | | | |
|cu. ft. | 1,265 | 1,401 | 1,060 | 1,070 | 1,184 | 1,180 | 1,275 | 1,940 | 1,400 |

¹ These results were obtained with a set having a generator of 4 feet internal diameter.

² This was the coke formerly used at the Streator plant. On account of the obstinate clinker, excess steam had to be used. It gave inferior results.

³ These results were obtained in another plant with a good grade of oven coke. They are given for comparison.

TABLE 15.—*Conditions and operating results with various coals in the Streator plant—Continued.*

| Coal used. | Perry County, Ill. | | Franklin County, Ill. | | Williamson County, Ill., A. | | Wil-
hamson
County,
Ill., B. | Retort
coke. | Indian-
apolis
oven
coke. |
|--|----------------------|------------------------------|-----------------------|------------------------------|-----------------------------|------------------------------|---------------------------------------|-----------------|------------------------------------|
| | With
blow
run. | With-
out
blow
run. | With
blow
run. | With-
out
blow
run. | With
blow
run. | With-
out
blow
run. | With
blow
run. | | |
| Steam per minute: | | | | | | | | | |
| Up run.....lbs.. | 35 | 31 | 35 | 32 | 35 | 34 | 34 | 40 | 22 |
| Down run.....do.. | 33 | 33 | 37 | 34 | 34 | 32 | 32 | 36 | 27 |
| Steam per 1,000 cubic feet fin-
ished gas.....lbs.. | 34.2 | 47 | 33.7 | 45 | 37.2 | 45.5 | 40.7 | 42 | 33 |
| Generator fuel per 1,000 cubic
feet.....lbs.. | 47.3 | 52 | 42.3 | 48.8 | 43 | 46 | 44.5 | 44.5 | 36.7 |
| Average weight of charge
.....lbs.. | 644 | 600 | 600 | 700 | 700 | 660 | 606 | 800 | |
| Charging intervals.....runs.. | 5 | 5 | 5 | 5 | 5 | 5 | 5 | 5 | |
| Oil per 1,000 cubic feet
.....gallons.. | 3.04 | 2.84 | 3.02 | 2.78 | 3.24 | 2.97 | 3.06 | 2.94 | 3.58 |
| Oil per run.....do.. | 10-11 | 7.5 | 10-12 | 8-9 | 10-12 | 8.5 | 10-11 | 8 | 13 |
| Daily operating time, hours.. | 6.35 | 6.9 | 6.45 | 7 | 6.36 | 7.35 | 7.15 | 8 | 11 |
| B. t. u. of city gas, 1 mile
from plant..... | 562 | 565 | 565 | 556 | 570 | 578 | 567 | 560 | 600 |
| Hydrogen sulphide in unpur-
ified gas..... | 150 | 150 | 160 | 160 | 200 | 200 | 420 | 100 | |
|grains per 100 cu. ft.. | (1) | (2) | (2) | (2) | (2) | (2) | (2) | (2) | |
| Nature of clinker..... | | | | | | | | | |

* Hard, some edgings, brittle.

* Rather soft.

* Hard on grate, easily removed.

* Hard, brittle, easily handled.

* Hard, excess steam necessary to make workable.

OPERATING RESULTS AT STREATOR.

Table 15 includes typical operating results obtained with various coals during the Streator experiments. These results are not averages, since the conditions were frequently changed during the course of the experiments to get what seemed to be the best combination, and in the course of such experimenting there were tried some operating methods that were obviously unsuitable, since they gave very poor results. Moreover, such unforeseen factors as low blast pressure and low steam pressure occasionally caused a poor record for a day, yet could in no way be attributed to the nature of the coal under test. Therefore, while neither the best nor the poorest results are given, and the results tabulated are not strictly averages, in the judgment of the authors they represent typical performances possible of duplication under plant conditions similar to those at Streator.

COALS USED IN TESTS.

All the coals used in obtaining these results were from the southern Illinois field. With the exception of the Perry County coal, which had been used in the Streator plant for some time prior to the experiments, one carload only of each coal was tested. The fuel consumption of the plant was only about 4 tons per day, therefore each carload of coal was sufficient for a run of about 10 days, varying according to the percentage of the coal that was too fine for the generator. This latter was, however, used as boiler fuel.

The coal forked from each carload was sampled and the samples subsequently analyzed in the laboratories of the University of Illinois at Urbana. The results of the analyses are given in Table 16 following:

TABLE 16.—*Analyses of coals used in Streator tests.*

| Source of coal. | Moisture. | Volatile matter. | Fixed carbon. | Ash. | Total. | B. t. u. | Total sulphur. |
|-----------------------------|------------------|------------------|------------------|------------------|------------------|----------|------------------|
| Perry County, Ill. | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | | <i>Per cent.</i> |
| As received..... | 7.77 | 33.86 | 49.30 | 9.07 | 100.0 | 11,827 | 1.29 |
| Air dry..... | 2.36 | 35.85 | 52.19 | 9.60 | 100.0 | 12,514 | 1.37 |
| Moisture free..... | | 36.72 | 53.45 | 9.83 | 100.0 | 12,816 | 1.40 |
| Franklin County, Ill.: | | | | | | | |
| As received..... | 9.98 | 32.54 | 47.49 | 9.99 | 100.0 | 11,419 | 1.25 |
| Air dry..... | 2.30 | 35.32 | 51.54 | 10.84 | 100.0 | 12,393 | 1.36 |
| Moisture free..... | | 36.15 | 52.75 | 11.10 | 100.0 | 12,684 | 1.39 |
| Williamson County, Ill., A: | | | | | | | |
| As received..... | 7.32 | 31.82 | 50.42 | 10.44 | 100.0 | 11,806 | 1.57 |
| Air dry..... | 2.94 | 33.33 | 52.80 | 10.93 | 100.0 | 12,364 | 1.64 |
| Moisture free..... | | 34.34 | 54.40 | 11.26 | 100.0 | 12,738 | 1.69 |
| Williamson County, Ill., B: | | | | | | | |
| As received..... | 6.40 | 33.26 | 50.68 | 9.66 | 100.0 | 12,109 | 2.29 |
| Air dry..... | 2.84 | 34.52 | 52.61 | 10.03 | 100.0 | 12,599 | 2.38 |
| Moisture free..... | | 35.53 | 54.15 | 10.32 | 100.0 | 12,936 | 2.45 |

In addition to the tests of the coals named in the above table, short tests were made with two other coals, one from La Salle County, Ill., the other from Sullivan County, Ind. Of these, the La Salle County coal was found to be so high in sulphur that operation with it was discontinued after about three hours. During this short run, the indications were that the coal was inferior in its gas-making qualities, since the fire burned down more rapidly than with the other coals, yet less gas seemed to be produced.

The Sullivan County, Ind., coal also was tried for but a short run. It gave a higher sulphur content to the gas than most of the Illinois coals although not so much as could not have been removed by the purifiers, had not the purifying capacity of the plant been already overworked by the use of some of the higher sulphur coals that were tried previous to this. Indiana coals from the same field have been successfully used for water-gas making, as will be shown a little later.

Of all the coals tested, that from Franklin County showed slightly the best general results, although it gave slightly more sulphur to the gas than did the Perry County coal. Its clinker was somewhat harder than that formed by other coals tested, and apparently fused at a lower temperature. The clinker from the Franklin County coal tended to form on the grate and to adhere less to the generator wall than that from the Perry County coal. Under the prevailing conditions, the harder clinker formed by this coal made the cleaning of the generator fire easier than with the other coals, since it held up the fire better. The resistance which this coal presented to the passage of air through the fire was distinctly greater than that of the Perry County coal. Nevertheless, a larger hourly make and better

fuel economy were obtained with it. Its effect on the oil consumption was about the same as that of the Perry County coal.

The Perry County coal showed slightly less sulphur in the gas than any of the other coals tried. A much longer run was made with this coal than with the others, and it was used in obtaining the results already given with fuel mixtures. Owing to the extensive experimenting with this coal, its peculiarities were better known than those of the other coals, and the data collected concerning it therefore made a good standard with which to compare them. The Perry County coal was considered very satisfactory for water-gas manufacture.

The Williamson County coal, A, did not give as good results with the blow-run method of operations as did the coals already discussed. Its slightly lower percentage of volatile matter and its higher ash content seem to be responsible for this. On the other hand, its higher fixed carbon content seemed to make it more adaptable to the usual method of operating—without the blow run—than the other coals, and with that method it gave a slightly greater hourly output.

The clinker formed from its ash resembled that of the Franklin County coal, its resistance to the passage of air blast was between that of the Franklin County and Perry County coals, and its sulphur content was somewhat higher than either.

The Williamson County coal, B, contained so much sulphur that it would not generally be considered available for gas making under present conditions, therefore only a short run was made with it and no very definite idea of its gas-producing qualities can be formed. The indications are that generally it would be a little inferior to the other coals tried.

SULPHUR IN GAS.

The tests at Streator indicate that, other conditions remaining the same, the hydrogen sulphide in the unpurified carbureted gas is roughly proportional to the per cent of total sulphur in the coal. In Figure 11 this relation is illustrated. There was, however, one marked exception among the seven coals tested, although this might have been due to the fact that the test with this coal was too short to permit many hydrogen sulphide tests to be made on the gas. Whether a similar relation would hold with other coals can not be stated. In all the tests, the same kind of oil was used and operating conditions were fairly similar, so that the percentage of sulphur in the coal might reasonably be expected to be the prime factor affecting changes in the sulphur content of the gas. However, different conditions in washing and cooling the gas alter the amount of H_2S in the unpurified gas; therefore the same relation between sulphur in coal and sulphur in gas would not exist if these conditions were changed.

USE OF BITUMINOUS GENERATOR FUEL IN PLANTS OTHER THAN STREATOR.

The work done at Streator has led to the use of bituminous coal fuel in water-gas manufacture at a number of plants in various parts of the country. Moreover, a number of companies who used bituminous coal in their plants prior to the Streator tests have been interested in comparing their own operating results with those obtained at Streator. Some of the results obtained by such plants in different parts of the country under various operating conditions with different coals are presented in Table 17. Some of these data show results obtained with eastern gas coals, although a majority of the plants mentioned used Illinois or Indiana coals.

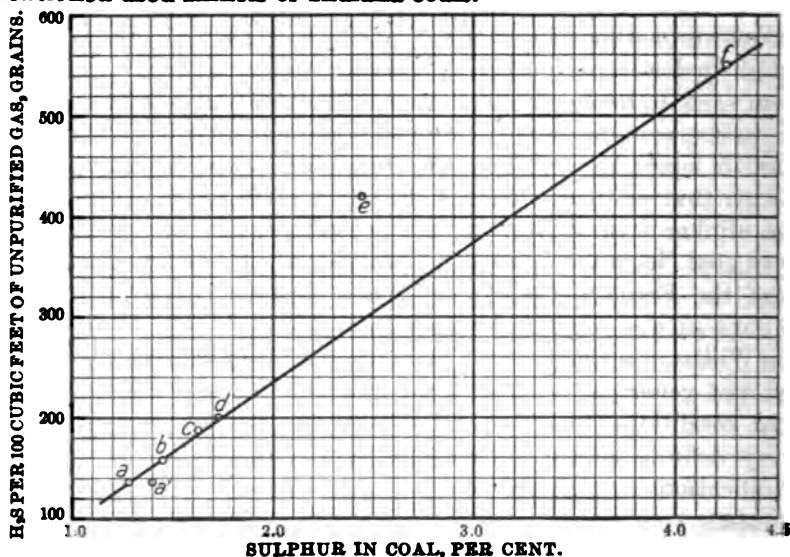


FIGURE 11.—Relation of sulphur in coal to hydrogen sulphide in unpurified carbureted gas. Key to sources of coal: a, Perry County, Ill. (old coal); a', Perry County, Ill. (new coal); b, Franklin County, Ill.; c, Sullivan County, Ind.; d, and e, Williamson County, Ill.; f, La Salle County, Ill.

TABLE 17.—Operating conditions and results with bituminous generator fuel in plants other than Streator.

| Plant.....
Internal diameter of generator, feet.....
Kind of fuel used..... | A | B | C | D | E | F | G |
|---|---|--|--|------------------------------|---|--|---|
| | 6
Clinton
County,
Ind.,
bituminous. | 4
Franklin
County,
Ill.,
bituminous. | 6½
Perry
County,
Ill.,
bituminous. | 5
Virginia
bituminous. | 8½
50 per cent
anthracite;
50 per cent
eastern gas
coal. | 4½
Harian
County,
Ky.,
bituminous. | Franklin
County,
Ill.,
bituminous. |
| Daily make, in 1,000 cubic feet corrected..... | 550 | 344 | 1,225 | 335 | 2,323 | 225 | 574 |
| Daily operating time, including cleans.....hours.. | 14.25 | 18 | 24 | 12.5 | 22.85 | 13 | |
| Cycle: | | | | | | | |
| Blow, minutes | 3 | 3 | 2.66 | 3 | 3 | 4 | 3 |
| Blow run, minutes..... | .5 | 0 | .23 | 0 | 0 | .5 | .5 |
| Steam run, minutes..... | 3.25 | 4.9 | 3.84 | 4 | 4 | 3.75 | 3.75 |
| Purge, minutes..... | .25 | .1 | .16 | 0 | 0 | .25 | .25 |

TABLE 17.—Operating conditions and results with bituminous generator fuel in plants other than Streator—Continued.

| Plant.....
Internal diameter of generator.....feet.
Kind of fuel used. | A
6
Clinton County, Ind.,
bituminous. | B
4
Franklin County, Ill.,
bituminous. | C
6½
Perry County, Ill.,
bituminous. | D
5
Virginia bituminous. | E
8½
50 per cent anthracite;
50 per cent eastern gas coal. | F
4½
Harian County, Ky.,
bituminous. | G
Franklin County, Ill.,
bituminous. |
|--|--|---|---|--------------------------------|---|---|--|
| Blast pressure during blow, inches..... | 17-20 | 16-24 | | | | 14 | 17-20 |
| Air: | | | | | | | |
| Per minute during blow, cu. ft. | 2,800 | 1,250 | | | 10,000? | 1,400 | 2,800 |
| Per minute during blow run, cu. ft. | | 0 | | | 0 | 1,200 | |
| Per 1,000 cu. ft. finished gas.....cu. ft. | 1,410 | 1,155 | | | 2,140 | 2,010 | |
| Percentage of up runs..... | 33 | 66 | | | | 66 | 33 |
| Steam: | | | | | | | |
| Per minute up.....lbs. | 60 | 26.8 | | 40? | 128 | 34 | 60 |
| Per minute down.....lbs. | 60 | 28.8 | | 40? | 123 | 31 | 60 |
| Per 1,000 cu. ft. finished gas.....lbs. | 34.6 | 42.5 | | 48.4? | 35.8 | 40.1 | 36.4 |
| Make: | | | | | | | |
| Per running hour, cu. ft. | 45,250 | 21,700 | 55,700 | 28,450 | 110,500 | 20,245 | 50,500 |
| Per set hour, including cleans, cu. ft. | 41,600 | 19,800 | 52,100 | 27,130 | 98,000 | 19,200 | 45,840 |
| Per run, cu. ft. | 5,205 | 3,119 | 6,594 | 3,311 | 14,000 | 3,082 | 6,176 |
| Per running hour per sq. ft. grate, cu. ft. | 1,000 | 1,736 | 1,688 | 1,445 | 1,960 | 1,190 | 1,785 |
| Generator fuel per 1,000 cubic feet, lbs. | 43.4 | 43.4 | 41.9 | 42.29 | 33.57 | 46 | 39.14 |
| Average weight charge.....lbs. | 800-1,000 | | | 800 | | 1,175 | 800-1,000 |
| Charging interval, runs..... | 4-5 | | | 3 | | 8-10 | 4-5 |
| Oil: | | | | | | | |
| Kind used..... | Texas 27° | Okl. 30-36° | Texas 28° | Okl. 33-34° | Mex. 34° | Texas 30° | Texas 27° |
| Introduced, minutes..... | 3 | 4½ | 3.5 | 3 | 3 | 4 | 3 |
| Per 1,000 cubic feet, gallons. | 3 | 3.22 | 3.19 | 2.90 | 2.94 | 2.90 | 3.07 |
| B. t. u.: | | | | | | | |
| Amount in finished gas. | 565 | 600 | 568 | 600 | 602 | c. p. 18 | 565 |
| Where record was taken..... | Plant. | 1 mile from plant. | Plant. | Plant. | Plant. | | Plant. |
| Cleaning time per day.....minutes. | 69 | 120 | 94 | 40 | 160 | 60 | 69 |
| Nature of clinker..... | Hard, but easily handled. | Easily removed. | Hard, considerable edgings. | Brittle, close to grates. | Soft. | Mostly ash on grate. | |
| Carbon in ash and clinker..... | Very little. | Considerable. | | Practically none. | Considerable. | None. | |
| Hydrogen sulphide, raw gas, grains per 100 cu. ft. | 200 | | 150 | 70 | | | |

An inspection of the table shows a considerable diversity in results that can not be attributed alone to the difficulties encountered in using bituminous coal. It is probable that with coke or anthracite fuel quite as great a variation would be experienced in plants of diverse sizes and operating conditions. The data in Table 17 show that in plants using 100 per cent coal no better results were obtained with good eastern gas coal than with Central District coals. In only one plant under the observation of the authors have both classes of coal been used, and here there was little apparent difference in the results with fuels from different fields, although the Central District coals operated somewhat more satisfactorily. The eastern gas coal used by this plant, while of very good grade, was of smaller size than would have been desirable, and this condition perhaps affected the results adversely. It is also quite possible that the weaker coking properties of Central District coals as compared with eastern gas coals is a consideration in favor of the former coals for this purpose. No extensive studies of the coking property of coals in relation to their values in water-gas manufacture have been made, although the indications are that such studies would give much interesting information. Statistics on plant F of Table 17 indicate that mixtures of anthracite and bituminous coals have been used with considerable success. The action of the anthracite in such mixtures is similar to that of coke; it makes the fire more penetrable to the blast, and by the interposition of lumps of noncoking fuel between the lumps of coking coal breaks up the continuity of the caking mass.

SUGGESTIONS FOR OPERATING WITH COAL FUEL.

It is impossible to formulate fixed rules for operation that will apply equally well to all conditions. The operator using coal as generator fuel for the first time will probably need to experiment a little to find the conditions that best apply to his plant. The following suggestions are presented merely as a guide to those undertaking the use of coal as generator fuel for the first time:

1. The experiments may well be begun with only 50 per cent of coal and the per cent of coal in the mixtures increased gradually from day to day as the experiments progress.
2. Select a coal of low sulphur content. This is particularly necessary when the purifying capacity is limited.
3. The coal should not be too fine, but preferably in 4 to 6 inch lumps. Very fine material should be excluded by forking the coal with the usual coke fork.
4. When operation is begun, do not make double charges of coal, for heavy charges at the start usually result in a cold generator, difficulty in bringing up the heat, and excessive smoke production.

It is better to have a deep fuel bed by making a large charge shortly before shutting down the night before; then the carburetor can be lighted and a run made before any coal is charged. This will give a quicker start, diminish the smoke, and result in a large make from the first.

5. Build up the generator fire by coal charges of normal size (the same weight as when coke is used). Make the charges at intervals of two runs until the fire is up to the normal working height—that is, just below the take-off connection; then charge at intervals of about five or six runs. Make the charges as rapidly as possible, to avoid loss of time. Heavier charges may be made with coals that do not cause serious arching and caking difficulties.

6. Proportion the up and down runs and the steam used so that the clinker will come down to the grate where it can be readily removed. The proportion can best be determined by trial. At Streator, most of the coals did best with three up runs to two down runs. Usually two successive up runs were made after charging, though occasionally alternate up and down runs seemed more satisfactory.

7. The length of the blow and of the steam run will depend upon the blast and steam available. In general, try so to proportion the blow and the run that the generator, carburetor, and superheater will attain normal working temperatures at the same time. There is frequently a tendency for the carburetor and superheater to become too hot before the generator is hot enough. This overheating of the checker chambers has been partly overcome by one or more of the following methods:

a. By allowing some gas to burn at the stack.

b. By oversteaming the generator and thereby diminishing the amount of combustible gas formed, although this cools the generator still more.

c. By overblowing the carburetor and superheater.

d. By using the blow-run cycle.

Of these, the writers believe that method *d* is the most economical and results in the largest capacity per set. If this cycle is not desired, then method *c* or method *a* seems generally the least objectionable, when conditions permit.

8. Use reduced steam and oil during the first four or five runs, while the set is being brought up to a working temperature.

9. Make the blow run, if adopted, of only sufficient length to maintain the normal heat balance in the set and to obtain the desired make of gas without an excessive oil consumption per 1,000 cubic feet.

10. *Close the carburetor and superheater blast valves before the stack cap is shut to make the blow run. Do not reverse the hot valves while*

making the blow run. Close the generator blast valve before turning on the steam to make the regular steam run.

11. Be sure that after the carburetor and superheater blast valves have been closed preparatory to making a blow run, the burned gases in the set are purged out before the stack lid is closed. The time consumed in operating the valves and closing the lid is usually sufficient for this purpose, but mechanically operated valves sometimes have closed so quickly that much burned gas was blown into the holder.

12. Make sure that the carburetor and superheater chambers are not being overblown or underblown by the secondary air blast if they remain cold when the blow-run cycle is used. If the checker chambers do not now heat properly, it will probably be necessary to increase the length of the blast period, decreasing the steaming period correspondingly. This will accomplish the desired results. Failure of the checker chambers to heat properly has been incorrectly attributed to too long a blow run, and the blow run was accordingly shortened. This, of course, is not necessary.

13. Considerably more tar is produced with coal than with coke fuel. Care should be taken to remove the tar from the gas before it reaches the purifiers. The operation of the tar extractor or shavings scrubber especially should be watched.

14. Remove the flanges on the bottom of the hot valves occasionally to clean out deposits of carbon or carbonized tar. The presence of such deposits is indicated by the failure of the valves to close tightly.

15. Leave the generator lid slightly open or "cracked" during lay-over periods. This seems to assist in heating the generator and in burning off any carbon which may have been deposited in the checkerwork of the carburetor and superheater. When the stack draft is not sufficient to draw air in through the lid opening, it is not advisable to leave this lid open unless artificial means of inducing a draft are employed, such as a steam jet in the stack.

16. Watch for sticking of the hot valves and oil-spray stems during lay-over periods. This is due to pitch deposits, and may be alleviated by occasionally smearing the stems with a mixture of lubricating oil and graphite.

17. Use a fuel spreader whenever conditions permit.

18. Make sure that green fuel does not pass through the fire with the ashes. When the up and down runs are properly proportioned as suggested in 6, there should be very little loss on this account.

19. Shut down for a few minutes after about 30 runs are made, or midway between cleans, and break up the clinker with bars. If no attempt is made to remove the clinker very little time will be consumed and the make per run will increase nearly to normal.

20. Test the oil spray occasionally and make sure it is atomizing the oil properly. It is desirable to have several oil sprays on hand so that they can be changed and cleaned frequently.

21. When an oil is used that causes considerable carbon trouble, it may be well to overblast the carburetor, thus burning off some of the deposited carbon. When a set is operated with a long lay-over period, a better way would be to draw air through the checker chambers by means of the stack draft. This can be done by removing the oil spray.

BY-PRODUCTS.

TAR.

Tar is a by-product of nearly all water-gas processes now in common use. The quality and quantity of it vary in different plants and in any given plant as the operating conditions are changed. Some of the factors that have important bearing on both the quality and the quantity of tar produced are the temperature in the checker chambers, the quality and quantity of blue gas in which the oil is cracked, the intimacy and time of contact of oil vapors with the checkering, the time and method of injecting oil in the carburetor, the kind of oil used, and the kind of fuel used.

Since gas is the main product sought and the formation of tar is looked upon as a necessary evil, the quantity of tar produced per gallon of oil used is of secondary consideration. The quality and quantity produced, when operating with a specified fuel and oil, may be said to vary as the temperature in the oil-cracking chambers varies. Excessively high temperatures in these chambers are favorable to the production of lampblack or naphthalene and the formation of less tar. On the other hand, low temperatures favor the formation of an increased amount of tar. In other words, the oil is not thoroughly cracked and fixed at low temperatures, and an increased per cent of it is converted into tar, while at excessively high temperatures the oil is said to be overcracked, more gas being made per gallon of oil used, simultaneous with the production of more carbon and less tar.

When oil is cracked, as in a water-gas set, the nature and amount of the different products formed depend to a considerable extent on the atmospheres in which the oil is cracked. In some atmospheres the per cent of tar formed is increased and in others it is decreased. For example, when oil is cracked in a blue gas containing excess steam, the production of tar and gas is increased, while the amount of carbon formed is decreased.

The time and the intimacy of contact likewise affect the quantity and nature of the products resulting from oil cracking. An increase in the total time that the oil vapors are in contact with the hot

brick tends to decrease the per cent of tar formed and to increase the per cent of carbon.

The method and time of injecting the oil into the carburetor governs the concentration of the oil vapors and the time of contact of the oil vapors with the bricks, and therefore directly affects the amount of tar formed. If the oil is injected too rapidly to be properly an excessive formation of tar results. Even though the rate of oil input is suitable, an inefficient oil spray may result in so poor a distribution of oil in the carburetor that the oil is not properly cracked. Under such circumstances an increased amount of tar would be produced.

Different oils, when subjected to the same heat temperature in the cracking process, produce different amounts of tar. One oil may produce 15 per cent of tar while another may produce 25 per cent under the same conditions.

The fuel used also regulates to some extent the amount of tar formed under a given set of conditions. This is true even if the effect on oil cracking caused by a difference in the quality of the gas produced is left out of consideration.

When coke is used as generator fuel, no appreciable amount of tar is produced during the steam run, but when a high volatile bituminous coal is used in place of coke, tar is produced in the generator. The green fuel on top of the fuel bed is being distilled during the run, the tarry matter passes on with the similar products from the oil cracking, and the result is an increase in the total amount of tar produced.

The quality of the tar produced is frequently of more importance than the quantity. This is apparent when consideration is given to the fact that sometimes a tar is produced with an exceedingly high free carbon and water content. Such a tar can not be readily separated from the water by ordinary settling and will not usually produce as valuable products as a tar with a low free carbon content.

A discussion of the influence of each of the variables affecting the quality and the value of water-gas tar will not be attempted in this paper, although the effect of a change from coke to coal fuel seems to be worthy of special mention.

It has been observed that the yield of tar figured as tar per gallon of oil used is considerably increased when a high-volatile bituminous coal is used in place of coke as generator fuel and the physical and chemical nature of the tar is materially altered. Enough information has not been obtained to say definitely how much tar can be produced when a particular coal is used. Obviously, the operating variables as well as the daily operating time influence the volume and properties of the tar. The indications are that the volume of tar may be increased as much as 25 per cent when a change to coal

fuel is made. Tar so produced is heavy and viscous, with the appearance of coal tar rather than of ordinary water-gas tar. The properties of water-gas tars are to be studied by one of the authors of this bulletin. A few analyses are given in Table 18, which follows.

Sample 1 was obtained when coal was used as fuel without the blow run. The temperatures in the carburetor and the superheater were excessive and it is likely that the high percentage of free carbon in the tar can be attributed to this cause.

Sample 2 was obtained with the blow run, the carburetor and superheater temperatures being normal.

TABLE 18.—*Differences in the composition of water-gas tars.*

| | Sample 1. | Sample 2. | Sample 3. |
|----------------------------------|------------------|------------------|-----------|
| Fuel used..... | Bituminous coal. | Bituminous coal. | Coke. |
| Specific gravity at 15.5° C..... | 1.17 | 1.16 | 1.065 |
| Free carbon.....per cent. | 9.14 | 4.85 | Trace. |
| Moisture.....do. | 8.40 | 3.00 | |

| Distillation of dry tar. | | | |
|----------------------------|----------------|-------|----------------|
| Up to 110° C.....per cent. | 0.0 | 0.0 | 0.2 |
| 110° to 170° C.....do. | .8 | .0 | 1.2 |
| 170° to 235° C.....do. | 4.3 | 1.6 | 15.7 |
| 235 to 270° C.....do. | 9.8 | 7.2 | 18.7 |
| 270 to 300° C.....do. | 15.3 | 9.1 | 20.6 |
| Residue.....do. | (brittle) 69.4 | 82.0 | (brittle) 38.8 |
| Undetermined.....do. | .4 | .1 | 4.8 |
| Total.....do. | 100.0 | 100.0 | 100.0 |
| Tar acids..... | Trace..... | 0.0 | |

The distillations were made in a 500-c. c. glass distillation flask, with the thermometer bulb at the offtake arm.

AMMONIA.

Ammonia is a by-product in coal-gas manufacture, but is not usually present to any extent in water gas. This is particularly true of water gas made from coke fuel. If bituminous coal is used in the generator, it may be anticipated that some ammonia would be formed during the run by the distillation of the green coal. Steam distillation under the prevailing temperature conditions might also be expected to augment ammonia production. However, consideration of the fact that only a small amount of fuel is consumed per 1,000 feet during the run leads to the conclusion that the quantity of ammonia produced would not be very great. A test for free ammonia (NH₃) in the blue gas sampled at the generator lid showed 45 grains of NH₃ per 100 cubic feet of blue gas made, an amount roughly equivalent to 36 grains of NH₃ per 100 cubic feet of finished gas. A calculation

on the basis of NH_3 per ton of coal used in the generator indicates that 2.6 pounds of NH_3 are obtained per ton of coal. Part of this ammonia is decomposed during the passage of the gas through the checker chambers. The gas at the outlet of the superheater, before the wash box, contained only 25 grains of NH_3 per 100 cubic feet of finished gas, the equivalent of 1.8 pounds of NH_3 per ton of coal consumed. The loss in passing through the checker chambers was 30.8 per cent.

The amount present in any particular plant with a given fuel and specified operating conditions can best be determined by analyses of each individual gas. The problem of recovery is an economic as well as a mechanical and chemical one. Fresh water was used in all the scrubbing apparatus and in the wash box during the course of the Streator experiments, and there was no attempt made to recover the ammonia produced.

DESIGN OF WATER-GAS APPARATUS.

Present-day water-gas apparatus has been designed for use with fuel that contains only a small percentage of volatile combustible and a high proportion of fixed carbon. Hence the same fuel and oil efficiencies would not be obtained with the same operating methods when high-volatile bituminous coals are substituted for high-carbon, low-volatile fuels.

Although an operating method has been worked out, as discussed in this bulletin, whereby some of the difficulties encountered when high-volatile fuels are used can be mitigated, the indications are that a set might be so designed that a greater oil and fuel economy could be obtained. A specially designed set of this type would be advantageous where local conditions necessitate the use of coal instead of coke as generator fuel. However, such a set has not yet been designed and its development and perfection are still problems for additional research.

ECONOMIC SIGNIFICANCE OF THE USE OF BITUMINOUS COAL AS GENERATOR FUEL.

Although a number of plants are now operating successfully with bituminous coal as generator fuel, some of them first resorted to this practice in anticipation of a war measure requiring them to do so. The substantial saving made with this fuel, which may be as high as 8 cents per 1,000 feet, is great enough to warrant other companies giving the matter serious consideration. The saving is due chiefly to the difference in price of coal and coke and to the decreased amount of oil used per 1,000 cubic feet of gas made. Since the prices of coal, coke, and oil are different in various parts of the country, the saving

to be realized can only be determined when these conditions are known. However, it is very generally true that coke costs considerably more per ton than coal, and in addition a good grade of coke is only obtainable at many plants by shipment from a considerable distance.

SULPHUR CONTENT AS A LIMITING FACTOR.

A large proportion of the total coal supply of the United States contains a rather high per cent of sulphur. When such coals are used as generator fuel, more sulphur passes into the unpurified gas and the amount of it seems to vary as the per cent of total sulphur in the coal varies. This means an added duty for the purifiers, with a resultant increase in the cost of the finished gas, if the sulphur is not recovered as a by-product. In some plants where the purifying capacity is limited an increased amount of sulphur in the raw gas offers a serious problem for the engineer. High-sulphur coals are very often lower in price delivered than low-sulphur coals and frequently have better gas-making properties as well. Although it is natural to specify low-sulphur coals from a desire to keep operating expenses as low as possible, the advantages to be gained by utilizing coals of higher sulphur content may sometimes be great enough to warrant increasing the purifying capacity.

USE OF COALS FROM OTHER DISTRICTS AS GENERATOR FUEL.

The authors' experiments were limited to Central District coals, therefore figures can not be given for the economies obtainable with other high-volatile coals. Many of them would, no doubt, answer equally well for this purpose. Some trouble may be anticipated, however, with coals that disintegrate when heated.

SUMMARY AND CONCLUSIONS.

The results at the Streator plant and in other plants where bituminous coal is in use as generator fuel seem to justify the following conclusions:

1. High volatile bituminous fuel can be used efficiently in the manufacture of water gas.
2. An oil economy can be realized when bituminous coal is used as generator fuel, due to the increased value of the blue gas occasioned by the presence of some of the volatile matter of the coal.
3. When coal fuel is used, it is not necessary to operate with a relatively cold generator, thereby materially reducing the make.
4. Operating methods can be so adjusted that the capacity when coal is used with the present apparatus can be made to approach that attained with a good grade of coke, and to equal or surpass that attained with an inferior coke. The blow-run method of operating is one example of such adjustment.

5. The reduced capacity frequently noted when bituminous coal (coking coal) is used in the generator is in part due to the resistance to the air blast occasioned chiefly by the coking and matting of the fuel.

6. The resistance to the passage of air through the fire is materially decreased by the uniform admixture of coke with the coal.

7. When mixtures of coal and coke containing more than 50 per cent of coal are used, there seems to be a definite mixture that is favorable for a maximum make. The amounts may vary with different coals, but for those used in the Streator experiments the most satisfactory mixture was 70 per cent coal and 30 per cent coke, without blow run.

8. When the blow run is employed with the 70-30 mixture of coal and coke, the make is further increased.

9. When the blow run is used with a fuel mixture containing 70 per cent coal and 30 per cent coke, the oil used per 1,000 feet was not noticeably increased over the amount used with 100 per cent coal.

10. The production of smoke which is associated by some operators with the use of bituminous coal, can be almost if not entirely eliminated by slight modification of the operating procedure.

11. Indications are that, if bituminous coal is used, with careful operation the amount of generator fuel per 1,000 cubic feet of gas made need not be very much greater than with coke fuel of approximately the same heating value. Plants are now operating consistently with a consumption of 37 to 40 pounds of Central District coal per 1,000 feet that previously used 33 to 37 pounds of coke to produce the same amount of gas.

12. The utilization of coal in place of coke as generator fuel, when shipment from distant points is to be avoided, conserves the transportation facilities of the country.

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DEPARTMENT OF THE INTERIOR

JOHN BARTON PAYNE, SECRETARY

BUREAU OF MINES

H. FOSTER BAIN, ACTING DIRECTOR

FLOTATION TESTS OF IDAHO ORES

BY

CLARENCE A. WRIGHT, JAMES G. PARMELEE
and JAMES T. NORTON

[This report was prepared in cooperation with the School of Mines,
University of Idaho, and the Idaho State Bureau of Mines and Geology]



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FLOTATION TESTS OF IDAHO ORES.

PART I.—TESTING COEUR D'ALENE LEAD-ZINC ORES FOR DIFFERENTIAL FLOTATION.

By CLARENCE A. WRIGHT and JAMES G. PARMELEE.

INTRODUCTION.

The object of this paper is to give to mining companies and to all others who are interested some idea of the possibilities in the treatment, by differential flotation, of lead-zinc ores of the Coeur d'Alene region and other districts. The purpose of these investigations is not to evolve a differential flotation process that would replace present methods of gravity concentration but rather one that could be applied to the relatively fine material and slime produced in the mills and treated unsuccessfully by gravity methods or by flotation. It is expected, of course, that the products to be obtained by differential flotation would be of much better grade than those now produced by flotation in the Coeur d'Alene mills.

In view of the fact that it has been difficult to obtain representative samples of flotation feed from the mills, owing to the necessity of shipping the material wet and free from oil or other foreign matter that might be deleterious to the differential separation of the sulphides, the writers have felt justified in running flotation tests with the mill feeds in order to ascertain whether a differential separation of the sulphides could be effected. However, two different lots of flotation feed were tested for differential flotation, and the results of tests of one of these are given in this report. It is hoped that it will be possible later to test out the flotation feed for differential flotation on a large scale in some of the mills.

This paper covers part of the investigations carried on by the United States Bureau of Mines in cooperation with the Idaho Bureau of Mines, the University of Idaho, and certain mining companies of the Coeur d'Alene region. Although the results of the tests included in this report are not to be considered final, they indicate possibilities and may suggest others leading to a solution of the problem of separating lead and zinc sulphides by differential flotation in the treatment of certain ores.

DEFINITIONS.

The following definitions of flotation have been adopted by the Bureau of Mines:

Flotation is the process or processes by which the valuable minerals in a mass of finely ground ore can be caused to float on a liquid into which the finely ground ore is fed.

Differential flotation is the flotation of one flotative mineral in the presence of another ordinarily flotative.

Preferential flotation is a special type of differential flotation in which a mixture of two flotative minerals is given a light roast in order that one may be oxidized while the other remains unchanged. Only the surface of one of the minerals is oxidized, but this suffices to keep it from floating.

ACKNOWLEDGMENTS.

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In addition, acknowledgment is made here to J. H. Jonté for the many chemical analyses of the ores and flotation products necessary for the compilation of material for this report, and to Martin S. Taylor for his assistance in running tests and performing many other duties in connection with the work.

LEAD-ZINC ORES OF THE COEUR D'ALENE REGION.

The chief metals produced from the ores of the Coeur d'Alene region are lead and zinc, with minor amounts of copper, silver, and antimony. Where the minerals are as finely disseminated as in the mines of the Pine Creek, Nine-mile, and other districts, it is almost impossible to make clean lead and zinc products by the usual methods of gravity concentration. Although at most mills a good percentage of lead and zinc is recovered in the concentrates from the jigs, fine grinding is essential to effect a reasonably clean separation of either mineral when as finely disseminated as in ores of this character. In fact, it has been found by examination under the micro-

scope that some of the lead and zinc particles are still mechanically combined in an ore that has been crushed to pass a 200-mesh screen. This fact is clearly indicated in the accompanying photomicrographs, which are described under each ore tested.

PRODUCTS OBTAINED BY PRESENT METHODS OF CONCENTRATION

The following examples will give some idea of the grade of lead-zinc ore treated and the products obtained by the present methods of concentration:

Assays of feed and concentrates.

| Ore 1. | Assays. | | | | | | |
|------------------------|------------------|------------------|------------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Ag. | Fe. | Mn. | S. | Insol. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Ounces per ton.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Feed to mill..... | 11.4 | 16.9 | 3.6 | 6.1 | 9.2 | 14.0 | 50.0 |
| Zinc concentrates..... | 15.4 | 33.4 | 5.0 | 9.3 | .3 | 23.1 | 16.6 |

| Ore 2. | Assays. | | |
|------------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Feed to mill..... | 3.0 | 15.0 | 8.0 |
| Lead concentrates..... | 45.0 | 18.0 | 15.0 |
| Zinc concentrates..... | 9.0 | 34.5 | 11.0 |

| Ore 3. | Assays. | | | |
|--|------------------|------------------|------------------|------------------------|
| | Pb. | Zn. | Fe. | Ag. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Ounces per ton.</i> |
| Feed to mill..... | 6.0 | 12.0 | 7.0 | 2.0 |
| Lead concentrates from mill..... | 50.0 | 12.0 | 7.0 | 16.0 |
| Zinc concentrates from mill..... | 14.0 | 35.0 | 5.0 | 5.0 |
| Lead concentrates from tables..... | 68.0 | 8.0 | 7.0 | 18.0 |
| Zinc concentrates from tables..... | 9.0 | 34.0 | 10.0 | 3.0 |
| Lead product from retreatment of flotation concentrates on tables..... | 65.0 | 10.0 | 7.0 | 15.0 |
| Zinc product from retreatment of flotation concentrates on tables..... | 12.0 | 43.0 | 5.0 | 4.0 |

| Ore 4. | Assays. | | |
|---------------------------------------|------------------|------------------|------------------------|
| | Pb. | Zn. | Ag. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Ounces per ton.</i> |
| Feed to mill..... | 5.1 | 5.8 | 1.9 |
| Lead concentrates from jig..... | 49.5 | 10.0 | 12.8 |
| Lead concentrates from tables..... | 56.6 | 11.2 | 9.7 |
| Zinc concentrates from tables..... | 11.7 | 23.4 | 8.6 |
| Feed to flotation cells..... | 2.7 | 5.7 | 1.8 |
| Zinc concentrates from flotation..... | 15.6 | 25.4 | 7.8 |

From the above examples it is evident that considerable zinc is lost in the lead concentrates and that the zinc concentrates carry a small amount of lead. The question arises, therefore, whether or

not a cleaner separation of the minerals is commercially feasible. On the assumption that cleaner products are possible, it is necessary to determine what step should be taken to effect a better separation, whether (1) by changing and improving the present methods, as by grinding the ore finer, which would necessitate more tables, possibly a smaller number of jigs, and a somewhat larger flotation plant; or (2) by combining the first improvement with differential flotation; or (3) by giving the concentrates a flash roast followed by preferential flotation; or (4) by some hydrometallurgical process.

In order to avoid any radical changes in the present methods and equipment in the mills, it seemed advisable to experiment along the first and second lines suggested, namely, to effect a better separation of the lead and zinc minerals by differential flotation, with whatever changes in the present treatment of the ores this might necessitate, should differential flotation prove successful. It is in that direction, therefore, that the Bureau of Mines, in cooperation with the University of Idaho, the Idaho Bureau of Mines and Geology, and certain mining companies of the Coeur d'Alene region, has done most of its experimenting in order to find some method of effecting a better commercial separation. From a number of tests run on various ores to date, interesting and encouraging results have been obtained.

TESTING THE LEAD-ZINC ORES FOR DIFFERENTIAL FLOTATION.

In the experiments conducted under the direction of the United States Bureau of Mines in the University of Idaho School of Mines laboratories, the principal object in testing an ore for differential flotation was to ascertain what flotative mixture would effect the best differential separation as regards commercial practicality. Prior to the actual testing of any ore for differential flotation, however, certain preliminary tests were made that have an important bearing on subsequent treatment. This preliminary work comprised a physical examination of the ore, a screen analysis after the ore had been crushed to $\frac{1}{4}$ -inch size or less, and an examination under a hand lens or a binocular microscope of the finer sizes of ore to ascertain the fineness of crushing necessary to liberate most of the mineral particles. These physical tests or examinations were followed by chemical analyses of the ore and of the products obtained from the screen analyses. The more important points brought out by the preliminary examination of the ore, considered with reference to subsequent flotation testing, were the necessary degree of fineness in grinding and the chemical composition of the ore with reference to the minerals sought and the gangue constituents present.

USE OF OILS AND CHEMICALS.

After the preliminary observations were made, actual flotation tests followed. The method of procedure was first to run a series of tests with each ore, using different oils alone to determine the relative value of each as to frothing and collecting properties, and its ability to effect a mineral selection. The procedure was then repeated with a number of the more common chemicals alone. From the results obtained in these two series of tests it was ascertained what possible combinations of oils and chemicals might be expected to effect a high recovery and a good selection of mineral with the particular ore in question. Finally, many combinations of the selected oils and chemicals were tried, followed by tests using different quantities of the oils and chemicals. It should be mentioned here that many of the derivatives of both mineral and vegetable oils, such as phenol, anthracene, eugenol, and guaiacol, were also tried. Some of the oil derivatives gave good results, whereas others were more or less inactive as far as concerns differential selection and recovery.

A mixture of coal-tar creosote, alcohol, and sodium hydroxide effected a fairly good differential separation of the lead and zinc sulphides in one ore, with a fair recovery. Distillates drawn off at different temperatures from mixtures of many coal tars and alcohol were used, but the results were not much more satisfactory than with "straight" mixtures of coal tar and alcohol.

The use of alcohol and coal-tar creosote with sodium hydroxide has been mentioned in an article by W. L. Zeigler.¹ Unfortunately this mixture did not seem to be applicable to all lead-zinc ores with equally good results, as actual tests have shown. In fact, the outstanding feature of the many tests run on different ores was that each ore seemed to require a somewhat different flotative mixture and that the effect of a certain oil or flotation agent on one ore may be different on another similar ore. This is mentioned to obviate any tendency to drawing the conclusion that a flotation mixture used with one ore will produce like results with another ore of similar mineral content. It is sometimes possible, however, to remove or at least to reduce the detrimental effect of soluble compounds present by washing the ore prior to flotation.

Mixtures of gasoline and coal tars have also been tested, giving a good selection of lead with rather low recoveries. A mixture of gasoline and coal tar with a small amount of cresylic acid has been used with fair success on a commercial scale in differential flotation of a lead-zinc ore.

¹ Zeigler, W. L., Differential flotation of lead and zinc: Eng. and Min. Jour., vol. 106, Apr. 20, 1918, pp. 741-742.

The reader will note that 20 pounds of chemical have been used in some of the tests described in this paper. This quantity is employed in the preliminary tests solely to determine the effectiveness of the chemical in producing a differential selection of mineral. In larger scale work the amount of chemical or chemicals can be considerably reduced, as proved by actual tests.

OPERATING CONDITIONS.

Nearly all the flotation experiments were made with mechanical-agitation machines of the Federal-Varley type (fig. 1) with ore that was crushed dry. In testing the ores included in this report, it was found that a better recovery was possible with an ore that had been crushed wet; hence results from tests which used ore that had been crushed dry might have been improved by wet crushing.

The following table, giving comparative results of tests, emphasizes this more clearly.

TABLE 1.—*Comparison of results obtained in the treatment of lead-zinc ore after dry and wet crushing.*

[Feed: Pb, 11.4 per cent; Zn, 3.9 per cent; Fe, 11.7 per cent.]

DRY CRUSHING.

| Flotation agents. | Pounds
per
ton. | Grade of product. | | | Recoveries. | | |
|--|-----------------------|-------------------|------------------|------------------|------------------|------------------|------------------|
| | | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| Distillate from mixture of coal tar and alcohol (1:4)..... | 1.0 | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Sodium hydroxide..... | 1.0 | 53.7 | 10.9 | 4.9 | 72.0 | 48.0 | 6.3 |
| Mixture of coal-tar creosote and alcohol (1:4)..... | 1.0 | | | | | | |
| Sodium hydroxide..... | 1.0 | 50.2 | 8.7 | 6.0 | 75.4 | 48.4 | 8.7 |
| Mixture of coal tar and alcohol (1:4)..... | 1.0 | | | | | | |
| Sodium hydroxide..... | 1.0 | 49.9 | 9.0 | 6.2 | 70.0 | 41.7 | 8.3 |
| Coal-tar creosote..... | .2 | | | | | | |
| Wood creosote..... | .3 | | | | | | |
| Sulphuric acid..... | 10.8 | 36.1 | 8.4 | 9.5 | 80.0 | 60.0 | 20.0 |

WET CRUSHING.

| | | | | | | | |
|--|------|------|------|-----|------|------|------|
| Distillate from mixture of coal tar and alcohol (1:4)..... | 1.0 | | | | | | |
| Sodium hydroxide..... | 1.0 | 54.5 | 9.1 | 6.3 | 90.6 | 46.5 | 9.5 |
| Mixture of coal tar creosote and alcohol (1:4)..... | 1.0 | | | | | | |
| Sodium hydroxide..... | 1.0 | 54.7 | 9.1 | 6.0 | 88.0 | 45.0 | 8.7 |
| Mixture of coal tar and alcohol (1:4)..... | 1.0 | | | | | | |
| Sodium hydroxide..... | 1.0 | 49.0 | 10.4 | 7.7 | 91.0 | 59.5 | 12.9 |
| Coal-tar creosote..... | .2 | | | | | | |
| Wood creosote..... | .3 | | | | | | |
| Sulphuric acid..... | 10.8 | 39.4 | 11.5 | 9.9 | 95.2 | 84.0 | 22.0 |

All preliminary tests were run in a small single-cell machine (fig. 1), using 1,000 grams of ore for each test. When a fair selection and a fair recovery were obtained from a preliminary test, the experiment was repeated in a larger machine, using 4,000 to 5,000 grams of ore.

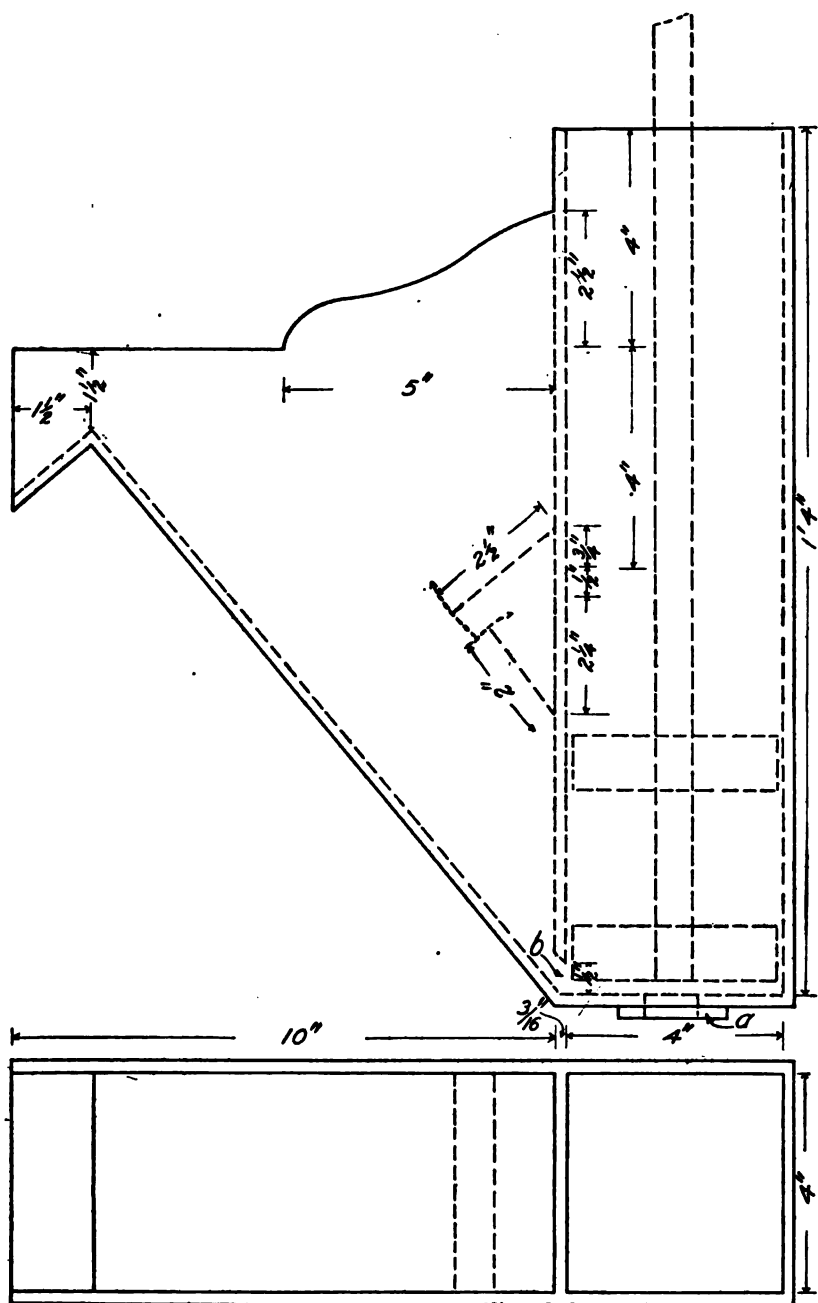


FIG. 1.—Experimental Flotation Machine. *a*, 2-inch by 1/4-inch brass ring bored for 1-inch plug; *b*, slot. To be constructed of sheet lead 1/4 inch thick. All joints to be water-tight and finished smooth on both sides.

Although practically all of the flotation tests described were made with mechanical-agitation machines, the type of machine utilized may have an important bearing on the differential separation of minerals. In fact, some modification of any of the existing types of machines might help to effect better differential separations than are at present possible. A study of the mechanical features and design of the machines is well worthy of consideration.

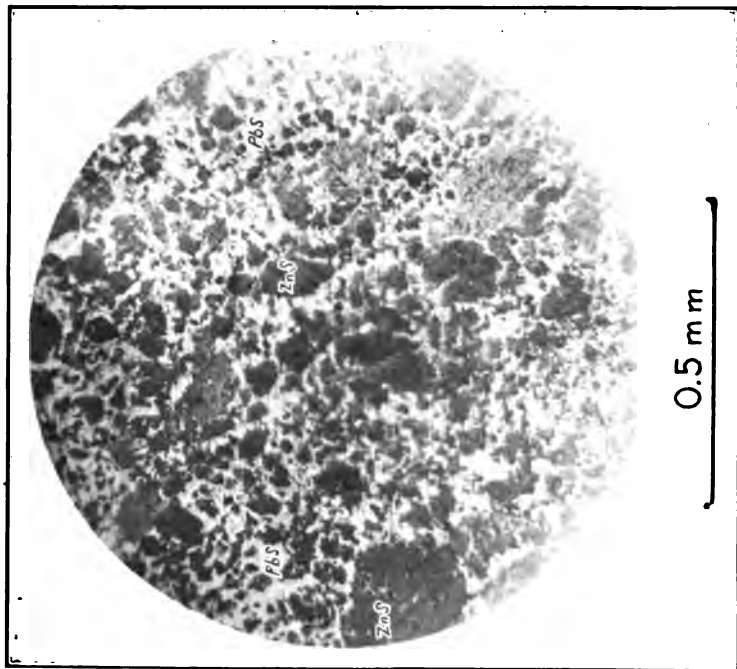
In running the tests it was found that a pulp density of about $2\frac{1}{2}$:1 to $3\frac{1}{2}$:1 (water to solids) seemed to give the best differential selection and mineral recovery at normal temperature. The pulp and flotative mixtures were preagitated from three to five minutes at a pulp density of about 1:1, after which the water level in the cell was raised and the agitation continued for about 10 minutes. It had been found that the bulk of the lead sulphide concentrate was drawn off during the first five minutes, but the agitation was continued in order that the time factor might be the same for each test. All products were obtained direct from a single cell without any attempt to clean them.

TESTING ORE NO. 1 FOR DIFFERENTIAL FLOTATION.

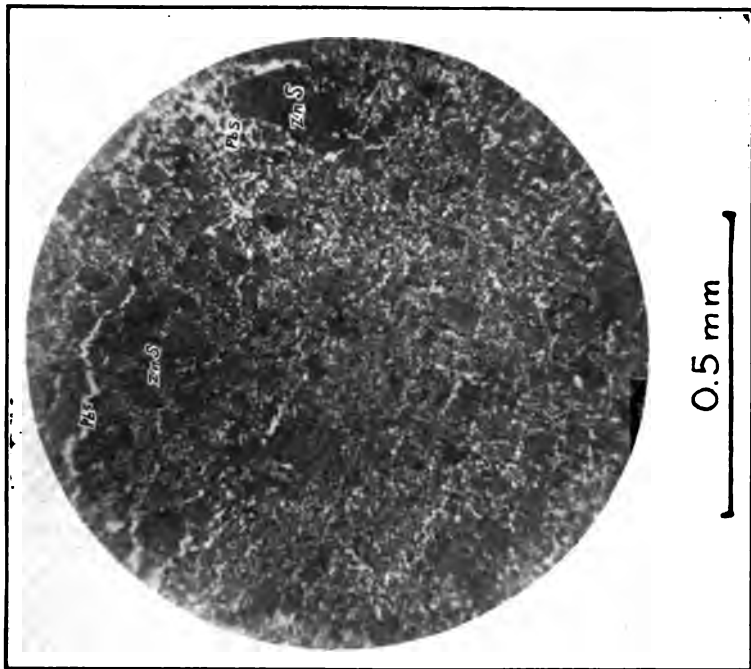
DESCRIPTION OF ORE.

The lead-zinc ore represented by No. 1 and No. 1, A, contained galena, sphalerite, small quantities of pyrites and chalcopyrite, included in a siderite-quartzite gangue. The lead and zinc minerals did not occur free in large pieces but were more or less finely disseminated. The sphalerite was light brown, quite different from the red and almost black varieties found in other mines of the district. Particles of supposedly clean galena often showed inclusions of quartz, sphalerite, and siderite when examined under a binocular microscope.

Polished sections of this ore, examined under the microscope, indicated that it was a typical example of one of the complex associations of lead and zinc sulphides whose separation and concentration present one of the many problems of ore treatment. A typical section of the ore is represented in the accompanying photomicrograph, Plate I, A. In the samples examined the lead sulphide occurred in two distinct combinations with reference to its relation to the zinc blende and the gangue. In one, represented by Plate I, B, a photomicrograph, the lead appeared in minute veins cutting the gangue in all directions and forming a lacelike network of mineral throughout portions of the siliceous ground mass. The galena occurring in this manner, although rather finely disseminated through the gangue, was sufficiently separated by fine grinding to be amenable to recovery by flotation, even though many of the grains were not



4. POLISHED SECTION OF ORE 1, SHOWING INTIMATE ASSOCIATION OF SPHALERITE AND GALENA.



5. POLISHED SECTION OF ORE 1, SHOWING MINUTE VEINS OF GALENA CUTTING THE GANGUE IN ALL DIRECTIONS.

entirely freed. In the other occurrence, the galena was intimately associated with sphalerite and could not be sufficiently freed by fine grinding, so that both the lead and zinc concentrates contained grains of locked sulphides of lead and zinc.

A careful study of briquettes formed of samples of the concentrates and of sealing wax indicated that where the zinc content predominated over the lead the grain went in the zinc concentrate, and, likewise, when galena formed the greater portion of the locked sulphide grain it went with the lead concentrate. It was evident that with a locked grain composed of equal proportions of the two minerals the action might be erratic, and it was reasonable to presume that such grains were as likely to go with the lead concentrate as with the zinc. Plate II, A (p. 24), is a photomicrograph of a grain of sphalerite in a sealing wax briquette showing included galena. Some of the minus 200-mesh material showed the same relation, and it also obtained in the lead concentrate grains containing inclusions of sphalerite.

From the observations above it is reasonable to conclude that good recoveries are possible if the ore is ground fine. It is also obvious that the zinc and lead sulphides remaining locked after fine grinding are in a class probably beyond the limits of separation by mechanical concentration.

The mill feed, represented by ore 1, contained about 7 per cent lead and from 5 to 6 per cent zinc, whereas the flotation feed from the mill, represented by ore 1A, contained about 5 per cent lead and 4 per cent zinc.

A chemical analysis of ore 1 or mill feed is as follows:

Chemical analysis of ore 1.

| | Per cent. | | Per cent. |
|---------|-----------|--------------------------------------|-----------|
| Pb..... | 7.4 | CaO..... | 3.1 |
| Zn..... | 5.8 | MgO..... | .4 |
| Cu..... | Tr. | MnO..... | 2.3 |
| Fe..... | 17.3 | SiO ₂ | 30.5 |
| S..... | 4.5 | Al ₂ O ₃ | 4.2 |

SCREEN ANALYSIS OF ORE 1.

After the sample was crushed by a small Blake crusher to about one-fourth-inch size, a screen analysis was made in a Tyler Ro-Tap shaking machine and then the different screen products were examined to ascertain the fineness of crushing necessary for flotation testing.

The screen analysis and examination of the screen products follow:

TABLE 2.—Screen analysis of ore 1.

[Head¹ contained Pb, 7.4 per cent; Zn, 5.8 per cent; Fe, 17.3 per cent.]

| Size of screen opening. | | | Weights. | | Assays (per cent). | | | Per cent of total head content. | | | Cumulative per cent of total head content. | |
|-----------------------------------|---------------|-------|-----------|----------------------|--------------------|-------|-------|---------------------------------|-------|-------|--|-------|
| Mesh. | Milli-meters. | Inch. | Per cent. | Cumulative per cent. | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. | Pb. | Zn. |
| On.....4 | 4.699 | 0.185 | 24.0 | 24.0 | 6.6 | 5.2 | 18.5 | 21.3 | 21.5 | 25.6 | | |
| 8 | 2.362 | .093 | 14.0 | 38.0 | 6.7 | 5.5 | 19.6 | 12.7 | 13.3 | 15.8 | 34.0 | 34.0 |
| 14 | 1.168 | .046 | 26.0 | 64.0 | 7.5 | 6.0 | 18.4 | 26.3 | 26.9 | 27.6 | 60.3 | 61.7 |
| 20 | .833 | .0328 | 6.0 | 70.0 | 7.1 | 6.6 | 17.8 | 5.7 | 6.7 | 6.2 | 66.0 | 68.4 |
| 35 | .417 | .0164 | 8.8 | 78.8 | 7.7 | 7.0 | 17.4 | 9.2 | 10.5 | 8.8 | 75.2 | 78.7 |
| 48 | .295 | .0116 | 3.5 | 82.3 | 7.8 | 7.1 | 16.8 | 3.6 | 4.3 | 3.4 | 78.8 | 82.1 |
| 65 | .208 | .0082 | 2.6 | 84.9 | 8.0 | 7.2 | 16.3 | 2.8 | 3.3 | 2.4 | 81.6 | 86.5 |
| 100 | .147 | .0058 | 3.0 | 87.9 | 8.6 | 6.9 | 15.6 | 2.5 | 3.4 | 2.7 | 85.1 | 90.0 |
| 150 | .104 | .0041 | 3.0 | 90.9 | 9.7 | 6.3 | 14.8 | 3.9 | 3.3 | 2.5 | 89.0 | 97.5 |
| 200 | .074 | .0029 | 1.6 | 92.5 | 11.1 | 7.2 | 13.5 | 2.4 | 1.9 | 1.2 | 91.4 | 98.7 |
| Through.....200 | .074 | .0029 | 7.2 | 99.7 | 10.1 | 4.5 | 12.4 | 9.9 | 5.5 | 5.1 | 101.3 | 100.0 |
| Loss and error by difference..... | | | +0.3 | | | | | -1.3 | -0.6 | -1.3 | | |
| Total..... | | | 100.0 | | | | | 100.0 | 100.0 | 100.0 | | |

¹ This term is used to denote sample tested.

EXAMINATION OF SCREEN PRODUCTS.

Product on 4-mesh screen.—A relatively small number of particles of seemingly clean galena, sphalerite, siderite, and quartzite were visible in this screen product, but most of the mineral particles present were mechanically combined.

Product on 8-mesh screen.—The galena appeared to be fine disseminated throughout the gangue, although it was also more or less mechanically combined with the sphalerite. Very little free galena or sphalerite was noted.

Products on 14, 20, and 35 mesh screens.—These screen products were similar to those on the 8-mesh screen, except that they seemed to carry more free mineral. Particles of galena and sphalerite were still mechanically combined with each other and with gangue particles.

Product on 48-mesh screen.—Although many mineral particles were still mechanically combined, considerable free galena seemed to be present.

Product on 65-mesh screen.—The material on a screen of this size still contained a quantity of mechanically combined products, but the combined mineral particles seemed to consist principally of galena and gangue particles. More free galena and sphalerite were present than in the preceding product.

Product on 100-mesh screen.—A large proportion of the product consisted of free mineral particles. Both the galena and sphalerite appeared to be free to a large extent, although included particles were still present. The material would seem to be suitable to flotation.

Product on 150 and 200 mesh screens.—This material was similar to the product on the 100-mesh screen. Particles of included galena, sphalerite, and gangue could still be seen.

Product through 200-mesh screen.—It was difficult to ascertain the presence of combined particles of galena and sphalerite in this fine material, although there is no doubt that such particles were present, as the photomicrograph shows, (see Plate II, A). A few particles of galena and gangue could, however, be seen in this screen product.

As a result of the examination of the various screen products, the ore was crushed to pass a 65-mesh screen. Practically all of the ore used in testing for differential flotation was crushed to this size or finer. A screen analysis of this material for flotation testing follows:

TABLE 3.—Screen analysis of material for flotation testing of ore 1.

[Head contained Pb, 7.4 per cent; Zn, 5.8 per cent; Fe, 17.3 per cent.]

| Size of screen opening. | | | Weights. | | Assays (per cent). | | | Per cent of total head content. | | | Cumulative per cent of total head content. | | |
|------------------------------|--------------|---------|-----------|----------------------|--------------------|-------|-------|---------------------------------|-------|-------|--|-------|-------|
| Mesh. | Millimeters. | Inches. | Per cent. | Cumulative per cent. | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| 100..... | 0.147 | 0.0068 | 11.4 | | 6.2 | 6.0 | 17.6 | 9.4 | 11.7 | 11.6 | | | |
| 150..... | .104 | .0041 | 26.0 | 36.4 | 6.1 | 5.9 | 17.8 | 20.5 | 25.5 | 25.7 | 29.9 | 37.0 | 37.3 |
| 200..... | .074 | .0029 | 7.4 | 43.8 | 5.7 | 5.8 | 17.2 | 5.7 | 7.2 | 7.4 | 35.6 | 44.4 | 44.7 |
| Through... 200 | .074 | .0029 | 55.6 | 99.4 | 8.8 | 5.3 | 16.0 | 66.2 | 60.7 | 51.6 | 101.8 | 95.1 | 96.3 |
| Loss or error by difference. | | | +0.6 | | | | | -1.8 | +4.9 | +3.7 | | | |
| Total..... | | | 100.0 | | | | | 100.0 | 100.0 | 100.0 | | | |

The analysis shows that 55 per cent of the material for flotation testing passed the 200-mesh screen and that this screen product contained 66 per cent of lead and 50 per cent of zinc.

RESULTS OF FLOTATION TESTS.

The results obtained by testing this lead-zinc ore for differential flotation did not indicate as high recoveries as were obtained with the flotation feed from the mill (ore 1, A). The results of tests given below have been selected from a number run on this ore. All tests were run without any attempt to clean the products.

Results of test 1.

[Head contained Pb, 7.4 per cent; Zn, 5.7 per cent; Fe, 17.3 per cent.]

| Grade of products. | Assay of products. | | | Recoveries. | | |
|-----------------------|--------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 57.6 | 7.4 | 5.5 | 68.5 | 11.4 | 2.8 |
| Zinc concentrate..... | 8.7 | 22.4 | 16.0 | 21.7 | 72.6 | 17.1 |
| Middling..... | 3.3 | 4.5 | 19.6 | 4.1 | 7.3 | 10.4 |
| Tailing..... | .9 | .6 | 18.9 | 7.7 | 6.7 | 69.3 |

Flotation agents.

| | Pounds
per ton. |
|---|--------------------|
| Soda ash..... | 5.0 |
| Charcoal..... | 1.0 |
| Barrett Co. No. 2 coal-tar creosote..... | .4 |
| Pensacola Tar & Turpentine Co. No. 350 crude wood pine oil..... | .2 |

Remarks.—Soda ash, charcoal, and coal-tar creosote were used for the lead concentrate, the pine oil being added to raise the zinc sulphide after the lead concentrate had been removed. The results showed a fair differential selection, especially as no attempt was made to clean the concentrates, but the recoveries of both the lead and the zinc were relatively low.

Results of test 12.

[Head contained Pb, 7.4 per cent; Zn, 5.7 per cent; Fe, 17.3 per cent.]

| Grade of products. | Assay of products. | | | Recoveries. | | |
|-----------------------|--------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 58.8 | 7.1 | 7.1 | 73.5 | 13.5 | 4.4 |
| Zinc concentrate..... | 4.4 | 26.9 | 15.1 | 7.2 | 67.5 | 12.4 |
| Middling..... | 1.5 | 5.1 | 23.1 | 1.3 | 6.7 | 10.2 |
| Tailing..... | 2.2 | 1.2 | 19.2 | 17.0 | 14.1 | 74.1 |

Flotation agents.

| | Pounds
per ton. |
|---|--------------------|
| Soda ash..... | 2.0 |
| Barrett Co. No. 2 coal-tar creosote..... | .4 |
| General Naval Stores Co. No. 5 s. d. pine oil..... | .1 |
| Copper sulphate..... | .5 |
| Pensacola Tar & Turpentine Co. No. 350 crude wood pine oil..... | .2 |

Remarks.—Of these flotation agents the first three were used to raise the lead and the last two to raise the zinc sulphide after the lead concentrate had been removed. The results indicated a fair differential selection of the two sulphides, as in test 1, but the recoveries of both lead and zinc were relatively low.

Results of test 41.

[Head contained Pb, 6.5 per cent; Zn, 5.3 per cent; Fe, 17.6 per cent.]

| Grade of products. | Assay of products. | | | Recoveries. | | |
|-----------------------|--------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 41.5 | 4.7 | 9.2 | 65.1 | 9.0 | 6.3 |
| Zinc concentrate..... | 7.7 | 25.8 | 14.5 | 18.0 | 74.0 | 12.5 |
| Middling..... | 5.5 | 7.7 | 21.0 | 4.8 | 8.3 | 6.8 |
| Tailing..... | .4 | .6 | 19.2 | 4.2 | 7.7 | 74.2 |

Flotation agents.

| | Pounds per ton. |
|---|-----------------|
| Barrett Co. (Vancouver) coal-tar creosote..... | 1.2 |
| General Naval Stores Co. No. 5 s. d. pine oil..... | .4 |
| Copper sulphate..... | 2.0 |
| Water glass, 40° B..... | 20.0 |
| Pensacola Tar & Turpentine Co. No. 350 crude wood pine oil..... | .4 |

Remarks.—In this test the pulp was pretreated with SO₂ gas, after which the first two flotation agents were added to raise the lead sulphide. After the removal of the lead concentrate the three remaining flotation agents were added to recover the zinc blende. The results indicated a differential selection, the SO₂ gas seeming to keep the zinc mineral down although the lead tenor of the lead concentrate was a bit low. Both the lead and zinc concentrates no doubt could have been improved by passing them through cleaner cells. Several tests were run with SO₂ gas with more or less favorable results.

Results of test 42.

[Head contained Pb, 6.5 per cent; Zn, 5.3 per cent; Fe, 17.5 per cent.]

| Grade of products. | Assay of products. | | | Recoveries. | | |
|-----------------------|--------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 38.9 | 2.0 | 9.9 | 68.8 | 4.3 | 6.5 |
| Tailing..... | 2.3 | 5.4 | 18.6 | 31.3 | 90.1 | 93.4 |

Flotation agents.

| | Pounds per ton. |
|--|-----------------|
| Vancouver coal-tar creosote..... | 0.6 |
| General Naval Stores Co. No. 5 s. d. pine oil..... | .4 |

Remarks.—As in test 41, pretreatment with SO₂ gas seemed to prevent the floating of the zinc sulphide. This fact was more marked here in that the zinc content of the lead concentrate was much lower. Although the grade of product was low, the lead tenor could have been raised by further treatment in cleaner flotation cells. This was one of several tests whose object was to get a lead concentrate low in zinc, no attempt being made to recover the zinc mineral after removal of the lead concentrate. The results of using SO₂ gas seemed to indicate possibilities in effecting a separation of the lead and zinc.

Results of test 63.

[Head contained Pb, 6.5 per cent; Zn, 5.3 per cent; Fe, 17.6 per cent.]

| Grade of products. | Assay of products. | | | Recoveries. | | |
|-----------------------|--------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 53.4 | 8.7 | 7.1 | 65.7 | 13.1 | 3.2 |
| Zinc concentrate..... | 10.5 | 33.2 | 10.6 | 19.7 | 76.4 | 7.3 |
| Middling..... | 4.8 | 4.6 | 23.2 | 8.1 | 9.6 | 14.5 |
| Tailing..... | .55 | .03 | 19.5 | 5.8 | .4 | 75.6 |

Flotation agents.

| | Pounds per ton. |
|---|-----------------|
| Soda ash..... | 4.0 |
| Barrett Co. Salt Lake heavy coal-tar creosote..... | .4 |
| General Naval Stores Co. No. 5 s. d. pine oil..... | .04 |
| Copper sulphate..... | 1.0 |
| Water glass, 40° B..... | 10.0 |
| Pensacola Tar & Turpentine Co. No. 350 crude wood pine oil..... | .2 |

Remarks.—The lead concentrate was obtained with the first three flotation agents, the copper sulphate and water glass being subsequently added to raise the zinc. Oil No. 350 was added after the zinc concentrate had been removed in order to produce the middling. Results indicated a fair differential selection of the sulphides. The grade of products could, no doubt, have been improved by cleaning.

TESTING ORE 1, A, FOR DIFFERENTIAL FLOTATION.**DESCRIPTION OF ORE.**

Ore 1, A, was the actual flotation feed from the mill after the mill feed (ore 1) had been treated by gravity concentration. Its chemical composition was very similar to that of the mill feed, although the lead and zinc content was a trifle lower, running about 5 and 4 per cent, respectively. A chemical analysis is as follows:

Chemical analysis of ore 1, A.

| Per cent. | | Per cent. | |
|-----------|------|--------------------------------------|------|
| Pb..... | 5.7 | CaO..... | 3.6 |
| Zn..... | 4.6 | MgO..... | .4 |
| Fe..... | 17.8 | MnO..... | 2.4 |
| Cu..... | Tr. | SiO ₂ | 30.8 |
| S..... | 3.8 | Al ₂ O ₃ | 4.7 |

SCREEN ANALYSIS OF ORE 1, A.

This flotation feed was found to be more readily amenable to differential flotation than the mill feed, especially as regards higher recoveries of both lead and zinc. These relatively higher recoveries, however, were probably due, in part at least, to the flotation feed being ground wet and considerably finer than the mill feed, which

was crushed to 65-mesh size. Referring to the screen analysis of the mill feed (ore 1) given in Table 3 (*see* p. 13), it will be seen that only 55 per cent passed the 200-mesh screen, whereas the screen analysis of the flotation feed, given below, shows that 81 per cent passed through the 200-mesh screen.

TABLE 4.—*Screen analysis of flotation feed from mill, ore 1, A.*

[Head contained Pb, 5.5 per cent; Zn, 4.3 per cent; Fe, 17.5 per cent.]

| Size of screen opening. | | | Weights. | | Assays (per cent). | | | Per cent of total head content. | | | Cumulative per cent of total head content. | | |
|----------------------------------|--------------|---------|-----------|----------------------|--------------------|-------|-------|---------------------------------|-------|-------|--|-------|-------|
| Mesh. | Millimeters. | Inches. | Per cent. | Cumulative per cent. | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| On..... 100 | 0.147 | 0.0058 | 1.5 | | 5.4 | 3.0 | 17.6 | 1.4 | 1.1 | 1.4 | | | |
| 150 | .104 | .0041 | 6.3 | 7.8 | 4.7 | 4.3 | 17.7 | 5.5 | 6.4 | 6.6 | 6.9 | 7.5 | 8.0 |
| 200 | .074 | .0029 | 10.7 | 18.5 | 4.3 | 4.0 | 17.1 | 8.2 | 9.9 | 10.6 | 15.1 | 17.4 | 18.6 |
| Through... 200 | .074 | .0029 | 81.0 | 99.5 | 5.5 | 4.5 | 17.1 | 80.9 | 84.9 | 79.1 | 96.0 | 102.3 | 97.7 |
| Loss or error by difference..... | | | .5 | | | | | +4.0 | -2.3 | +2.3 | | | |
| Total..... | | | 100.0 | | | | | 100.0 | 100.0 | 100.0 | | | |

The screen analysis shows that the material passing the 200-mesh screen contained practically 81 per cent of the total lead and 85 per cent of the total zinc.

EXAMINATION OF SCREEN PRODUCTS.

Product on 100-mesh screen.—When this screen product was examined free mineral particles could be seen, although there were also many included mineral particles, especially of galena and siderite. The principal minerals observed were galena, sphalerite, siderite, and quartz.

Products on 150 and 200 mesh screens.—These products appeared to be similar to those on the 100-mesh screens, although there seemed to be more free mineral. Galena particles combined with siderite and with quartz could still be seen.

Product through 200-mesh screen.—This product contained considerably more free mineral particles, although included mineral particles of galena and quartz were still present. Aside from the galena particles all other minerals appeared to be free.

RESULTS OF FLOTATION TESTS.

The results obtained in testing the flotation feed of ore 1, A, were in general better than those obtained when using the mill feed, ore 1, especially in regard to recoveries of the lead and zinc sulphides. As already stated, these relatively higher recoveries were probably

due to the fact that the material had been crushed wet and considerably finer than the mill feed and had not been permitted to dry before flotation. All products were obtained direct from a single lot without any attempt at cleaning. The results of tests given below have been selected from a number.

Results of test 12.

[Head contained Pb, 5.8 per cent; Zn, 4.6 per cent; Fe, 17.8 per cent.]

| Grade of products | Assay of products. | | | Recoveries. | | |
|-----------------------|--------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 46.0 | 10.0 | 8.8 | 81.2 | 22.5 | 70.8 |
| Zinc concentrate..... | 5.9 | 25.4 | 15.7 | 13.0 | 70.8 | 4.6 |
| Middling..... | 2.5 | 2.8 | 24.2 | 3.2 | 4.6 | 1.5 |
| Tailing..... | .4 | .1 | 19.5 | 4.7 | | |

Flotation agents.

Soda ash.....
 Charcoal.....
 Barrett Co. No. 2 coal-tar creosote.....
 General Naval Stores Co. No. 5 s. d. pine oil.....
 Copper sulphate.....
 Pensacola Tar & Turpentine Co. No. 350 crude wood pine oil.....

Remarks.—The first four flotation agents were used to raise the lead, the copper sulphate and No. 350 oil being added to raise the zinc blende after the lead concentrate had been removed. Without the copper sulphate, relatively low recoveries of lead were obtained. The grade of products could have been improved by cleaning, but the recoveries of both the lead and the zinc were fair.

Results of test 14.

[Head contained Pb, 5.8 per cent; Zn, 4.6 per cent; Fe, 17.8 per cent.]

| Grade of products. | Assay of products. | | | Recoveries. | | |
|-----------------------|--------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 50.5 | 8.2 | 7.8 | 82.3 | 16.8 | 78.6 |
| Zinc concentrate..... | 4.1 | 24.2 | 16.5 | 10.5 | 78.6 | 1.5 |
| Middling..... | 2.0 | .8 | 26.5 | 3.0 | 1.5 | .7 |
| Tailing..... | .15 | .05 | 19.2 | 1.7 | | |

Flotation agents.

Soda ash.....
 Charcoal.....
 Barrett Co. No. 2 coal-tar creosote.....
 General Naval Stores Co. No. 5 s. d. pine oil.....
 Copper sulphate.....
 Pensacola Tar & Turpentine Co. No. 350 crude wood pine oil.....

Remarks.—The grade of the lead product was better than in previous tests, probably because less No. 5 pine oil was used than in test 12. The recoveries of both minerals were fairly good and the grade of zinc product could have been improved by further treatment in cleaner cells. The soda ash, charcoal, Barrett No. 2, and G. N. S. No. 5 were used to raise the lead sulphide, whereas the copper sulphate and No. 350 oil were used for the zinc concentrate and middling.

Results of test 16.

[Head contained Pb, 5.8 per cent; Zn, 4.6 per cent; Fe, 17.8 per cent.]

| Grade of products. | Assay of products. | | | Recoveries. | | |
|-----------------------|--------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 52.1 | 9.3 | 7.5 | 81.2 | 18.3 | 3.8 |
| Zinc concentrate..... | 5.3 | 28.7 | 14.8 | 11.7 | 79.8 | 10.6 |
| Middling..... | 1.5 | 1.1 | 23.7 | 2.2 | 2.0 | 11.1 |
| Milling..... | .2 | .2 | 20.0 | 2.4 | 3.0 | 78.0 |

Flotation agents.

| | Pounds per ton. |
|---|-----------------|
| Soda ash..... | 2.0 |
| Barrett Co. No. 2 coal-tar creosote..... | .4 |
| General Naval Stores Co. No. 5 s. d. pine oil..... | .05 |
| Copper sulphate..... | 1.0 |
| Pensacola Tar & Turpentine Co. No. 350 crude wood pine oil..... | .2 |

Remarks.—This test was practically the same as test 14 except that no charcoal was used. The grade of lead product was fair, and a good grade of zinc product could have been obtained by further treatment in cleaner cells. The recoveries of lead and zinc were fairly good. Comparing this test with test 14 (*see p. 18*), there did not seem to be much difference in the grade of products with or without the addition of charcoal.

Results of test 18.

[Head contained Pb, 5.8 per cent; Zn, 4.6 per cent; Fe, 17.8 per cent.]

| Grade of products. | Assay of products. | | | Recoveries. | | |
|-----------------------|--------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 64.7 | 8.1 | 6.5 | 76.7 | 14.3 | 3.0 |
| Zinc concentrate..... | 6.4 | 29.3 | 13.4 | 14.1 | 81.0 | 9.6 |
| Middling..... | 1.8 | 1.0 | 26.9 | 3.2 | 2.3 | 15.5 |
| Milling..... | .4 | .1 | 19.2 | 4.7 | 1.5 | 73.5 |

Flotation agents.

| | Pounds per ton. |
|---|-----------------|
| Soda ash..... | 2.0 |
| Barrett Co. No. 2 coal-tar creosote..... | .4 |
| General Naval Stores Co. No. 5 s. d. pine oil..... | .1 |
| Copper sulphate..... | .5 |
| Imperial fuel oil..... | .4 |
| Pensacola Tar & Turpentine Co. No. 350 crude wood pine oil..... | .1 |

Remarks.—Both products from this test were of fairly good grade, although an attempt was made to clean them. The first three flotation agents were used to produce the lead concentrate, the copper sulphate and imperial fuel oil being added later to recover the zinc sulphide. Oil No. 350 was used only for the middling.

Results of test 52.

[Head contained Pb, 5.5 per cent; Zn, 4.3 per cent; Fe, 17.5 per cent.]

| Grade of products. | Assay of products. | | | Recoveries. | | |
|-----------------------|--------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 45.3 | 8.7 | 6.6 | 84.3 | 20.7 | |
| Zinc concentrate..... | 4.8 | 36.7 | 12.5 | 8.1 | 79.0 | |
| Middling..... | 2.2 | .8 | 24.0 | 1.4 | .7 | |
| Tailing..... | .2 | .07 | 18.6 | 2.8 | 1.2 | |

Flotation agents.

Potash alum.....
 Barrett Co. No. 2 coal-tar creosote.....
 General Naval Stores Co. No. 5 s. d. pine oil.....
 Copper sulphate.....
 Pensacola Tar & Turpentine Co. No. 350 crude wood pine oil.....

Remarks.—The results indicated a good recovery of lead and a fair grade of product. The zinc product was of higher grade than previous tests showed, probably because of the alum in the pulp. Potash alum was used in place of the soda ash, although it seemed to give a fairly good differential selection of the sulphides. It was more expensive than other chemicals, and other alums did not produce as good results. The first three flotation agents were used to float the lead mineral and the copper sulphate, oil No. 350 being added after the lead concentrate had been removed.

Results of test 43.

[Head contained Pb, 5.5 per cent; Zn, 4.3 per cent; Fe, 17.5 per cent.]

| Grade of products. | Assay of products. | | | Recoveries. | | |
|-----------------------|--------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 44.6 | 8.2 | 9.0 | 79.5 | 18.7 | |
| Zinc concentrate..... | 6.4 | 26.8 | 14.1 | 14.6 | 78.4 | |
| Tailing..... | .6 | .1 | 19.6 | 8.5 | 1.8 | |

Flotation agents.

Soda ash.....
 Barrett Co. No. 2 coal-tar creosote.....
 Georgia Pine & Turpentine Co. No. 194 crude turpentine.....
 Copper sulphate.....

Remarks.—All of the flotation agents except the copper sulphate were used to float the lead mineral. Oil No. 194 was apparently as satisfactory as the General Naval Stores Co. No. 5 oil used in other tests. The copper sulphate was the only flotation agent used to float the zinc mineral after the lead concentrate had been removed. The grade of the products was fair and could readily have been improved by treatment in cleaner cells.

Remarks.—The grade of the lead product was better than in previous tests, probably because less No. 5 pine oil was used than in test 12. The recoveries of both minerals were fairly good and the grade of zinc product could have been improved by further treatment in cleaner cells. The soda ash, charcoal, Barrett No. 2, and G. N. S. Co. No. 5 were used to raise the lead sulphide, whereas the copper sulphate and No. 350 oil were used for the zinc concentrate and middling.

Results of test 16.

[Head contained Pb, 5.8 per cent; Zn, 4.6 per cent; Fe, 17.8 per cent.]

| Grade of products. | Assay of products. | | | Recoveries. | | |
|-----------------------|--------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 52.1 | 9.3 | 7.5 | 81.2 | 18.3 | 3.8 |
| Zinc concentrate..... | 5.3 | 28.7 | 14.8 | 11.7 | 79.8 | 10.6 |
| Middling..... | 1.5 | 1.1 | 23.7 | 2.2 | 2.0 | 11.1 |
| Tailing..... | .2 | .2 | 20.0 | 2.4 | 3.0 | 78.0 |

Flotation agents.

| | Pounds per ton. |
|---|-----------------|
| Soda ash..... | 2.0 |
| Barrett Co. No. 2 coal-tar creosote..... | .4 |
| General Naval Stores Co. No. 5 s. d. pine oil..... | .05 |
| Copper sulphate..... | 1.0 |
| Pensacola Tar & Turpentine Co. No. 350 crude wood pine oil..... | .2 |

Remarks.—This test was practically the same as test 14 except that no charcoal was used. The grade of lead product was fair, and a good grade of zinc product could have been obtained by further treatment in cleaner cells. The recoveries of lead and zinc were fairly good. Comparing this test with test 14 (*see p. 18*), there did not seem to be much difference in the grade of products with or without the addition of charcoal.

Results of test 18.

[Head contained Pb, 5.8 per cent; Zn, 4.6 per cent; Fe, 17.8 per cent.]

| Grade of products. | Assay of products. | | | Recoveries. | | |
|-----------------------|--------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 54.7 | 8.1 | 6.5 | 76.7 | 14.3 | 3.0 |
| Zinc concentrate..... | 6.4 | 28.3 | 13.4 | 14.1 | 81.0 | 9.6 |
| Middling..... | 1.8 | 1.0 | 26.9 | 3.2 | 2.3 | 15.5 |
| Tailing..... | .4 | .1 | 19.2 | 4.7 | 1.5 | 78.5 |

Flotation agents.

| | Pounds per ton. |
|---|-----------------|
| Soda ash..... | 2.0 |
| Barrett Co. No. 2 coal-tar creosote..... | .4 |
| General Naval Stores Co. No. 5 s. d. pine oil..... | .1 |
| Copper sulphate..... | .5 |
| Imperial fuel oil..... | .4 |
| Pensacola Tar & Turpentine Co. No. 350 crude wood pine oil..... | .1 |

due to the fact that the material had been crushed wet and considerably finer than the mill feed and had not been permitted to dry before flotation. All products were obtained direct from a single cell without any attempt at cleaning. The results of tests given below have been selected from a number.

Results of test 12.

[Head contained Pb, 5.8 per cent; Zn, 4.6 per cent; Fe, 17.8 per cent.]

| Grade of products | Assay of products. | | | Recoveries. | | |
|-----------------------|--------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 46.0 | 10.0 | 8.8 | 81.2 | 22.5 | 5.1 |
| Zinc concentrate..... | 5.9 | 25.4 | 15.7 | 13.0 | 70.8 | 11.3 |
| Middling..... | 2.5 | 2.8 | 24.2 | 3.2 | 4.6 | 10.2 |
| Tailing..... | .4 | .1 | 19.5 | 4.7 | 1.5 | 75.6 |

Flotation agents.

| | Pounds per ton. |
|---|-----------------|
| Soda ash..... | 2.0 |
| Charcoal..... | 1.0 |
| Barrett Co. No. 2 coal-tar creosote..... | .4 |
| General Naval Stores Co. No. 5 s. d. pine oil..... | .1 |
| Copper sulphate..... | 1.0 |
| Pensacola Tar & Turpentine Co. No. 350 crude wood pine oil..... | .2 |

Remarks.—The first four flotation agents were used to raise the lead, the copper sulphate and No. 350 oil being added to raise the zinc blende after the lead concentrate had been removed. Without the copper sulphate, relatively low recoveries of zinc were obtained. The grade of products could have been improved by cleaning, though the recoveries of both the lead and the zinc were fair

Results of test 14.

[Head contained Pb, 5.8 per cent; Zn, 4.6 per cent; Fe, 17.8 per cent.]

| Grade of products. | Assay of products. | | | Recoveries. | | |
|-----------------------|--------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 50.5 | 8.2 | 7.8 | 82.3 | 16.8 | 4.1 |
| Zinc concentrate..... | 4.1 | 24.2 | 16.5 | 10.5 | 78.6 | 13.8 |
| Middling..... | 2.0 | .8 | 26.5 | 3.0 | 1.5 | 12.8 |
| Tailing..... | .15 | .05 | 19.2 | 1.7 | .7 | 72.1 |

Flotation agents.

| | Pounds per ton. |
|---|-----------------|
| Soda ash..... | 2.0 |
| Charcoal..... | 1.0 |
| Barrett Co. No. 2 coal-tar creosote..... | .4 |
| General Naval Stores Co. No. 5 s. d. pine oil..... | .05 |
| Copper sulphate..... | 1.0 |
| Pensacola Tar & Turpentine Co. No. 350 crude wood pine oil..... | .2 |

Remarks.—The grade of the lead product was better than in previous tests, probably because less No. 5 pine oil was used than in test 12. The recoveries of both minerals were fairly good and the grade of zinc product could have been improved by further treatment in cleaner cells. The soda ash, charcoal, Barrett No. 2, and G. N. S. Co. No. 5 were used to raise the lead sulphide, whereas the copper sulphate and No. 350 oil were used for the zinc concentrate and middling.

Results of test 16.

[Head contained Pb, 5.8 per cent; Zn, 4.6 per cent; Fe, 17.8 per cent.]

| Grade of products. | Assay of products. | | | Recoveries. | | |
|-----------------------|--------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 52.1 | 9.3 | 7.5 | 81.2 | 18.3 | 3.8 |
| Zinc concentrate..... | 5.3 | 28.7 | 14.8 | 11.7 | 79.8 | 10.6 |
| Middling..... | 1.5 | 1.1 | 23.7 | 2.2 | 2.0 | 11.1 |
| Tailing..... | .2 | .2 | 20.0 | 2.4 | 3.0 | 78.0 |

Flotation agents.

| | Pounds per ton. |
|---|-----------------|
| Soda ash..... | 2.0 |
| Barrett Co. No. 2 coal-tar creosote..... | .4 |
| General Naval Stores Co. No. 5 s. d. pine oil..... | .05 |
| Copper sulphate..... | 1.0 |
| Pensacola Tar & Turpentine Co. No. 350 crude wood pine oil..... | .2 |

Remarks.—This test was practically the same as test 14 except that no charcoal was used. The grade of lead product was fair, and a good grade of zinc product could have been obtained by further treatment in cleaner cells. The recoveries of lead and zinc were fairly good. Comparing this test with test 14 (*see p. 18*), there did not seem to be much difference in the grade of products with or without the addition of charcoal.

Results of test 18.

[Head contained Pb, 5.8 per cent; Zn, 4.6 per cent; Fe, 17.8 per cent.]

| Grade of products. | Assay of products. | | | Recoveries. | | |
|-----------------------|--------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 54.7 | 8.1 | 6.5 | 76.7 | 14.3 | 3.0 |
| Zinc concentrate..... | 6.4 | 29.3 | 13.4 | 14.1 | 81.0 | 9.6 |
| Middling..... | 1.8 | 1.0 | 26.9 | 3.2 | 2.3 | 15.5 |
| Tailing..... | .4 | .1 | 19.2 | 4.7 | 1.5 | 78.5 |

Flotation agents.

| | Pounds per ton. |
|---|-----------------|
| Soda ash..... | 2.0 |
| Barrett Co. No. 2 coal-tar creosote..... | .4 |
| General Naval Stores Co. No. 5 s. d. pine oil..... | .1 |
| Copper sulphate..... | .5 |
| Imperial fuel oil..... | .4 |
| Pensacola Tar & Turpentine Co. No. 350 crude wood pine oil..... | .1 |

Remarks.—Both products from this test were of fairly good grade, although no attempt was made to clean them. The first three flotation agents were used to produce the lead concentrate, the copper sulphate and imperial fuel oil being added later to recover the zinc sulphide. Oil No. 350 was used only for the middling.

Results of test 32.

[Head contained Pb, 5.5 per cent; Zn, 4.3 per cent; Fe, 17.5 per cent.]

| Grade of products. | Assay of products. | | | Recoveries. | | |
|-----------------------|--------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 45.3 | 8.7 | 6.6 | 84.3 | 20.7 | 3.9 |
| Zinc concentrate..... | 4.8 | 36.7 | 12.5 | 8.1 | 79.0 | 6.6 |
| Middling..... | 2.2 | .8 | 24.0 | 1.4 | .7 | 4.8 |
| Tailing..... | .2 | .07 | 18.6 | 2.8 | 1.2 | 79.6 |

Flotation agents.

| | Pounds.
per ton |
|---|--------------------|
| Potash alum..... | 2.0 |
| Barrett Co. No. 2 coal-tar creosote..... | .4 |
| General Naval Stores Co. No. 5 s. d. pine oil..... | .1 |
| Copper sulphate..... | .5 |
| Pensacola Tar & Turpentine Co. No. 350 crude wood pine oil..... | .2 |

Remarks.—The results indicated a good recovery of lead and a fair grade lead product. The zinc product was of higher grade than previous tests showed, probably because of the alum in the pulp. Potash alum was used in place of the soda ash; although it seemed to give a fairly good differential selection of the sulphides, it was more expensive than other chemicals, and other alums did not produce as good results. The first three flotation agents were used to float the lead mineral and the copper sulphate, oil No. 350 being added after the lead concentrate had been removed.

Results of test 43.

[Head contained Pb, 5.5 per cent; Zn, 4.3 per cent; Fe, 17.5 per cent.]

| Grade of products. | Assay of products. | | | Recoveries. | | |
|-----------------------|--------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 44.6 | 8.2 | 9.0 | 79.5 | 18.7 | 5.0 |
| Zinc concentrate..... | 6.4 | 26.8 | 14.1 | 14.6 | 78.4 | 10.1 |
| Tailing..... | .6 | .1 | 19.6 | 8.5 | 1.8 | 86.9 |

Flotation agents.

| | Pounds.
per ton |
|---|--------------------|
| Soda ash..... | 2.0 |
| Barrett Co. No. 2 coal-tar creosote..... | .4 |
| Georgia Pine & Turpentine Co. No. 194 crude turpentine..... | .2 |
| Copper sulphate..... | .5 |

Remarks.—All of the flotation agents except the copper sulphate were used to raise the lead mineral. Oil No. 194 was apparently as satisfactory as the General Naval Stores Co. No. 5 oil used in other tests. The copper sulphate was the only flotation agent used to float the zinc mineral after the lead concentrate had been removed. The grade of the products was fair and could readily have been improved by treatment in cleaner cells.

Results of test 72.

[Head contained Pb, 3.4 per cent; Zn, 4.3 per cent; Fe, 17.5 per cent.]

| Grade of products. | Assay of products. | | | Recoveries. | | |
|-----------------------|--------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 45.8 | 8.0 | 8.2 | 75.7 | 10.6 | 2.7 |
| Zinc concentrate..... | 3.7 | 28.3 | 14.4 | 14.2 | 85.7 | 10.7 |
| Middling..... | 1.7 | 2.3 | 23.2 | 2.5 | 2.6 | 6.5 |
| Tailing..... | .2 | .08 | 18.3 | 4.5 | 1.4 | 79.8 |

Flotation agents.

| | Pounds
per ton. |
|---|--------------------|
| Soda ash..... | 2.0 |
| Mixture (1 to 4) of cresylic acid and alcohol..... | 1.0 |
| Copper sulphate..... | .5 |
| Pensacola Tar & Turpentine Co. No. 350 crude wood pine oil..... | .2 |

Remarks.—The mixture of cresylic acid and alcohol seemed to work fairly well with this ore in effecting a good lead product. The zinc concentrate was produced by the addition of the copper sulphate and 0.05 pound of No. 350 oil, the rest being used for the middling. The grade of the zinc product could, no doubt, have been improved by cleaning.

Results of test 73.

[Head contained Pb, 3.4 per cent; Zn, 4.3 per cent; Fe, 17.5 per cent.]

| Grade of products. | Assay of products. | | | Recoveries. | | |
|-----------------------|--------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 48.1 | 7.9 | 7.5 | 70.3 | 9.1 | 6.1 |
| Zinc concentrate..... | 5.7 | 38.1 | 10.4 | 16.6 | 88.2 | 5.9 |
| Middling..... | 2.9 | 2.1 | 26.5 | 6.8 | 3.9 | 12.0 |
| Tailing..... | .4 | .03 | 18.7 | 9.0 | .5 | 82.4 |

Flotation agents.

| | Pounds
per ton. |
|--|--------------------|
| Soda ash..... | 2.0 |
| Cresylic acid..... | .4 |
| Barrett Co. No. 2 coal-tar creosote..... | .4 |
| Copper sulphate..... | .5 |
| General Naval Stores Co. No. 5 s. d. pine oil..... | .1 |

Remarks.—Soda ash, cresylic acid, and Barrett No. 2 oil were used to float the lead mineral. The copper sulphate was the only flotation agent added to produce the zinc concentrate after the lead concentrate had been removed, oil No. 5 being used for the middling. Both lead and zinc products were good, the zinc recovered in the zinc concentrate being especially good.

Results of test 75.

[Head contained Pb, 3.4 per cent; Zn, 4.3 per cent; Fe, 17.5 per cent.]

| Grade of products. | Assay of products. | | | Recoveries. | | |
|---------------------------|--------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 56.4 | 9.5 | 5.0 | 61.9 | 8.2 | 1.0 |
| Lead concentrate (2)..... | 24.3 | 19.1 | 9.4 | 10.0 | 6.2 | .7 |
| Zinc concentrate..... | 4.4 | 35.2 | 11.3 | 12.7 | 81.2 | 6.4 |
| Middling..... | 3.8 | .3 | 27.0 | 6.6 | .4 | 9.1 |
| Tailing..... | .2 | .13 | 19.2 | 4.6 | 2.4 | 86.7 |

Flotation agents.

| | Pounds per ton. |
|---|-----------------|
| Soda ash..... | 2.0 |
| Mixture (1 to 4) of cresylic acid and alcohol..... | 1.0 |
| General Naval Stores Co. No. 5 s. d. pine oil..... | .2 |
| Copper sulphate..... | 1.0 |
| Pensacola Tar & Turpentine Co. No. 350 crude wood pine oil..... | .2 |

Remarks.—Lead concentrate (2) could have been either returned to the flotation feed or re-treated separately. This test was run with 4,000 grams of ore.

Results of test 80.

[Head contained Pb, 3.4 per cent; Zn, 4.3 per cent; Fe, 17.5 per cent.]

| Grade of products. | Assay of products. | | | Recoveries. | | |
|-----------------------|--------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 44.0 | 10.7 | 6.1 | 66.7 | 12.8 | 1.8 |
| Zinc concentrate..... | 5.3 | 40.8 | 8.3 | 13.7 | 83.4 | 4.2 |
| Middling..... | 4.0 | 2.3 | 23.4 | 9.4 | 4.3 | 10.8 |
| Tailing..... | .4 | .1 | 18.4 | 9.2 | 1.8 | 82.6 |

Flotation agents.

| | Pounds per ton. |
|---|-----------------|
| Soda ash..... | 2.0 |
| Barrett Co. No. 2 coal-tar creosote..... | .4 |
| General Naval Stores Co. No. 5 s. d. pine oil..... | .2 |
| Copper carbonate..... | 1.0 |
| Pensacola Tar & Turpentine Co. No. 350 crude wood pine oil..... | .2 |

Remarks.—Soda ash, Barrett No. 2, and G. N. S. Co. No. 5 were used to float the lead mineral. After the lead concentrate was removed, copper carbonate was the only flotation agent added to produce the zinc concentrate, oil No. 350 being added for the middling. The results indicated that copper carbonate can be used in the presence of soda ash to replace copper sulphate in floating the zinc mineral.

CONCLUSIONS FROM TESTS ON ORES 1 AND 1, A.

On the basis of the experimental work done on ores 1 and 1, A, the following conclusions seem justified:

1. A complete separation of the galena and sphalerite by gravity or by differential flotation is impossible, because of the intimate association of the sulphides and their dissemination through the gangue.

2. The ore must be crushed to 100-mesh size to effect a good separation of the lead and zinc sulphide minerals.

3. Lead and zinc of good grade and satisfactory recoveries of both minerals can be obtained by differential flotation with the proper flotation mixture, especially by a roughing and cleaning system.

4. Soda ash seems to be essential in effecting a lead product of good grade with this ore.

5. A coal-tar creosote, cresylic acid, or alcohol, with possibly a small amount of some pine oil or some mixture of any of these flotation agents, seems to effect a good grade lead product in the presence of soda ash, under the proper conditions.

6. This ore seems to be amenable to treatment by the Bradford SO_2 process in effecting a separation of the lead and zinc sulphides.

7. Copper sulphate or some other copper compound, such as copper carbonate (*see test 80, p. 22*), added to the pulp containing soda ash after the lead concentrate has been removed, seems to give the best results in floating the zinc mineral.

8. Water glass seems to increase the recovery of the zinc mineral, to a small extent, at least.

9. Better results are obtained with the flotation feed than with the mill feed, especially in effecting higher recoveries of both the lead and zinc sulphides. It must be noted, however, that these higher recoveries are probably due, in part at least, to the flotation feed having been ground wet and considerably finer than the mill feed.

TESTING ORE 2 FOR DIFFERENTIAL FLOTATION.

DESCRIPTION OF ORE.

Ore 2 was another lead-zinc ore containing a relatively high percentage of zinc blende, with galena and some pyrite, in a quartz-siderite gangue. Some of the ore is more or less coarsely disseminated, indicating the possibility of gravity concentration, at least on tables, but the sulphide minerals were intimately associated in much of the ore, which would require fine grinding to liberate them from the gangue and from each other. The zinc blende, or sphalerite, was reddish brown, quite different from the zinc mineral in ore 1.

Under the microscope the lead sulphide appeared to be closely associated with quartz and occurred as inclusions disseminated through the sphalerite. The photomicrograph given as Plate III, A,

shows inclusions of galena in sphalerite and its occurrence along a small fissure or quartz vein in the sphalerite. Examination of a polished section of a briquette of sealing wax and flotation tailing from this ore under relatively high-power ocular and objective showed that, although the lead and zinc minerals had been largely freed or separated from each other, a number of grains of sphalerite contained inclusions of galena, as is clearly brought out in Plate II, *B*, a photomicrograph of one of these grains. This grain or mineral particle would probably pass a 150-mesh screen.

Another mineral particle of sphalerite with inclusions of galena is shown in Plate II, *C*, a photomicrograph illustrating further the impracticability of attaining a complete separation of the two minerals by crushing them to this size. Plate II, *D*, is a photomicrograph of a grain or particle of galena including a smaller particle of sphalerite. This relation is exceptional but illustrates the possibilities of mineral combination in a complex ore.

The following analysis will give some idea of the chemical composition of this ore:

Chemical analysis of ore 2.

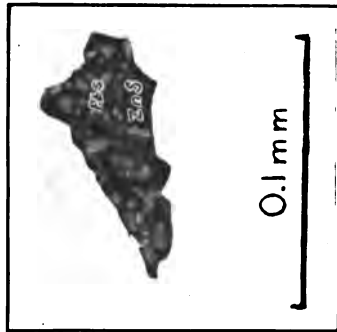
| | Per cent. | | Per cent. |
|---------|-----------|--------------------------------------|-----------|
| Pb..... | 8.0 | CaO..... | 1.03 |
| Zn..... | 24.4 | MgO..... | .25 |
| Fe..... | 7.0 | SiO ₂ | 30.8 |
| Cu..... | Tr. | Al ₂ O ₃ | 12.69 |
| S..... | 15.3 | | |

SCREEN ANALYSIS OF ORE 2.

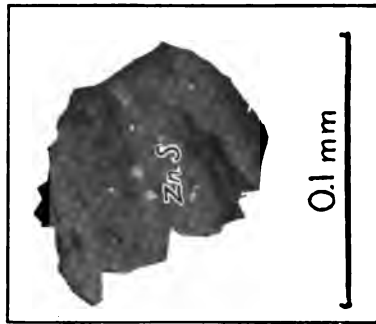
The ore was crushed, by a small Blake crusher, to about $\frac{1}{4}$ -inch size or less and a sample taken for screen analysis. The screen analysis and the results of examination of the screen products are as follows:



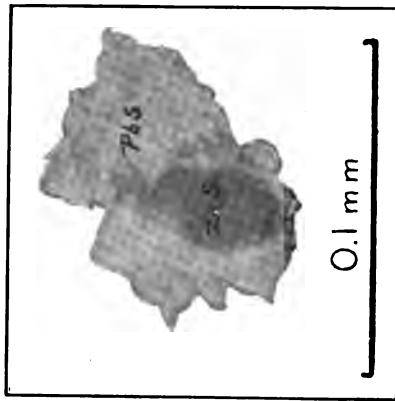
4. A GRAIN OF ORE 1, SHOWING INCLUSIONS OF GALENA IN SPHALERITE.



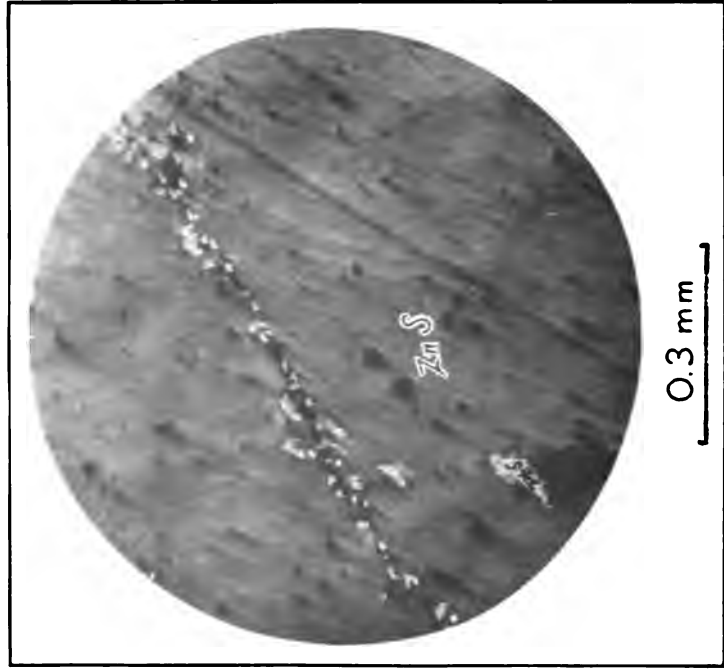
5. A PARTICLE OF SPHALERITE, ORE 2, WITH INCLUSIONS OF GALENA.



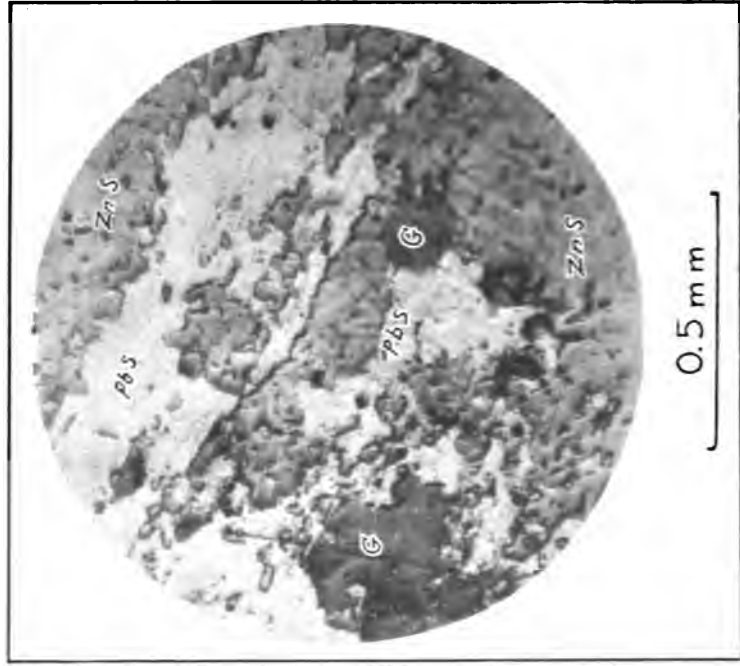
6. A SMALL GRAIN OF SPHALERITE, ORE 2, WITH INCLUDED GALENA.



7. SPHALERITE INCLUDED IN A SMALL GRAIN OF GALENA, ORE 2.



4. POLISHED SECTION OF ORE 2, SHOWING INCLUSIONS OF GALENA AND OCCURRENCE ALONG A SMALL FISSURE OR VEINLET IN SPHALERITE.



B. POLISHED SECTION SHOWING INTIMATE ASSOCIATION OF GALENA AND SPHALERITE IN ORE 3.

TABLE 5.—Screen analysis of ore 2.

[Head contained Pb, 7 per cent; Zn, 12 per cent; Fe, 6.1 per cent.]

| Size of screen opening. | | | Weights. | | Assays (per cent). | | | Per cent of total head content. | | | Cumulative per cent of total head content. | | |
|---------------------------------|---------------|---------|-----------|----------------------|--------------------|------|------|---------------------------------|-------|-------|--|------|-------|
| Mesh. | Milli-meters. | Inches. | Per cent. | Cumulative per cent. | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| On | 8 | 2.362 | 0.083 | | | | | | | | | | |
| 10 | 1.651 | .085 | 9.0 | | 5.5 | 3.8 | 5.9 | 7.1 | 2.6 | 8.7 | | | |
| 14 | 1.168 | .046 | 10.0 | 19.0 | 6.5 | 8.9 | 6.2 | 9.3 | 6.9 | 10.1 | 16.3 | 9.5 | 18.8 |
| 20 | .833 | .0328 | 12.7 | 31.7 | 6.0 | 11.1 | 6.1 | 10.8 | 10.9 | 12.6 | 27.2 | 20.6 | 31.4 |
| 28 | .599 | .0232 | 12.1 | 43.8 | 5.7 | 13.5 | 6.3 | 9.8 | 12.6 | 12.4 | 37.0 | 33.0 | 43.8 |
| 35 | .417 | .0164 | 10.5 | 54.3 | 6.1 | 15.7 | 6.6 | 9.1 | 12.8 | 11.3 | 46.1 | 45.8 | 55.1 |
| 48 | .295 | .0116 | 8.1 | 62.4 | 5.9 | 17.0 | 6.3 | 6.8 | 10.6 | 8.3 | 52.9 | 56.4 | 63.4 |
| 65 | .208 | .0082 | 6.8 | 69.2 | 6.2 | 17.3 | 6.1 | 6.0 | 9.1 | 6.7 | 58.9 | 65.5 | 70.1 |
| 100 | .147 | .0055 | 6.1 | 75.3 | 6.4 | 18.1 | 6.5 | 5.6 | 8.5 | 6.4 | 64.5 | 74.0 | 76.5 |
| 150 | .104 | .0041 | 4.8 | 80.1 | 7.0 | 19.8 | 6.4 | 4.8 | 7.3 | 5.1 | 69.3 | 81.3 | 81.6 |
| 200 | .074 | .0029 | 1.2 | 81.3 | 7.6 | 19.5 | 6.8 | 1.3 | 1.8 | 1.3 | 70.6 | 83.1 | 82.9 |
| Through .. | 200 | .074 | .0029 | 18.3 | 100.0 | 10.6 | 13.9 | 6.4 | 27.7 | 19.7 | 19.2 | 98.3 | 102.1 |
| Loss or error by difference.... | | | | | | | | | | | | | |
| Total..... | | | | | | | | | 100.0 | 100.0 | 100.0 | | |

EXAMINATION OF SCREEN PRODUCTS.

Products on 8 to 20 mesh screens.—When these screen products were examined it was noted that most of the minerals appeared to be mechanically combined, although some free sphalerite could be seen. The presence of free zinc mineral was probably due to the relatively high content of sphalerite.

Product on 28-mesh screen.—This screen product contained some free zinc blende. The galena and pyrite were combined with the zinc mineral and with the gangue.

Product on 35-mesh screen.—In this product there appeared to be considerably more free sphalerite and some free quartz. Some free galena could be seen.

Products on 48 and 65 mesh screens.—Much clean zinc mineral and quartz could be seen, with some free galena, although most of the galena and pyrite were mechanically combined with the zinc and quartz or with each other.

Product on 100-mesh screen.—Most of the sphalerite and quartz appeared to be free, although a small amount of the zinc mineral was mechanically combined with galena.

Product on 150-mesh screen.—Nearly all of the minerals, such as sphalerite, galena, pyrite, and quartz, appeared to be free, although some included particles of galena and sphalerite were no doubt present, as indicated by the photomicrographs. A few particles of galena combined with pyrite could also be seen.

Products on and through 200-mesh screen.—These screen products were similar to that on the 150-mesh screen, but considerably more pyrite and some chalcopyrite seemed to be present.

Examination of these screen products indicated it would be advisable to crush or grind the material to pass a 100-mesh screen in order to liberate most of the minerals. In all the tests with this ore, therefore, it was ground so that practically all would pass a 100-mesh screen. A screen analysis of the material used for flotation is as follows:

TABLE 6.—*Screen analysis of ore 2 for flotation testing.*

[Head contained Pb, 7 per cent; Zn, 12.3 per cent; Fe, 5.6 per cent.]

| Size of screen opening. | | | Weights. | | Assays (per cent). | | | Per cent of total head content. | | | Cumulative per cent of total head content. | | |
|----------------------------------|---------------|---------|-----------|----------------------|--------------------|-------|-------|---------------------------------|-------|-------|--|-------|-------|
| Mesh. | Milli-meters. | Inches. | Per cent. | Cumulative per cent. | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| On..... 100 | 0.147 | 0.0058 | 5.6 | | 6.6 | 16.2 | 6.7 | 5.1 | 7.3 | 6.8 | | | |
| 150 | .104 | .0041 | 16.0 | 21.6 | 6.3 | 15.2 | 6.1 | 14.3 | 19.7 | 17.5 | 19.4 | 27.0 | 24.3 |
| 200 | .074 | .0029 | 4.2 | 25.8 | 6.3 | 15.3 | 6.2 | 3.7 | 5.2 | 4.6 | 23.1 | 32.2 | 28.9 |
| Through... 200 | .074 | .0029 | 73.8 | 99.6 | 7.7 | 11.4 | 5.6 | 81.1 | 68.3 | 73.5 | 104.2 | 100.5 | 102.4 |
| Loss or error by difference..... | | | | | | | | -4.2 | -0.5 | -2.4 | | | |
| Total..... | | | 100.0 | | | | | 100.0 | 100.0 | 100.0 | | | |

The screen analysis shows that about 75 per cent of the material crushed for flotation testing passed the 200-mesh screen and that about 80 per cent of the lead value and nearly 70 per cent of the zinc value was contained in the 200-mesh product.

RESULTS OF FLOTATION TESTS.

When this ore was tested for differential flotation obtaining a high-grade lead product, a high recovery of the lead mineral was at first difficult because of the relatively high zinc content. With the proper flotation agents, however, it was later possible to obtain fairly satisfactory results, with good-grade products and excellent recoveries of both lead and zinc. Some results of the tests included here will indicate the possibilities in the treatment of this ore by differential flotation. It is, therefore, believed that the lead and zinc minerals in this ore could be separated commercially by flotation on a large scale. The following results of the tests have been selected from a number; no attempt was made to clean the products.

Results of test 104.

[Head contained Pb, 8 per cent; Zn, 24.4 per cent; Fe, 7 per cent.]

| Grade of product. | Assay of product. | | | Recoveries. | | |
|-----------------------|--------------------------|-------------------------|-------------------------|--------------------------|-------------------------|-------------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| Lead concentrate..... | <i>Per cent.</i>
57.4 | <i>Per cent.</i>
9.0 | <i>Per cent.</i>
3.0 | <i>Per cent.</i>
41.3 | <i>Per cent.</i>
2.1 | <i>Per cent.</i>
2.5 |

Flotation agent.

Eugenol..... Pounds
per ton.
0.13

Remarks.—Eugenol seemed to give a fair lead product, but was too expensive for commercial use. Guaiacol gives similar results and, although cheaper, is also too expensive.

Results of test 527.

[Head contained Pb, 9 per cent; Zn, 19 per cent; Fe, 7.2 per cent.]

| Grade of product. | Assay of product. | | | Recoveries. | | |
|-----------------------|--------------------------|-------------------------|-------------------------|--------------------------|-------------------------|--------------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| Lead concentrate..... | <i>Per cent.</i>
52.5 | <i>Per cent.</i>
8.7 | <i>Per cent.</i>
5.6 | <i>Per cent.</i>
79.3 | <i>Per cent.</i>
6.2 | <i>Per cent.</i>
10.6 |

Flotation agents.

Sodium phosphate..... Pounds
per ton.
20.0

Pensacola Tar & Turpentine Co. No. 400 wood creosote oil..... .1

Potassium acid sulphate..... 20.0

Remarks.—Both the grade of the lead product and the recovery of lead were fair. The flotation agents were too expensive for commercial use, especially when in the large quantities that would be needed.

Results of test 528.

[Head contained Pb, 9 per cent; Zn, 19 per cent; Fe, 7.2 per cent.]

| Grade of product. | Assay of product. | | | Recoveries. | | |
|-----------------------|--------------------------|-------------------------|-------------------------|--------------------------|-------------------------|-------------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| Lead concentrate..... | <i>Per cent.</i>
64.6 | <i>Per cent.</i>
8.5 | <i>Per cent.</i>
3.1 | <i>Per cent.</i>
62.4 | <i>Per cent.</i>
3.9 | <i>Per cent.</i>
3.7 |

Flotation agents.

Sodium phosphate..... Pounds
per ton.
20

Sodium thiosulphate..... 20

Remarks.—The lead product was of fair grade, but the recovery of lead was relatively low. The froth was thin but heavily mineralized. The cost of the flotation agents was high, but results showed a fair differential separation of the galena from the zinc blende and the possibility of flotation without oil.

Results of test 532.

[Head contained Pb, 9 per cent; Zn, 19 per cent; Fe, 7.2 per cent.]

| Grade of products. | Assay of products. | | | Recoveries. | | |
|-----------------------|--------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 56.3 | 9.2 | 3.8 | 70.7 | 5.5 | 6.0 |
| Tailing..... | 3.4 | 20.5 | 7.8 | 33.4 | 95.4 | 96.8 |

*Flotation agents.*Pounds
per ton.

| | |
|---|------|
| Sodium phosphate..... | 20.0 |
| Sodium thiosulphate..... | 20.0 |
| Mixture of 4 parts wood alcohol and 1 part coal-tar creosote..... | .5 |

Remarks.—Except for the addition of the oil mixture, this was run the same as test 328. As the results show, the oil mixture increased the recovery of lead but lowered the grade of the product. As in test 528, the chemical reagents were too expensive for commercial use.

Results of test 533.

[Head contained Pb, 9 per cent; Zn, 19 per cent; Fe, 7.2 per cent.]

| Grade of product. | Assay of product. | | | Recoveries. | | |
|-----------------------|-------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 43.2 | 14.9 | 5.0 | 76.8 | 12.5 | 11.1 |

*Flotation agents.*Pounds
per ton.

| | |
|---|-----|
| Sodium phosphate..... | 2.0 |
| Sodium thiosulphate..... | 2.0 |
| Pensacola Tar & Turpentine Co. No. 400 wood creosote oil..... | .1 |

Remarks.—The same chemical reagents were used as in the two preceding tests, Nos. 528 and 532, but the quantity was much less. Results indicated that the smaller quantity of the reagents permitted more zinc to be floated and therefore lowered the grade of the product. On the other hand, oil No. 400 seemed to increase the recovery of both the lead and the zinc, and its presence might counteract or modify somewhat the selective action of the reagents.

Results of test 537.

[Head contained Pb, 7 per cent; Zn, 12.9 per cent; Fe, 6.1 per cent.]

| Grade of product. | Assay of product. | | | Recoveries. | | |
|-----------------------|-------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 59.7 | 6.7 | 2.0 | 78.5 | 4.8 | 3.0 |

Flotation agents.

| | Pounds
per ton. |
|--------------------------|--------------------|
| Sodium phosphate..... | 10 |
| Sodium thiosulphate..... | 10 |

Remarks.—In this test the feed differed from that in previous tests, the lead and zinc content being lower. The results indicated a product of good grade and a good recovery of lead.

Results of test 554.

[Head contained Pb, 7 per cent; Zn, 12.9 per cent; Fe, 6.1 per cent.]

| Grade of products. | Assay of products. | | | Recoveries. | | |
|-----------------------|--------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 42.2 | 8.0 | 4.8 | 66.4 | 68.0 | 8.6 |
| Zinc concentrate..... | 7.3 | 34.7 | 9.6 | 31.0 | 80.0 | 46.7 |
| Tailing..... | .8 | 2.9 | 4.9 | 6.7 | 13.1 | 47.0 |

Flotation agents.

| | Pounds
per ton. |
|---|--------------------|
| Water glass (40° B.)..... | 10.0 |
| Barrett No. 2 coal-tar creosote..... | .2 |
| Pensacola Tar & Turpentine Co. No. 400 wood creosote oil..... | .1 |
| Sulphuric acid..... | 5.0 |
| Pensacola Tar & Turpentine Co. No. 350 crude wood pine oil..... | .4 |

Remarks.—The grade of both concentrates was fair, considering that they were rougher products. Cleaning would probably improve the grade considerably. The recovery of lead was somewhat low but the zinc recovery fairly good. Undoubtedly the quantities of flotation agents could be reduced in practice. The water glass and Barrett No. 2 oil were used to float the lead; the other flotation agents, the zinc.

Results of test 563.

[Head contained Pb, 7 per cent; Zn, 12.9 per cent; Fe, 6.1 per cent.]

| Grade of products. | Assay of products. | | | Recoveries. | | |
|-----------------------|--------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 47.5 | 9.3 | 4.0 | 76.0 | 8.1 | 7.4 |
| Zinc concentrate..... | 5.0 | 37.3 | 7.7 | 21.6 | 87.3 | 38.1 |
| Middling..... | 2.7 | 8.4 | 8.0 | 2.3 | 3.9 | 8.0 |
| Tailing..... | .5 | .3 | 5.4 | 3.7 | 1.2 | 46.3 |

Flotation agents.

| | Pounds
per ton. |
|---|--------------------|
| Soda ash..... | 5.0 |
| Barrett Co. No. 2 coal-tar creosote..... | .4 |
| Water glass (40° B.)..... | 10.0 |
| Copper sulphate..... | 2.0 |
| Pensacola Tar & Turpentine Co. No. 350 crude wood pine oil..... | .4 |

Remarks.—The recoveries of both lead and zinc were better than in the preceding test, No. 554. The grades of the rougher products could have been improved by cleaning. Soda ash and oil No. 2 were employed to float the lead, the other flotation agents being added after the lead concentrate had been removed.

Results of test 566.

[Head contained Pb, 7 per cent; Zn, 12.9 per cent; Fe, 6.1 per cent.]

| Grade of products. | Assay of products. | | | Recoveries. | | |
|-----------------------|--------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 53.5 | 5.1 | 6.3 | 76.7 | 3.9 | 10.3 |
| Zinc concentrate..... | 4.6 | 40.2 | 8.5 | 17.7 | 84.1 | 37.6 |
| Middling..... | 1.9 | 5.7 | 8.5 | 3.2 | 5.2 | 16.4 |
| Tailing..... | .2 | 1.4 | 4.7 | 1.4 | 5.5 | 39.0 |

Flotation agents.

| | Pounds per ton. |
|---|-----------------|
| Soda ash..... | 5.0 |
| Water glass (40° B.)..... | 20.0 |
| Barrett Co. No. 2 coal-tar creosote..... | .4 |
| Pensacola Tar & Turpentine Co. No. 400 wood creosote oil..... | .1 |
| Copper sulphate..... | 1.0 |
| Pensacola Tar & Turpentine Co. No. 350 crude wood pine oil..... | .2 |

Remarks.—The grade of products and recoveries was good. The soda ash, Barrett No. 2 oil, Pensacola Tar & Turpentine Co. No. 400, and half of the quantity of water glass given above were used to float the lead sulphide, the remainder being added to float the zinc mineral after the lead concentrate had been removed. The quantities of the flotation agents would have to be cut down in practice.

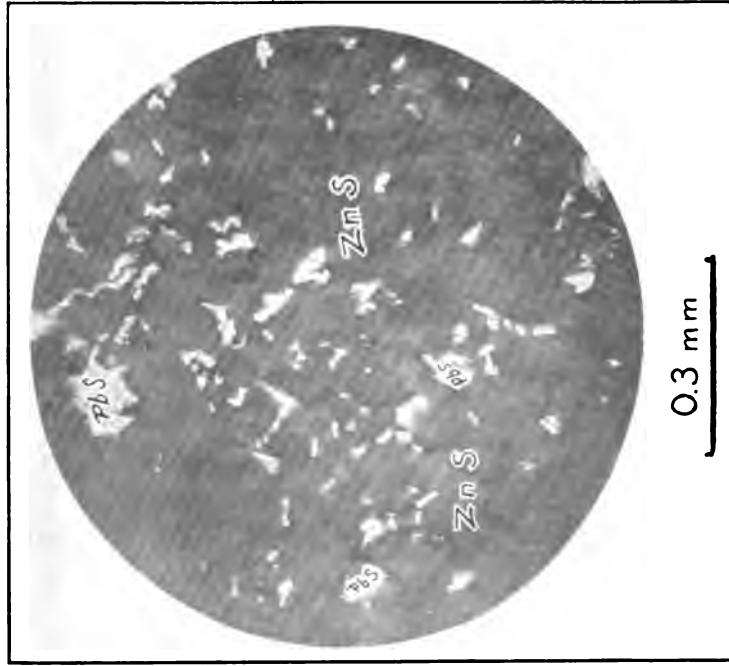
Results of test 567.

[Head contained Pb, 7 per cent; Zn, 12.9 per cent; Fe, 6.1 per cent.]

| Grade of products. | Assay of products. | | | Recoveries. | | |
|-----------------------|--------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 59.3 | 6.5 | 3.9 | 72.9 | 4.3 | 5.5 |
| Zinc concentrate..... | 4.8 | 39.2 | 8.5 | 20.3 | 90.1 | 41.2 |
| Middling..... | .7 | 1.4 | 9.3 | 1.1 | 1.2 | 16.8 |
| Tailing..... | 1.2 | .2 | 4.3 | 8.7 | .8 | 35.7 |

Flotation agents.

| | Pounds per ton. |
|---|-----------------|
| Soda ash..... | 10.0 |
| Barrett Co. No. 2 coal-tar creosote..... | 1.0 |
| Pensacola Tar & Turpentine Co. No. 400 wood creosote oil..... | .1 |
| Copper sulphate..... | 1.0 |
| Water glass (40° B.)..... | 10.0 |
| General Naval Stores Co. No. 5 s. d. pine oil..... | .1 |
| Pensacola Tar & Turpentine Co. No. 350 crude wood pine oil..... | .2 |



4. POLISHED SECTION SHOWING INCLUSIONS OF GALENA IN SPHALERITE, ORE 3.



5. A 48-MESH PARTICLE OF ORE 3, SHOWING ASSOCIATION OF GALENA AND SPHALERITE.

Remarks.—The recoveries of both lead and zinc were better than in the preceding test, No. 554. The grades of the rougher products could have been improved by cleaning. Soda ash and oil No. 2 were employed to float the lead, the other flotation agents being added after the lead concentrate had been removed.

Results of test 566.

[Head contained Pb, 7 per cent; Zn, 12.9 per cent; Fe, 6.1 per cent.]

| Grade of products. | Assay of products. | | | Recoveries. | | |
|-----------------------|--------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 53.5 | 5.1 | 6.3 | 76.7 | 3.9 | 10.3 |
| Zinc concentrate..... | 4.6 | 40.2 | 8.5 | 17.7 | 84.1 | 37.6 |
| Middling..... | 1.9 | 5.7 | 8.5 | 3.2 | 5.2 | 16.4 |
| Tailing..... | .2 | 1.4 | 4.7 | 1.4 | 5.5 | 39.0 |

Flotation agents.

| | Pounds per ton. |
|---|-----------------|
| Soda ash..... | 5.0 |
| Water glass (40° B.)..... | 20.0 |
| Barrett Co. No. 2 coal-tar creosote..... | .4 |
| Pensacola Tar & Turpentine Co. No. 400 wood creosote oil..... | .1 |
| Copper sulphate..... | 1.0 |
| Pensacola Tar & Turpentine Co. No. 350 crude wood pine oil..... | .2 |

Remarks.—The grade of products and recoveries was good. The soda ash, Barrett No. 2 oil, Pensacola Tar & Turpentine Co. No. 400, and half of the quantity of water glass given above were used to float the lead sulphide, the remainder being added to float the zinc mineral after the lead concentrate had been removed. The quantities of the flotation agents would have to be cut down in practice.

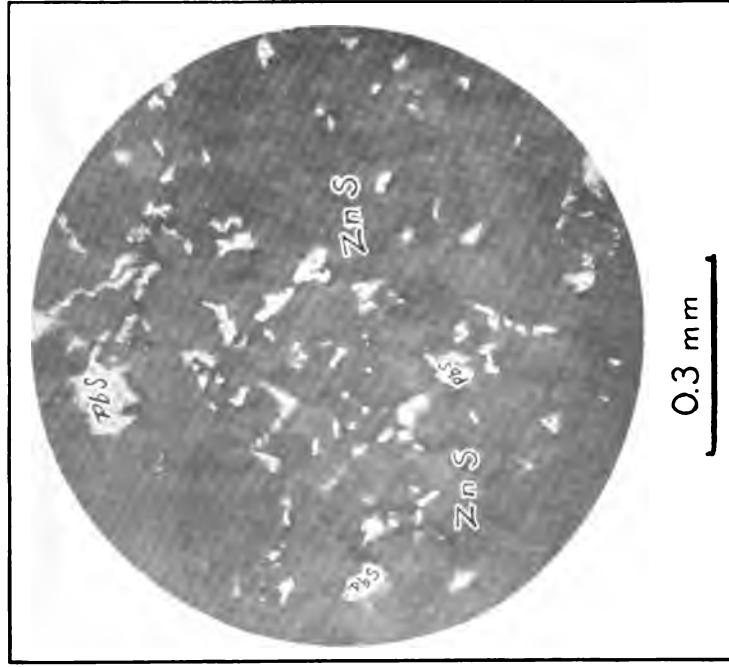
Results of test 567.

[Head contained Pb, 7 per cent; Zn, 12.9 per cent; Fe, 6.1 per cent.]

| Grade of products. | Assay of products. | | | Recoveries. | | |
|-----------------------|--------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 59.3 | 6.5 | 3.9 | 72.9 | 4.3 | 5.5 |
| Zinc concentrate..... | 4.8 | 39.2 | 8.5 | 20.3 | 90.1 | 41.2 |
| Middling..... | .7 | 1.4 | 9.3 | 1.1 | 1.2 | 16.8 |
| Tailing..... | 1.2 | .2 | 4.3 | 8.7 | .8 | 35.7 |

Flotation agents.

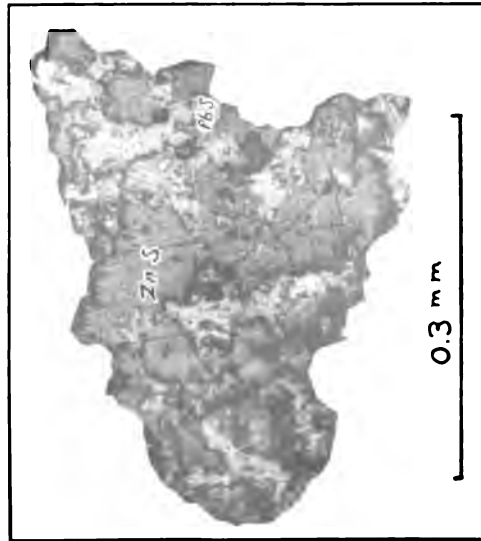
| | Pounds per ton. |
|---|-----------------|
| Soda ash..... | 10.0 |
| Barrett Co. No. 2 coal-tar creosote..... | 1.0 |
| Pensacola Tar & Turpentine Co. No. 400 wood creosote oil..... | .1 |
| Copper sulphate..... | 1.0 |
| Water glass (40° B.)..... | 10.0 |
| General Naval Stores Co. No. 5 s. d. pine oil..... | .1 |
| Pensacola Tar & Turpentine Co. No. 350 crude wood pine oil..... | .2 |



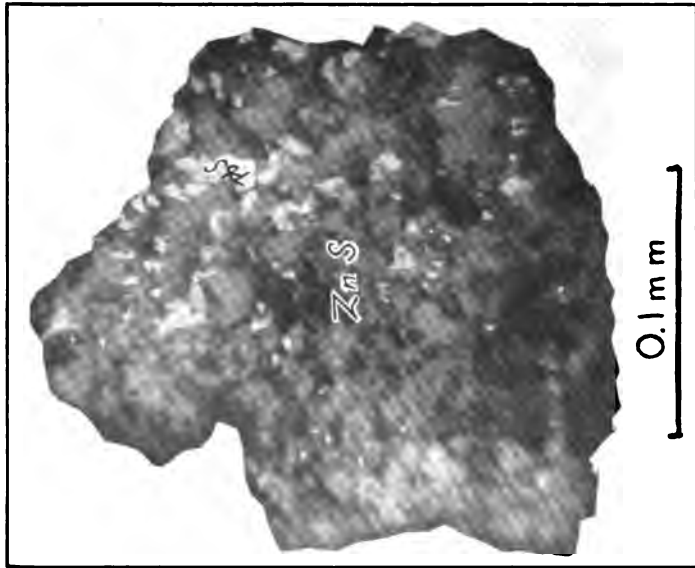
4. POLISHED SECTION SHOWING INCLUSIONS OF GALENA IN SPHALERITE, ORE 3.



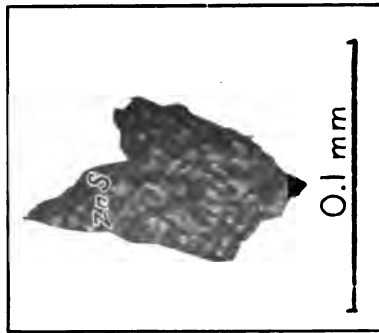
B. A 48-MESH PARTICLE OF ORE 3, SHOWING ASSOCIATION OF GALENA AND SPHALERITE.



A. A 65-MESH PARTICLE OF ORE 3, SHOWING ASSOCIATION OF THE GALENA AND ZINC BLENDE.



B. GALENA DISSEMINATED IN A SMALL GRAIN OF ZINC BLENDE, ORE 3.



C. GALENA IN A SMALLER PARTICLE OF ZINC BLENDE, ORE 3.

Remarks.—The grade of products and the zinc recovery were good, but a better grade could have been obtained readily by cleaning. The soda ash and oils No. 2 and No. 400 were used to float the galena, the other flotation agents being added to float the zinc.

TESTING ORE 3 FOR DIFFERENTIAL FLOTATION.

DESCRIPTION OF ORE.

Ore 3 was a lead-zinc ore containing galena and sphalerite, with a little pyrite, in a quartzite-siderite gangue. The lead and zinc minerals were quite intimately associated and rather finely disseminated throughout the gangue. The zinc blende was reddish-brown, similar in color to that in ore 2.

Examination of polished sections under the microscope showed an association of galena and sphalerite typical of the occurrence of these two minerals in the complex ores of lead and zinc. The interlocking grains, shown in the photomicrograph, Plate III, *B* (p. 25), were so small and the distribution of the lead and zinc so uniform that their complete separation by crushing would be very difficult and often practically impossible. The photomicrograph, Plate IV, *A*, shows galena in fractures and interstices in the sphalerite.

Examination of the grains of lead and zinc sulphides in a briquette of sealing wax and crushed ore showed that although much of the galena and sphalerite had been freed or separated by the crushing, an appreciable amount of the intimately associated galena and sphalerite had not been separated by this treatment. Plates IV and V illustrate clearly the intimate association of the minerals in the briquetted crushed sulphides. Plates IV, *B*, and V, *A*, photomicrographs of 48 and 65 mesh particles, show the close association of the galena and zinc blende. Plate V, *B*, a photomicrograph of a grain of zinc blende of probably the maximum size that would pass through a 100-mesh screen, shows clearly to what extent the disseminated galena occurs in the crushed material. Another sample of the same ore examined after crushing to pass a 100-mesh screen revealed the same conditions as described above. The included galena in the blende is shown in Plate V, *C*, a photomicrograph of the same magnification as Plate V, *B*, but of a grain of considerably smaller size.

This ore contained relatively high values in zinc, as clearly indicated by the following chemical analyses of three different lots:

Chemical analyses of ore 3.

| Constituents. | Lot 1. | Lot 2. | Lot 3. |
|--------------------------------------|------------------|------------------|------------------|
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Pb..... | 4.91 | 9.56 | 4.54 |
| Zn..... | 18.68 | 13.94 | 19.26 |
| Fe..... | 5.04 | 4.35 | 4.35 |
| Cu..... | Tr. | Tr. | Tr. |
| S..... | 10.41 | 9.28 | 11.11 |
| CaO..... | 1.67 | 9.9 | 3.40 |
| MgO..... | .92 | .37 | 3.43 |
| SiO ₂ | 47.74 | 44.08 | 47.27 |
| Al ₂ O ₃ | 9.96 | 8.27 | 9.65 |

SCREEN ANALYSIS OF ORE NO. 3.

A screen analysis was made of a sample of this ore after going through a Blake crusher and a set of rolls, the coarsest particles being about one-fourth inch in diameter. The screen products were examined to ascertain the size to which it would be necessary to crush the ore for flotation testing. The screen analysis and examination of the screen products follow:

TABLE 7.—*Screen analysis of ore 3.*

[Head contained Pb, 5.8 per cent; Zn, 11.1 per cent; Fe, 4.3 per cent.]

| Size of screen opening. | | | Weights. | | Assays (per cent). | | | Per cent of total head content. | | | Cumulative per cent of total head content. | | |
|--------------------------------|---------------|---------|-----------|----------------------|--------------------|-------|-------|---------------------------------|-------|-------|--|-------|-------|
| Mesh. | Milli-meters. | Inches. | Per cent. | Cumulative per cent. | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| On..... | 4 | 4.699 | 0.185 | 13.0 | 2.5 | 6.6 | 4.7 | 5.5 | 7.7 | 14.2 | | | |
| | 8 | 2.362 | .093 | 18.2 | 4.1 | 6.8 | 4.3 | 12.9 | 11.2 | 18.1 | 18.4 | 18.9 | 32.3 |
| | 10 | 1.651 | .065 | 9.2 | 40.4 | 4.9 | 10.1 | 4.7 | 7.7 | 8.4 | 10.0 | 26.1 | 27.3 |
| | 14 | 1.168 | .046 | 7.7 | 48.1 | 5.4 | 12.8 | 4.6 | 7.2 | 8.8 | 8.1 | 33.3 | 36.1 |
| | 20 | .833 | .0328 | 6.5 | 54.6 | 5.7 | 14.0 | 4.4 | 6.4 | 8.2 | 6.7 | 39.7 | 44.3 |
| | 28 | .589 | .0232 | 5.9 | 60.5 | 5.9 | 14.6 | 4.7 | 6.0 | 7.7 | 6.5 | 45.7 | 52.0 |
| | 35 | .417 | .0164 | 5.0 | 65.5 | 5.7 | 15.6 | 4.5 | 4.8 | 7.1 | 5.1 | 50.5 | 59.1 |
| | 48 | .295 | .0116 | 4.2 | 69.7 | 6.3 | 15.9 | 4.5 | 4.5 | 6.0 | 4.4 | 55.0 | 65.1 |
| | 65 | .208 | .0082 | 4.7 | 74.4 | 7.2 | 17.2 | 4.8 | 5.9 | 7.3 | 5.1 | 60.9 | 72.4 |
| | 100 | .147 | .0058 | 4.1 | 78.5 | 7.2 | 16.2 | 5.1 | 5.2 | 5.9 | 4.9 | 66.1 | 78.3 |
| | 150 | .104 | .0041 | 5.4 | 83.9 | 7.7 | 15.7 | 4.5 | 7.2 | 7.6 | 5.6 | 73.3 | 85.9 |
| | 200 | .074 | .0029 | 2.6 | 86.5 | 9.0 | 15.5 | 4.5 | 4.0 | 3.6 | 2.8 | 77.3 | 89.5 |
| Through..... | 200 | .074 | .0029 | 13.2 | 99.7 | 9.8 | 13.1 | 4.0 | 22.2 | 15.5 | 12.3 | 99.5 | 105.0 |
| Loss and error by difference.. | | | +0.3 | | | | | +0.5 | -5.0 | -3.8 | | | |
| Total..... | | | 100.0 | | | | | 100.0 | 100.0 | 100.0 | | | |

EXAMINATION OF SCREEN PRODUCTS.

Products on 4 to 28 mesh screens.—Under the microscope much combined galena, sphalerite, siderite, and quartz were visible. Sphalerite was the predominating mineral, although little could be found free from the associated minerals.

Product on 35-mesh screens.—This product contained free particles of galena, sphalerite, and quartz, but the greater percentage of galena and a comparatively large quantity of sphalerite were mechanically combined either with other minerals or with each other. The gangue appeared to be combined with galena and siderite rather than with the sphalerite.

Products on 48 and 65 mesh screens.—These products were similar to the material on a 35-mesh screen.

Products on 100-mesh screen.—This material contained a considerable proportion of free particles of galena, sphalerite, and quartz, although mechanically combined particles of these minerals could still be seen under the microscope. Most of this material would be suitable for flotation.

Products on 150, 200, and through 200 mesh screens.—Most of the mineral particles were free, but some of the quartz and siderite particles still contained included galena. It was, of course, difficult to determine particles of sphalerite containing included galena except by the microscope of higher power used in making the photomicrographs.

It was decided to crush the ore to pass a 65-mesh screen, so that practically all of the ore used in testing for differential flotation was crushed to this size or finer. A screen analysis of this crushed material is as follows:

TABLE 8.—*Screen analysis of ore 3 for flotation testing.*

[Head contained Pb, 5.7 per cent; Zn, 11 per cent; Fe, 4.4 per cent.]

| Size of screen opening. | | | Weights. | | Assays (per cent). | | | Per cent of total head content. | | | Cumulative per cent of total head content. | | |
|-------------------------------|---------------|---------|-----------|----------------------|--------------------|-------|-------|---------------------------------|-------|-------|--|-------|-------|
| Mesh. | Milli-meters. | Inches. | Per cent. | Cumulative per cent. | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| On.....65 | 0.208 | 0.0082 | 5.0 | | 3.2 | 11.2 | 3.9 | 2.8 | 5.1 | 4.5 | | | |
| 100 | .147 | .0058 | 14.6 | 19.6 | 4.0 | 11.7 | 3.5 | 10.2 | 15.6 | 11.6 | 13.0 | 20.7 | 16.1 |
| 150 | .104 | .0041 | 20.4 | 40.0 | 4.7 | 11.7 | 3.4 | 16.8 | 21.7 | 15.7 | 29.8 | 42.4 | 31.8 |
| 200 | .074 | .0029 | 15.8 | 55.8 | 5.3 | 11.5 | 3.9 | 14.7 | 10.6 | 14.1 | 44.5 | 59.0 | 45.9 |
| Through...200 | .074 | .0029 | 43.8 | 99.6 | 6.9 | 10.2 | 5.0 | 53.0 | 40.6 | 49.8 | 97.5 | 99.6 | 96.7 |
| Loss and error by difference. | | | +0.4 | | | | | +2.5 | +0.4 | +4.3 | | | |
| Total..... | | | 100.0 | | | | | 100.0 | 100.0 | 100.0 | | | |

RESULTS OF FLOTATION TESTS.

The following tests were selected from a number and indicate the possibilities of making commercial products by differential flotation. All tests were run without any attempt to clean the products.

Results of test 192.

[Head contained Pb, 4.9 per cent; Zn, 18.7 per cent; Fe, 5 per cent.]

| Grade of products. | Assay of products. | | | Recoveries. | | |
|---------------------------|--------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate | 59.4 | 14.5 | 2.5 | 65.0 | 4.2 | 2.8 |
| Zinc concentrate (1)..... | 7.3 | 45.2 | 6.9 | 24.8 | 52.4 | 30.0 |
| Zinc concentrate (2)..... | 2.6 | 37.4 | 7.2 | 8.7 | 33.5 | 23.8 |
| Tailing..... | .3 | .4 | 4.4 | 3.3 | 1.2 | 48.0 |

Flotation agents.

| | Pounds per ton. |
|---|-----------------|
| Barrett Co. No. 2 coal-tar creosote..... | 0.3 |
| Georgia Pine & Turpentine Co. No. 208 wood creosote oil..... | .1 |
| Pensacola Tar & Turpentine Co. No. 400 wood creosote oil..... | .4 |
| Sulphuric acid..... | 2.0 |

Remarks.—For the lead concentrate 0.3 pound of Barrett No. 2 oil and 0.1 pound of Pensacola Tar & Turpentine Co. oil No. 400 were used. The remaining flotation agents were added to float the zinc mineral. The grade of products and the recoveries were good, especially the zinc concentrates. The two zinc concentrates combined gave a product containing 4.3 per cent lead and 41.7 per cent zinc with a recovery of 85 per cent of the zinc.

Results of test 212.

[Head contained Pb, 4.7 per cent; Zn, 17.9 per cent; Fe, 4.8 per cent.]

| Grade of product. | Assay of product. | | | Recoveries. | | |
|-----------------------|-------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 63.1 | 9.9 | 2.6 | 71.2 | 2.9 | 2.9 |

Flotation agents.

| | Pounds per ton. |
|--|-----------------|
| Soda ash..... | 5.0 |
| Charcoal..... | 1.0 |
| Barrett Co. (Vancouver) light coal-tar creosote..... | .4 |

Remarks.—In this test, as in many others, the lead concentrate was the only product saved, in order to eliminate many chemical analyses when the ore was tested with different coal-tar creosotes or, when a series of tests was run, to avoid changing the quantity, make, or grade of the flotation oil. The lead product was fairly good in view of the relatively high zinc content of the head sample. The recovery of lead was fair.

Results of test 213.

[Head contained Pb, 4.7 per cent; Zn, 17.9 per cent; Fe, 4.8 per cent.]

| Grade of product. | Assay of product. | | | Recoveries. | | |
|-----------------------|-------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 58.2 | 12.2 | 2.2 | 80.5 | 4.4 | 2.9 |

Flotation agents.

| | Pounds
per ton. |
|--|--------------------|
| Soda ash..... | 5.0 |
| Charcoal..... | 1.0 |
| Barrett Co. (Vancouver) heavy coal-tar creosote..... | .4 |

Remarks.—As compared with the results of test No. 212, the heavy coal-tar creosote seemed to give a higher recovery but raised more zinc, causing a product of somewhat lower grade.

Results of test 224.

[Head contained Pb, 4.7 per cent; Zn, 17.9 per cent; Fe, 4.8 per cent.]

| Grade of product. | Assay of product. | | | Recoveries. | | |
|-----------------------|-------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 58.0 | 12.5 | 2.8 | 71.4 | 4.2 | 3.5 |

Flotation agents.

| | Pounds
per ton. |
|--|--------------------|
| Borax..... | 5.0 |
| Charcoal..... | 1.0 |
| Barrett Co. No. 4 coal-tar creosote..... | .4 |

Remarks.—In this test borax was used in place of soda ash. In such tests borax did not seem to give better results than the ash.

Results of test 239.

[Head contained Pb, 6 per cent; Zn, 10.6 per cent; Fe, 3.9 per cent.]

| Grade of product. | Assay of product. | | | Recoveries. | | |
|-----------------------|-------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 54.4 | 11.4 | 2.6 | 68.0 | 8.0 | 4.9 |

Flotation agents.

| | Pounds
per ton. |
|-------------------|--------------------|
| Soda ash..... | 5 |
| Charcoal..... | 1 |
| Wood alcohol..... | 2 |

Remarks.—Alcohol seemed to give fairly good results with ore 3 when substituted for a flotation oil, such as a coal-tar creosote, to raise the lead mineral.

Results of test 382.

[Head contained Pb. 5.7 per cent; Zn, 10.8 per cent; Fe, 4.4 per cent.]

| Grade of products. | Assay of products. | | | Recoveries. | | |
|-----------------------|--------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 53.1 | 11.2 | 3.7 | 50.9 | 5.7 | 4.6 |
| Zinc concentrate..... | 8.2 | 32.3 | 7.1 | 41.6 | 86.0 | 46.7 |
| Middling..... | 3.1 | 5.2 | 5.6 | 1.4 | 1.2 | 3.3 |
| Tailing..... | .3 | .9 | 3.0 | 3.3 | 5.3 | 43.0 |

Flotation agents.

| | Pounds per ton. |
|---|-----------------|
| Soda ash..... | 5.0 |
| Barrett Co. No. 2 coal-tar creosote..... | .4 |
| Sodium hydroxide..... | 2.0 |
| Pensacola Tar & Turpentine Co. No. 350 crude wood pine oil..... | .4 |
| Imperial fuel oil..... | .2 |
| Copper sulphate..... | 1.0 |

Remarks.—For the lead concentrate the soda ash and Barrett No. 2 creosote were used. The zinc concentrate was produced by adding the sodium hydroxide, oil No. 350, and Imperial fuel oil. After the lead and zinc concentrates were removed, copper sulphate was added for the middling. The recovery of zinc in the zinc concentrate was good. The grade of product could have been improved by cleaning.

Results of test 385.

[Head contained Pb, 5.7 per cent; Zn, 10.8 per cent; Fe, 4.4 per cent.]

| Grade of products. | Assay of products. | | | Recoveries. | | |
|-----------------------|--------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 50.8 | 10.5 | 4.2 | 61.1 | 6.6 | 6.6 |
| Zinc concentrate..... | 7.8 | 42.5 | 6.1 | 30.7 | 88.2 | 31.1 |
| Middling..... | 3.1 | 10.3 | 7.8 | 2.7 | 4.8 | 8.9 |
| Tailing..... | .2 | .8 | 3.4 | 2.3 | 4.7 | 49.6 |

Flotation agents.

| | Pounds per ton. |
|--|-----------------|
| Soda ash..... | 5.0 |
| Mixture (1:16) of G. N. S. Co. No. 5 pine oil and carbolic creosote..... | .4 |
| Water glass (40° B.)..... | 10.0 |
| Copper sulphate..... | 1.0 |
| Pensacola Tar & Turpentine Co. No. 350 crude wood pine oil..... | .4 |
| Imperial fuel oil..... | .2 |

Remarks.—The soda ash and mixture were used to raise the lead mineral. For the zinc concentrate 5 pounds of water glass, 0.5 pound of copper sulphate, 0.2 pound of No. 350 oil, and 0.1 pound of fuel oil were used, the remainder being added for the middling after the zinc concentrate had been removed. The grade of products and the recoveries, especially of zinc, were fairly good. Better products could have been made by roughing and cleaning.

Results of test 390.

[Head contained Pb, 5.8 per cent; Zn, 11.1 per cent; Fe, 4.3 per cent.]

| Grade of products. | Assay of products. | | | Recoveries. | | |
|-----------------------|--------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 55.9 | 8.9 | 3.2 | 50.1 | 4.2 | 3.8 |
| Zinc concentrate..... | 10.7 | 40.7 | 6.0 | 44.3 | 88.0 | 33.5 |
| Middling..... | 5.8 | 11.2 | 7.4 | 4.6 | 4.6 | 7.9 |
| Tailing..... | .2 | .3 | 3.8 | 2.3 | 1.8 | 58.7 |

Flotation agents.

| | Pounds per ton. |
|---|-----------------|
| Soda ash..... | 5.0 |
| Charcoal..... | 1.0 |
| Barrett Co. No. 2 coal-tar creosote..... | .4 |
| Water glass (40° B.)..... | 5.0 |
| Copper sulphate..... | .5 |
| Pensacola Tar & Turpentine Co. No. 350 crude wood pine oil..... | .4 |
| Imperial fuel oil..... | .2 |

Remarks.—Soda ash, charcoal, and Barrett No. 2 creosote were used for the lead concentrate. The other flotation agents were added for the zinc concentrate and middling after the lead concentrate had been removed. The grade of products was fair but could have been improved by further treatment in cleaning cells. The recovery of zinc in the zinc concentrate was good.

Results of test 392.

[Head contained Pb, 5.8 per cent; Zn, 11.1 per cent; Fe, 4.3 per cent.]

| Grade of products. | Assay of products. | | | Recoveries. | | |
|-----------------------|--------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 50.6 | 10.8 | 3.7 | 59.3 | 6.6 | 5.9 |
| Zinc concentrate..... | 8.0 | 33.7 | 6.9 | 28.7 | 72.5 | 33.4 |
| Middling..... | 2.9 | 14.8 | 9.2 | 5.2 | 13.5 | 21.6 |
| Tailing..... | .4 | 1.1 | 2.5 | 4.2 | 6.0 | 35.2 |

Flotation agents.

| | Pounds per ton. |
|---|-----------------|
| Soda ash..... | 5.0 |
| Barrett Co. No. 2 coal-tar creosote..... | .4 |
| Wood alcohol..... | 1.6 |
| Pensacola Tar & Turpentine Co. No. 350 crude wood pine oil..... | .4 |
| Imperial fuel oil..... | .2 |

Remarks.—Barrett No. 2 creosote and alcohol were used to produce the lead concentrate, soda ash and the other flotation agents being employed for the zinc concentrate and middling. It will be noticed that the zinc recovery was much lower than in other tests where copper sulphate or water glass or both were used. For rougher concentrates, however, the grade of the products was fair.

Results of test 396.

[Head contained Pb, 5.8 per cent; Zn, 11.1 per cent; Fe, 4.3 per cent.]

| Grade of products. | Assay of products. | | | Recoveries. | | |
|-----------------------|--------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 50.7 | 14.9 | 4.5 | 76.9 | 11.8 | 9.2 |
| Zinc concentrate..... | 3.5 | 32.2 | 8.0 | 17.9 | 86.1 | 55.2 |
| Middling..... | 2.6 | 11.8 | 4.4 | 2.2 | 6.2 | 5.6 |
| Tailing..... | .2 | .13 | 2.5 | 1.9 | .6 | 32.8 |

Flotation agents.

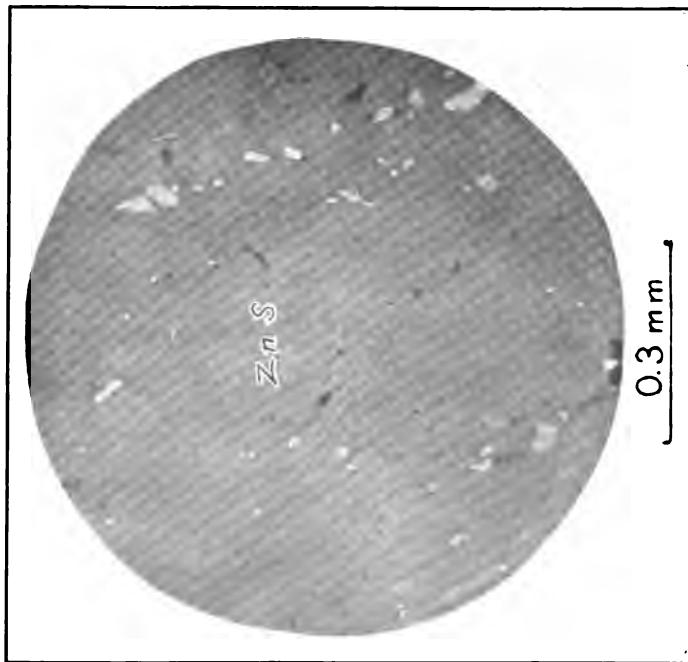
| | Pounds per ton. |
|---|-----------------|
| Sulphuric acid..... | 3.7 |
| Soda ash..... | 10.6 |
| Barrett Co. No. 2 coal-tar creosote..... | .4 |
| Copper sulphate..... | .5 |
| Water glass (40° B.)..... | 5.0 |
| Pensacola Tar & Turpentine Co. No. 350 crude wood pine oil..... | .6 |

Remarks.—Sulphuric acid was agitated with the pulp before other agents were added, in order to brighten the mineral faces, and thus possibly increase the lead recovery. After this preagitation the soda ash and Barrett No. 2 creosote were added for the lead concentrate. For the zinc concentrate 0.5 pound of copper sulphate and 0.3 pound of No. 350 oil were added, and for the middling, the water glass and 0.3 pound of No. 350 oil. The sulphuric acid seemed to increase the recovery of lead, but also permitted more zinc in the lead concentrate. However, from these two products high-grade concentrates of both lead and zinc could, no doubt, have been made by cleaning, with good recoveries of both minerals.

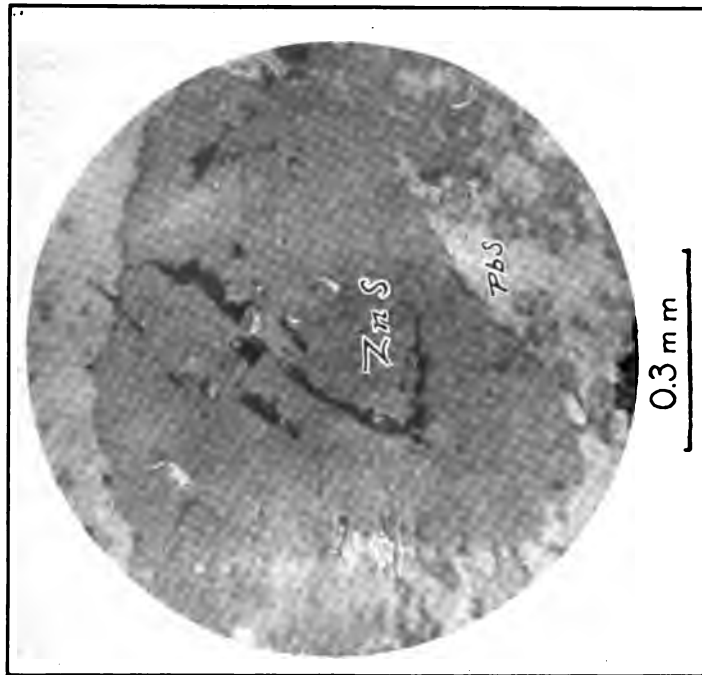
TESTING ORE 4 FOR DIFFERENTIAL FLOTATION.**DESCRIPTION OF ORE.**

Ore 4 contained sphalerite, galena, and some pyrite in a gangue composed chiefly of quartzite and schist. The sulphides were quite intimately associated with each other, as was clearly seen on examining polished section under the microscope.

Plate VI, *A*, is a photomicrograph of a polished section in which sphalerite is the chief constituent, the lead content in the form of galena being relatively small. Galena occurs as inclusions in the sphalerite in grains and minute particles ranging in size from those easily detected under the microscope at low magnification to specks which are only resolved by higher power. Plate VI, *A*, also shows how the galena is disseminated through the zinc blende and indicates the relative size of the galena particles, many of which are invisible at this magnification. In another specimen of this ore the galena and sphalerite seemed to occur in somewhat larger areas, and were associated as interlocking grains of varying sizes. The inclusions of galena in the sphalerite are readily distinguished in Plate VI, *B*.



4. POLISHED SECTION OF ORE 4, SHOWING GRAINS AND MINUTE PARTICLES OF GALENA IN SPHALERITE.



B. POLISHED SECTION OF ORE 4, SHOWING INTERLOCKING OF LARGER GRAINS OF SPHALERITE AND GALENA.

Making, by differential flotation, a high recovery of lead without floating a relatively large proportion of the zinc sulphide proved difficult. As the analysis below shows, the zinc content was only about 3 to 4 per cent and the lead about 11 to 12 per cent, so that even though a relatively good grade of lead concentrate was obtained, the quantity of zinc sulphide floated with the lead represented 30 to 40 per cent of the zinc in the ore. It is not difficult to obtain a high recovery of lead in a product assaying 50 to 55 per cent lead with 6 to 8 per cent zinc, but the recovery of zinc will be 30 to 40 per cent. In testing this ore, therefore, the object was to recover as much lead as possible in the lead concentrate while keeping the zinc down to a minimum.

Chemical analysis of ore 4.

| | Per cent. | | Per cent. |
|---------|-----------|--------------------------------------|-----------|
| Pb..... | 11.40 | CaO..... | 0.84 |
| Zn..... | 3.90 | MgO..... | .20 |
| Fe..... | 11.71 | SiO ₂ | 44.54 |
| Cu..... | Tr. | Al ₂ O ₃ | 22.53 |
| S..... | 4.76 | | |

SCREEN ANALYSIS OF ORE 4.

A screen analysis was made of a sample of this ore that had been crushed by a Blake crusher and a set of rolls so that the coarsest particle was about one-fourth inch in diameter. A set of Tyler standard screen sieves was used and the various screen products were examined to ascertain the size to which it would be necessary to crush the ore for flotation. The screen analysis and the examination of the screen products follow.

TABLE 9.—*Screen analysis of ore 4.*

[Head contained Pb, 11.4 per cent; Zn, 3.9 per cent; Fe, 11.7 per cent.]

| Size of screen opening. | | | Weights. | | Assays (percent). | | | Per cent of total head content. | | | Cumulative per cent of total head content. | | | |
|--------------------------------|---------------|-------|-----------|----------------------|-------------------|-------|-------|---------------------------------|-------|-------|--|-------|-------|------|
| Mesh. | Milli-meters. | Inch. | Per cent. | Cumulative per cent. | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. | |
| On | 4 | 4.699 | 0.185 | 29.9 | 10.8 | 3.0 | 12.7 | 28.3 | 23.0 | 32.4 | | | | |
| 8 | 2.362 | .093 | 18.9 | 48.8 | 9.8 | 3.5 | 11.6 | 16.2 | 16.9 | 18.7 | 44.5 | 39.9 | 51.1 | |
| 10 | 1.651 | .065 | 6.6 | 55.4 | 8.9 | 3.8 | 12.2 | 5.1 | 6.5 | 6.9 | 49.6 | 46.4 | 58.0 | |
| 14 | 1.168 | .046 | 7.1 | 62.5 | 8.8 | 4.0 | 12.7 | 5.5 | 7.3 | 7.7 | 55.1 | 53.7 | 65.7 | |
| 20 | .833 | .0325 | 6.2 | 68.7 | 8.6 | 4.3 | 11.6 | 4.7 | 6.9 | 6.2 | 59.8 | 60.6 | 71.9 | |
| 28 | .589 | .0232 | 4.9 | 73.6 | 9.1 | 4.0 | 11.0 | 3.9 | 5.0 | 4.6 | 63.7 | 65.6 | 76.5 | |
| 35 | .417 | .0164 | 4.1 | 77.7 | 8.4 | 4.3 | 11.8 | 3.1 | 4.6 | 4.2 | 66.8 | 70.2 | 80.7 | |
| 48 | .295 | .0116 | 2.9 | 80.6 | 7.8 | 4.5 | 10.6 | 2.0 | 3.3 | 2.6 | 68.8 | 73.5 | 83.3 | |
| 65 | .208 | .0082 | 2.6 | 83.2 | 10.1 | 4.6 | 11.4 | 2.3 | 3.0 | 2.5 | 71.1 | 76.5 | 85.8 | |
| 100 | .147 | .0058 | 2.7 | 85.9 | 11.5 | 4.8 | 12.5 | 2.7 | 2.9 | 2.9 | 73.8 | 79.4 | 88.7 | |
| 150 | .104 | .0041 | 2.2 | 88.1 | 13.0 | 4.4 | 8.2 | 2.5 | 2.4 | 1.5 | 76.3 | 81.8 | 90.2 | |
| 200 | .074 | .0029 | 1.3 | 89.4 | 14.5 | 4.1 | 7.9 | 1.7 | 1.4 | .9 | 78.0 | 83.2 | 91.1 | |
| Through..... | 200 | .074 | .0029 | 10.3 | 99.7 | 20.3 | 4.5 | 8.5 | 18.4 | 11.9 | 7.5 | 96.4 | 95.1 | 98.6 |
| Loss and error by difference.. | | | +0.3 | | | | | +3.6 | +4.9 | +1.3 | | | | |
| Total..... | | | | 100.0 | | | | 100.0 | 100.0 | 100.0 | | | | |

EXAMINATION OF SCREEN PRODUCTS.

Products on 4 to 28 mesh screens.—Much mechanically combined galena, sphalerite, and gangue could be seen under the microscope. Although there was some free mineral, most of the minerals were combined with each other. Galena predominated over sphalerite and relatively more galena than any of the other minerals present seemed to be free.

Products on 35 and 48 mesh screens.—Some galena and some sphalerite were free, but a large proportion of the minerals appeared to be mechanically combined.

Products on 65-mesh screen.—This screen product contained relatively more free galena and free sphalerite than the coarser products but most of the pyrite appeared to be combined with the gangue.

Product on 100-mesh screen.—A much greater proportion of the galena and sphalerite particles appeared to be free, although gangue particles combined with other minerals could be seen under the microscope. This size of screen product might prove satisfactory for flotation, but finer crushing would permit a better separation of the minerals by flotation and a more satisfactory recovery.

Product on 150-mesh screen.—This material contained principally free galena, sphalerite, pyrite, and quartz, although a few particles of quartz showed inclusions of lead and zinc, and particles of pyrite combined with zinc could be seen. As most of the mineral particles appeared to be free, it was decided to crush the ore to this size for flotation.

Products on 200 and through 200 mesh screens.—Most of the mineral particles in this material were free, although some of the quartz particles still showed inclusions of the sulphides. Because of the difficulty of determining particles of sphalerite containing inclusions of galena, except with the higher power microscope used in making photomicrographs, it is probable that a small proportion of the zinc particles still contained inclusions of galena.

After examination of the screen products it was thought best to crush this ore to pass a 100-mesh screen, so that practically all of the ore used in testing for differential flotation was crushed to this size or finer. A screen analysis of the material used for flotation follows.

TABLE 10.—*Screen analysis of ore 4 for flotation testing.*

[Head contained Pb, 11.4 per cent; Zn, 3.9 per cent; Fe, 11.7 per cent.]

| Size of screen opening. | | | Weights. | | Assays (per cent). | | | Per cent of total head content. | | | Cumulative per cent of total head content. | | |
|-------------------------------|---------------|---------|-----------|----------------------|--------------------|-------|-------|---------------------------------|-------|-------|--|-------|-------|
| Mesh. | Milli-meters. | Inches. | Per cent. | Cumulative per cent. | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| On.....100 | 0.147 | 0.0058 | 4.4 | | 6.9 | 2.9 | 10.6 | 2.6 | 3.1 | 3.9 | | | |
| 150 | .104 | .0041 | 14.0 | 18.4 | 8.1 | 3.1 | 11.4 | 10.0 | 11.3 | 13.7 | 12.6 | 14.4 | 17.6 |
| 200 | .074 | .0029 | 9.8 | 28.2 | 9.8 | 3.2 | 11.4 | 8.4 | 8.2 | 9.6 | 21.0 | 22.6 | 27.2 |
| Through.....200 | .074 | .0029 | 71.6 | 99.8 | 12.5 | 4.0 | 11.7 | 78.6 | 73.3 | 71.6 | 98.6 | 95.9 | 98.8 |
| Loss and error by difference. | | | +0.2 | | | | | +0.4 | +4.1 | +1.2 | | | |
| Total..... | | | 100.0 | | | | | 100.0 | 100.0 | 100.0 | | | |

RESULTS OF FLOTATION TESTS.

Many flotation oils and chemicals were used in tests of this ore for differential flotation, some of which have given promising results in effecting a good grade of lead product low in zinc. Charcoal in combination with soda ash and coal-tar creosote has seemed to give a better lead product than some of the other mixtures, but results in a somewhat lower recovery of lead. However, it is probable that better recoveries could be obtained in practice than are indicated by the results given below.

Results of test 155.

[Head contained Pb, 11.4 per cent; Zn, 3.9 per cent; Fe, 11.7 per cent.]

| Grade of product. | Assay of product. | | | Recoveries. | | |
|-----------------------|--------------------------|-------------------------|-------------------------|--------------------------|--------------------------|-------------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| Lead concentrate..... | <i>Per cent.</i>
52.6 | <i>Per cent.</i>
4.7 | <i>Per cent.</i>
4.9 | <i>Per cent.</i>
66.4 | <i>Per cent.</i>
17.4 | <i>Per cent.</i>
7.3 |

*Flotation agents.*Pounds
per ton.

Mixture (2:1) of wood alcohol and a distillate taken off from a mixture of (1:1) of coal-tar creosote and wood alcohol at 82° to 120° C. 2.5

Remarks.—The grade of the product was fair, especially with regard to the zinc content of the lead concentrate, but the recovery of lead was relatively low.

Results of test 180.

[Head contained Pb, 11.4 per cent; Zn, 3.9 per cent; Fe, 11.7 per cent.]

| Grade of product. | Assay of product. | | | Recoveries. | | |
|-----------------------|--------------------------|-------------------------|-------------------------|--------------------------|--------------------------|-------------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| Lead concentrate..... | <i>Per cent.</i>
50.1 | <i>Per cent.</i>
6.3 | <i>Per cent.</i>
5.6 | <i>Per cent.</i>
69.5 | <i>Per cent.</i>
21.6 | <i>Per cent.</i>
6.4 |

Flotation agents.

Mixture (4:1) of wood alcohol and a distillate taken off from a mixture (1:1) of coal-tar creosote and alcohol at 80° C.....
Sodium hydroxide.....

Remarks.—The grade of product was fair, but the recovery of lead was comparatively low. The sodium hydroxide seemed to effect a somewhat higher recovery.

Results of test 185.

[Head contained Pb, 11.4 per cent; Zn, 3.9 per cent; Fe, 11.7 per cent.]

| Grade of product. | Assay of product. | | | Recoveries. | | |
|-----------------------|--------------------------|-------------------------|-------------------------|--------------------------|--------------------------|--------------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| Lead concentrate..... | <i>Per cent.</i>
55.8 | <i>Per cent.</i>
7.2 | <i>Per cent.</i>
5.7 | <i>Per cent.</i>
76.4 | <i>Per cent.</i>
29.0 | <i>Per cent.</i>
11.7 |

Flotation agents.

Sodium hydroxide.....
General Naval Stores Co. No. 20 coal-tar creosote.....

Remarks.—This product contained more zinc than the product of the preceding test, but the recovery of lead was higher; the grade may be considered good, especially as it was a rougher product and could no doubt have been improved by cleaning.

Results of test 207.

[Head contained Pb, 11.4 per cent; Zn, 3.9 per cent; Fe, 11.7 per cent.]

| Grade of product. | Assay of product. | | | Recoveries. | | |
|-----------------------|--------------------------|-------------------------|-------------------------|--------------------------|--------------------------|--------------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| Lead concentrate..... | <i>Per cent.</i>
56.9 | <i>Per cent.</i>
4.9 | <i>Per cent.</i>
4.9 | <i>Per cent.</i>
77.4 | <i>Per cent.</i>
19.5 | <i>Per cent.</i>
11.7 |

Flotation agents.

Sodium hydroxide.....
Mixture (4:1) of wood alcohol and Flotation Oil & Chemical Co. No. 21 coal-tar creosote.....

Remarks.—Both the grade of product and the lead recovery were fairly good.

Results of test 572.

[Head contained Pb, 10.4 per cent; Zn, 3.4 per cent; Fe, 11.5 per cent.]

| Grade of product. | Assay of product. | | | Recoveries. | | |
|-----------------------|--------------------------|-------------------------|-------------------------|--------------------------|--------------------------|--------------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| Lead concentrate..... | <i>Per cent.</i>
54.7 | <i>Per cent.</i>
9.1 | <i>Per cent.</i>
6.9 | <i>Per cent.</i>
87.8 | <i>Per cent.</i>
43.6 | <i>Per cent.</i>
11.5 |

Flotation agents.

| | Pounds
per ton. |
|---|--------------------|
| Sodium hydroxide..... | 1 |
| Mixture (4:1) of wood alcohol and Flotation Oil & Chemical Co. No. 21 coal-tar
creosote..... | 1 |

Remarks.—The grade of the product was fair; the lead recovery was good, but the zinc content was relatively high. The higher recovery of lead was undoubtedly due to sample having been crushed wet. To compare wet and dry crushing see the preceding test, No. 207.

Results of test 585.

[Head contained Pb, 8.8 per cent; Zn, 3.5 per cent; Fe, 12 per cent.]

| Grade of product. | Assay of product. | | | Recoveries. | | |
|-----------------------|-------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 51.4 | 6.4 | 5.3 | 73.6 | 23.0 | 5.6 |

Flotation agent.

| | Pounds
per ton. |
|---------------|--------------------|
| Soda ash..... | 20 |

Remarks.—The grade of product was fair, but the recovery of lead was relatively low. The quantity of the flotation agent was too large for commercial use but it could probably be reduced in practice.

Results of test 792.

[Head contained Pb, 12.1 per cent; Zn, 3.2 per cent; Fe, 11.5 per cent.]

| Grade of products. | Assay of products. | | | Recoveries. | | |
|-----------------------|--------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 56.5 | 6.1 | 6.3 | 81.2 | 33.2 | 9.5 |
| Middling..... | 12.0 | 9.2 | 16.6 | 17.8 | 51.8 | 26.0 |
| Tailing..... | 1.0 | .6 | 12.3 | 5.3 | 12.1 | 69.0 |

Flotation agents.

| | Pounds
per ton. |
|---|--------------------|
| Soda ash..... | 20.0 |
| Flotation Oil & Chemical Co. No. 21 coal-tar creosote..... | .2 |
| Pensacola Tar & Turpentine Co. No. 400 wood creosote..... | .1 |
| Copper sulphate..... | 2.0 |
| Pensacola Tar & Turpentine Co. No. 350 crude wood pine oil..... | .4 |

Remarks.—Both the grade of product and the lead recovery were fairly good. The middling product could have been treated in separate flotation cells. The quantity of soda ash used would have to be reduced in practice. The copper sulphate and oil No. 350 were added after the lead concentrate had been removed.

Results of test 795.

[Head contained Pb, 12.1 per cent; Zn, 3.2 per cent; Fe, 11.5 per cent.]

| Grade of product. | Assay of product. | | | Recoveries. | | |
|-----------------------|-------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 59.4 | 7.4 | 5.5 | 77.1 | 36.3 | 7.3 |

Flotation agents.

| | Pounds
per ton. |
|--|--------------------|
| Mixture (1:4) of Barrett Co. No. 633 coal tar creosote and wood alcohol..... | 2 |
| Sodium hydroxide..... | 1 |

Remarks.—The grade of product and the recovery of lead were only fair.*Results of test 828.*

[Head contained Pb, 12.1 per cent; Zn, 3.2 per cent; Fe, 11.5 per cent.]

| Grade of products. | Assay of products. | | | Recoveries. | | |
|-----------------------|--------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 54.9 | 8.9 | 6.1 | 72.6 | 34.5 | 8.5 |
| Middling (1)..... | 31.4 | 11.3 | 12.8 | 8.8 | 11.9 | 3.8 |
| Middling (2)..... | 14.9 | 10.7 | 15.0 | 13.5 | 36.8 | 14.3 |

Flotation agents.

| | Pounds
per ton. |
|---|--------------------|
| Soda ash..... | 2.5 |
| General Naval Stores Co. No. 22 coal-tar creosote..... | .25 |
| Pensacola Tar & Turpentine Co. No. 350 crude wood pine oil..... | .25 |

Remarks.—The grade of product was fair, but the recovery of lead was low. This test was run with 4,000 grams of ore.*Results of test 841.*

[Head contained Pb, 12.1 per cent; Zn, 3.2 per cent; Fe, 11.5 per cent.]

| Grade of products. | Assay of products. | | | Recoveries. | | |
|-----------------------|--------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 51.9 | 8.5 | 7.6 | 79.4 | 49.1 | 12.2 |
| Middling (1)..... | 14.9 | 9.0 | 13.0 | 4.9 | 11.1 | 4.5 |
| Middling (2)..... | 7.3 | 4.8 | 15.0 | 4.4 | 11.1 | 9.6 |

Flotation agents.

| | Pounds
per ton. |
|--|--------------------|
| Soda ash..... | 2.5 |
| Sodium hydroxide..... | 1.0 |
| Mixture (1:4) of Flotation Oil & Chemical Co. coal tar creosote No. 21 and wood alcohol..... | 2.0 |
| General Naval Stores Co. No. 5 s. d. pine oil..... | .1 |

Remarks.—Soda ash, sodium hydroxide, and 1 pound per ton of mixture were used for the lead concentrate, and the remaining flotation agents for the middling products. The lead concentrate contained too much zinc, but the product might have been improved by further treatment in cleaner cells or by tabling. The low zinc content of the lead made difficult the effecting of a good lead recovery without recovery of a large proportion of the zinc in the lead concentrate.

Results of test 863.

[Head contained Pb, 12.1 per cent; Zn, 3.2 per cent; Fe, 11.5 per cent.]

| Grade of product. | Assay of product. | | | Recoveries. | | |
|------------------------|--------------------------|-------------------------|-------------------------|--------------------------|--------------------------|--------------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| Lead concentrate | <i>Per cent.</i>
39.2 | <i>Per cent.</i>
5.4 | <i>Per cent.</i>
9.0 | <i>Per cent.</i>
56.7 | <i>Per cent.</i>
29.5 | <i>Per cent.</i>
13.7 |

Flotation agents.

| | Pounds
per ton. |
|---|--------------------|
| Gasoline..... | 0.3 |
| Flotation Oil & Chemical Co. No. 21 coal-tar creosote..... | .4 |
| Pensacola Tar & Turpentine Co. No. 400 wood creosote oil..... | .2 |

Remarks.—Gasoline seemed to keep down the zinc in the lead concentrate, but the grade of the latter was very low. If a high lead recovery could have been obtained with this mixture and the rougher product had been put through cleaner cells, a good-grade product low in zinc might have been possible.

Results of test 867.

[Head contained Pb, 12.1 per cent; Zn, 3.2 per cent; Fe, 11.5 per cent.]

| Grade of product. | Assay of product. | | | Recoveries. | | |
|------------------------|--------------------------|-------------------------|-------------------------|--------------------------|--------------------------|--------------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| Lead concentrate | <i>Per cent.</i>
41.9 | <i>Per cent.</i>
6.1 | <i>Per cent.</i>
7.6 | <i>Per cent.</i>
56.4 | <i>Per cent.</i>
30.5 | <i>Per cent.</i>
10.6 |

Flotation agents.

| | Pounds
per ton. |
|---|--------------------|
| Gasoline..... | 4.8 |
| Flotation Oil & Chemical Co. No. 21 coal-tar creosote..... | .4 |
| Pensacola Tar & Turpentine Co. No. 400 wood creosote oil..... | .2 |

Remarks.—The relatively large quantity of gasoline did not seem to improve the grade of the product or to increase the lead recovery. The zinc content of the lead concentrates was also higher in this test than in that preceding (No. 863).

Results of test 869.

[Head contained Pb, 12.1 per cent; Zn, 3.2 per cent; Fe, 11.5 per cent.]

| Grade of product. | Assay of product. | | | Recoveries. | | |
|------------------------|--------------------------|-------------------------|-------------------------|--------------------------|--------------------------|-------------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| Lead concentrate | <i>Per cent.</i>
56.4 | <i>Per cent.</i>
5.2 | <i>Per cent.</i>
6.4 | <i>Per cent.</i>
68.5 | <i>Per cent.</i>
23.9 | <i>Per cent.</i>
8.2 |

Flotation agents.

| | Pounds
per ton. |
|---|--------------------|
| Soda ash..... | 5.0 |
| Gasoline..... | 1.2 |
| Flotation Oil & Chemical Co. No. 21 coal-tar creosote..... | .4 |
| Pensacola Tar & Turpentine Co. No. 400 wood creosote oil..... | .2 |

Remarks.—Comparison of this test with Nos. 863 and 867 shows that the addition of soda ash improved both the grade of product and the recovery of lead, although the latter is still low.

Results of test 877.

[Head contained Pb, 12.1 per cent; Zn, 3.2 per cent; Fe, 11.5 per cent.]

| Grade of product. | Assay of product. | | | Recoveries. | | |
|-----------------------|--------------------------|-------------------------|-------------------------|--------------------------|--------------------------|-------------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| Lead concentrate..... | <i>Per cent.</i>
60.9 | <i>Per cent.</i>
4.2 | <i>Per cent.</i>
5.4 | <i>Per cent.</i>
75.4 | <i>Per cent.</i>
19.7 | <i>Per cent.</i>
7.0 |

Flotation agents.

| | Pounds
per ton. |
|---|--------------------|
| Soda ash..... | 4.0 |
| Charcoal..... | .5 |
| *Flotation Oil & Chemical Co. No. 21 coal-tar creosote..... | .4 |

Remarks.—The use of charcoal and soda ash seemed to improve the grade of the product. The recovery of lead, however, was only fair.

Results of test 890.

[Head contained Pb, 12.1 per cent; Zn, 3.2 per cent; Fe, 11.5 per cent.]

| Grade of product. | Assay of product. | | | Recoveries. | | |
|-----------------------|--------------------------|-------------------------|-------------------------|--------------------------|--------------------------|-------------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| Lead concentrate..... | <i>Per cent.</i>
60.0 | <i>Per cent.</i>
4.8 | <i>Per cent.</i>
5.9 | <i>Per cent.</i>
74.9 | <i>Per cent.</i>
22.6 | <i>Per cent.</i>
7.7 |

Flotation agents.

| | Pounds
per ton. |
|--|--------------------|
| Soda ash..... | 5.0 |
| Charcoal..... | 1.0 |
| Wood alcohol..... | 1.0 |
| Flotation Oil & Chemical Co. No. 21 coal-tar creosote..... | .4 |

Remarks.—The addition of wood alcohol did not seem to improve the grade of product or to increase the recovery. Compare with test No. 877.

Results of test 892.

[Head contained Pb, 12.1 per cent; Zn, 3.2 per cent; Fe, 11.5 per cent.]

| Grade of product. | Assay of product. | | | Recoveries. | | |
|-----------------------|--------------------------|-------------------------|-------------------------|--------------------------|--------------------------|-------------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| Lead concentrate..... | <i>Per cent.</i>
55.5 | <i>Per cent.</i>
5.6 | <i>Per cent.</i>
6.6 | <i>Per cent.</i>
77.1 | <i>Per cent.</i>
29.4 | <i>Per cent.</i>
9.7 |

Flotation agents.

| | Pounds
per ton. |
|--|--------------------|
| Soda ash..... | 5.0 |
| Charcoal..... | 1.0 |
| Cresylic acid..... | .4 |
| Flotation Oil & Chemical Co. No. 21 coal-tar creosote..... | .4 |

Remarks.—The addition of cresylic acid in place of wood alcohol seemed to increase the recovery somewhat, but the grade of product was not quite so good, there being more zinc in the lead concentrate.

Results of test 893.

[Head contained Pb, 12.1 per cent; Zn, 3.2 per cent; Fe, 11.5 per cent.]

| Grade of product. | Assay of product. | | | Recoveries. | | |
|-----------------------|-------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 59.7 | 6.0 | 5.7 | 72.5 | 27.6 | 7.3 |

Flotation agents.

| | Pounds
per ton. |
|--------------------|--------------------|
| Soda ash..... | 5.0 |
| Charcoal..... | 1.0 |
| Wood alcohol..... | 1.0 |
| Cresylic acid..... | .2 |

Remarks.—This test was run without oil No. 21, a coal-tar creosote, and the results indicate that the lead recovery was lower than in other tests in which that oil was used.

Results of test 894.

[Head contained Pb, 12.1 per cent; Zn, 3.2 per cent; Fe, 11.5 per cent.]

| Grade of product. | Assay of product. | | | Recoveries. | | |
|-----------------------|-------------------|------------------|------------------|------------------|------------------|------------------|
| | Pb. | Zn. | Fe. | Pb. | Zn. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Lead concentrate..... | 57.3 | 5.7 | 5.2 | 76.7 | 28.8 | 7.3 |

Flotation agents.

| | Pounds
per ton. |
|--|--------------------|
| Soda ash..... | 5.0 |
| Charcoal..... | 1.0 |
| Alcohol..... | 1.0 |
| Cresylic acid..... | .2 |
| Flotation Oil & Chemical Co. No. 21 coal-tar creosote..... | .4 |

Remarks.—With the coal-tar creosote, oil No. 21, the recovery seems somewhat higher than in tests where it was not added to the flotation mixture. Wood alcohol had very little effect, if any, on the flotation of this ore.

GENERAL CONCLUSIONS.

On the basis of the experimental work the following conclusions seem justified, so far as Coeur d'Alene lead-zinc ores are concerned:

1. A complete separation of the lead and zinc sulphides by differential flotation is impossible, owing to the intimate association of the minerals, as shown by the accompanying photomicrographs.

2. Fine grinding is essential to liberate most of the sulphide minerals and to permit their separation by differential flotation.

3. A better recovery is possible with an ore that has been crushed wet than with one that has been crushed dry.

4. A pulp density of about $2\frac{1}{2}$:1 to $3\frac{1}{2}$:1 (water to solids) seems to give the best differential selection and mineral recovery at normal temperature.

5. Each ore seems to require a somewhat different flotative mixture and the effect of a certain oil or chemical on one ore may differ from that on another similar ore.

6. Differential flotation effects a more satisfactory separation of the sulphide minerals than has been obtained by existing methods.

7. A better differential selection of the lead from the zinc sulphide is effected with an alkaline or neutral pulp than with an acid pulp.

8. The quantity of the chemical or flotation agents used in the small scale tests can be reduced in large scale tests or in practice.

9. Soda ash, sodium hydroxide, sodium silicate (water glass), salt, lime, coal-tar creosote, alcohol, gasoline, and charcoal seem to assist in the selective separation and recovery of lead sulphide in the presence of zinc and iron sulphides. Their effectiveness, however, may be modified by the gangue constituents of the ore.

10. Copper sulphate or some other copper compound, such as copper carbonate, when added to a pulp containing soda ash after the lead concentrate has been removed, seems to give good results in floating the zinc mineral in some of the Coeur d'Alene lead-zinc ores.

11. The Bradford SO_2 process seems to be adaptable to the separation of the lead and zinc sulphides of some of the lead-zinc ores of Coeur d'Alene district.

12. Certain Coeur d'Alene lead-zinc ores should be amenable to differential flotation, and it should be possible to obtain good-grade lead and zinc products with the proper flotation mixture under the proper conditions, especially by a roughing and cleaning system, as well as good recoveries of both minerals.

PART II.—TESTS OF TWO COPPER ORES AND ONE ANTIMONIAL SILVER ORE FOR FLOTATION.

By CLARENCE A. WRIGHT, JAMES T. NORTON, AND JAMES C. PARMELLEE.

INTRODUCTION.

This part of the present bulletin deals with the flotation of three Idaho ores other than the lead and zinc ores of the Coeur d'Alene district. The ores tested for flotation were a chalcopyrite-pyrrhotite copper ore from Latah County; an antimonial silver ore, the silver mineral being principally polybasite, from a property near Florence, in Idaho County; and a copper ore containing gold and silver, which is being mined near Winchester, in Lewis County. The tests included in the pages following are not to be considered as final information on the flotation of these ores, but they indicate, and may suggest, possibilities in the treatment of these or similar ores by flotation.

TESTS OF A CHALCOPYRITE-PYRRHOTITE COPPER ORE.

DESCRIPTION OF ORE.

The copper contained in this ore was in the form of chalcopyrite intimately associated with pyrrhotite and pyrite in a schistose gangue composed chiefly of muscovite, calcite, quartz, and hornblende. The chalcopyrite was both coarsely and finely disseminated. Close examination showed that some of the pieces of chalcopyrite contained grains of interlocked pyrrhotite and pyrite, which indicate the necessity of fine grinding in order to liberate the chalcopyrite from the other sulphide minerals. Examination of the screen products from a screen analysis of the crushed ore showed that the material passing the 200-mesh screen still contained some chalcopyrite mechanically combined with pyrrhotite and pyrite.

A microscopic study² of polished surfaces of this ore developed some interesting facts in the association of chalcopyrite and pyrrhotite. In addition to occurrences of the normal chalcopyrite and pyrrhotite, which were readily identified by differences in color and character of surface there were many areas in which the texture presented a marked difference from that of either of these two minerals. These areas had an exceedingly pitted or spongelike surface which did not polish readily, but retained its peculiar appearance long after a good surface had been obtained on the normal chalcopyrite and pyrrhotite. Polishing for a considerable time finally produced a surface of sufficient flatness to render examination possible and disclosed that these rough areas were what might be termed "cupriferous pyrrhotite,"

² Study and photomicrographs by R. E. Head, U. S. Bureau of Mines, Salt Lake City, Utah.

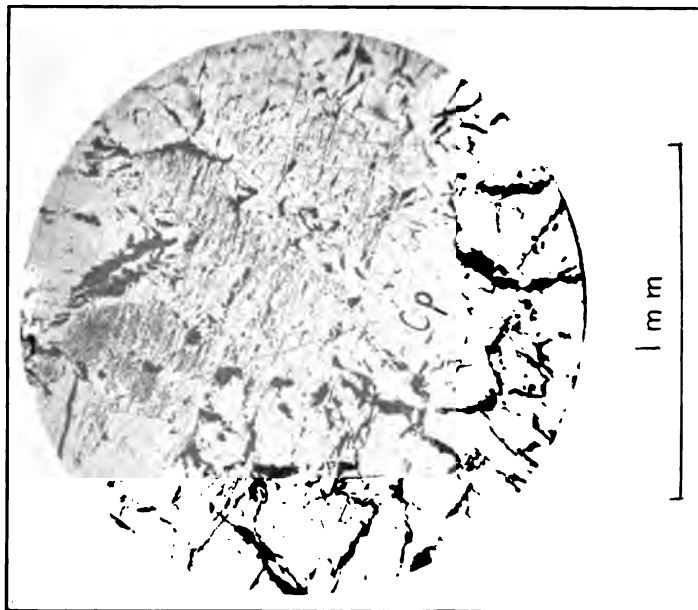
as they consisted of varying amounts of chalcopyrite and pyrrhotite, the chalcopyrite occurring as a replacement mineral filling the pits and cracks in the pyrrhotite, the roughened appearance persisting until the replacement was nearly complete. These semireplaced areas of pyrrhotite were of considerable size and the association of the minerals was so intimate and complex that separation by fine crushing would have been physically impossible, even the finest grains comprising both pyrrhotite and chalcopyrite. The application of differential flotation to the normal unaltered pyrrhotite and chalcopyrite was, no doubt, responsible for the separations obtained, and it was evident that the difficulty of effecting better separations lay in the material of complex association and variable composition. Careful study of these replacement areas showed that they comprised mixtures of the two minerals in all proportions, ranging from slightly altered pyrrhotite containing but little copper sulphide to a mixture in which chalcopyrite predominated. The latter would, in all probability, separate out with the true chalcopyrite, whereas those grains in which pyrrhotite was the chief constituent would remain with the normal pyrrhotite.

The accompanying photomicrographs are included to bring out the intimate association of the chalcopyrite and pyrrhotite. In Plate VII, *A*, the lower half represents the normal chalcopyrite, and the upper half, being composed largely of cupriferous pyrrhotite, shows the roughened, finely pitted appearance of the partly replaced pyrrhotite. These roughened areas represent the mixtures of the two sulphides which would not be separated by fine crushing.

Plate VII, *B*, is a photomicrograph of a polished section showing the relation of the normal sulphides, the lower half of the area being chalcopyrite and the upper pyrrhotite.

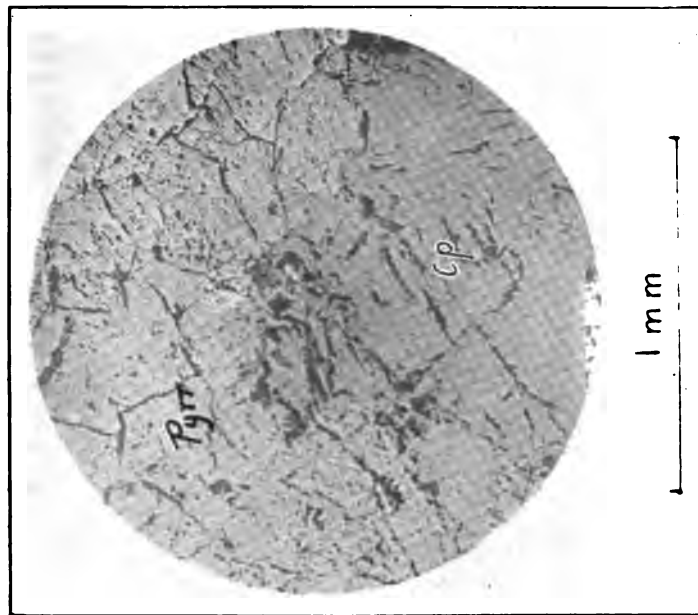
The left half of Plate VIII, *A*, shows chalcopyrite that has almost completely replaced the pyrrhotite, although remains of the latter mineral are still visible, as indicated by the pitted areas. Extending through the center of the section shown is an area of unreplaced pyrrhotite, which is bordered on the extreme right by chalcopyrite. The darker mineral in the lower right portion is quartz. These photomicrographs were taken at low magnifications and the complex relations of the minerals appear to represent appreciable areas in the ore.

Owing to the similarity in color of pyrrhotite and chalcopyrite, it is difficult to show in a photomicrograph a marked difference between the minerals. Therefore no attempt was made to photograph the crushed material.

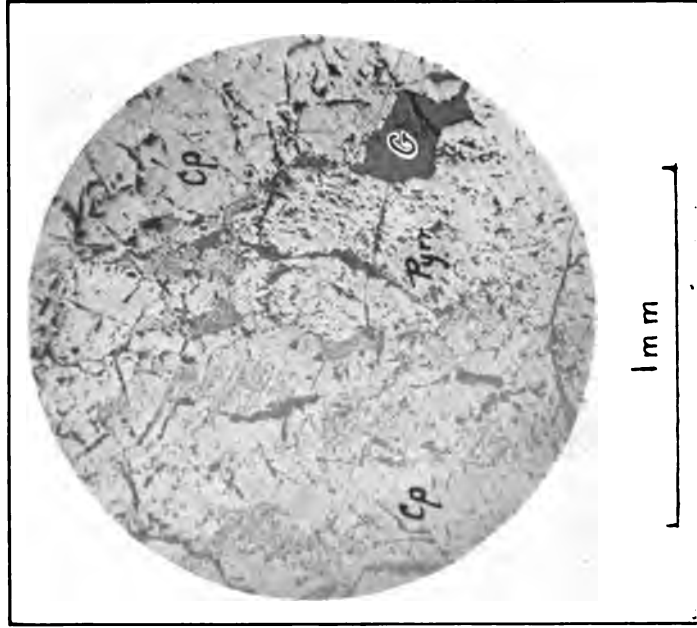


4. POLISHED SECTION SHOWING NORMAL CHALCOCOPYRITE (LOWER HALF) AND CUPRIFEROUS PYRRHOTITE.

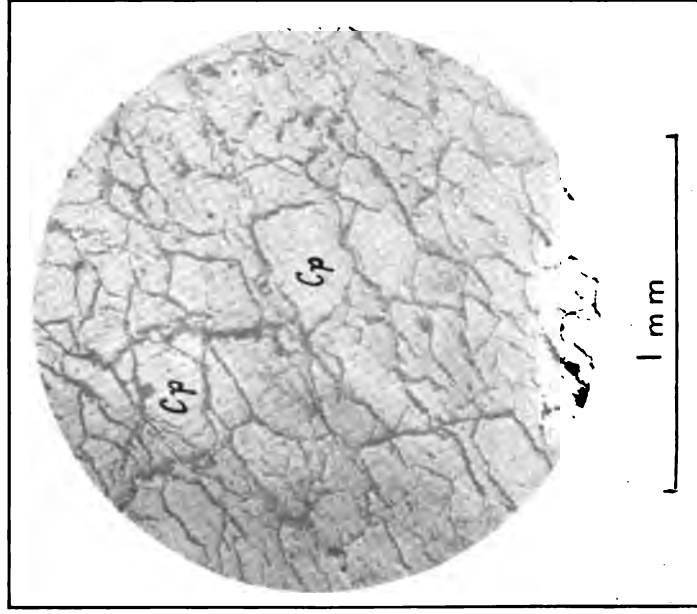
Note finely fluted aspect of partly replaced pyrrhotite.



B. POLISHED SECTION SHOWING RELATION OF NORMAL SULPHIDES.



4. POLISHED SECTION SHOWING CHALCOPYRITE (AT LEFT) THAT HAS ALMOST COMPLETELY REPLACED PYRRHOTITE, REMAINS OF THE LATTER BEING INDICATED BY PITTED AREAS.



5. POLISHED SECTION SHOWING ASSOCIATION OF CHALCOCITE AND CHALCOPYRITE, THE DARKER CHALCOCITE OCCURRING IN FRACTURES AND VEINLETS IN THE CHALCOPYRITE.

SCREEN ANALYSIS.

After the sample had been crushed, by a small Blake crusher, to about one-fourth-inch size, a screen analysis was made in a Tyler Ro-Tap shaking machine, and then the various screen products were examined to ascertain the fineness of crushing that would be necessary for flotation testing. Results of the screen analysis and the examination of the screen products follow:

TABLE 11.—*Screen analysis of chalcopyrite-pyrrhotite ore.*

| Size of screen opening. | | | Weights. | | |
|-------------------------|--------------|---------|----------|--------------------|-------------------------------|
| Mesh. | Millimeters. | Inches. | Grams. | Weight (per cent). | Cumulative weight (per cent). |
| On.....10 | 1.651 | 0.065 | 573 | 57.3 | ----- |
| 14 | 1.183 | .046 | 51 | 5.1 | 62.4 |
| 20 | .833 | .0328 | 38 | 3.8 | 66.2 |
| 35 | .417 | .0164 | 78 | 7.8 | 74.0 |
| 65 | .208 | .0082 | 74 | 7.4 | 81.4 |
| 100 | .147 | .0058 | 52 | 5.2 | 86.6 |
| 150 | .104 | .0041 | 34 | 3.4 | 90.0 |
| 200 | .074 | .0029 | 15 | 1.5 | 91.5 |
| Through 200 | .074 | .0029 | 84 | 8.4 | 99.9 |
| Loss..... | ----- | ----- | 1 | .1 | 100.0 |
| Total.... | ----- | ----- | 1,000 | 100.0 | ----- |

EXAMINATION OF THE SCREEN PRODUCTS.

Product on 10-mesh screen.—The chief minerals observed were chalcopyrite, pyrrhotite, pyrite, calcite, and quartz, with minor amounts of mica and hornblende. Free chalcopyrite could be seen but a large proportion was mechanically combined with the gangue or other sulphide minerals.

Products on 14 and 20 mesh screens.—More of the chalcopyrite had been liberated than in the previous screen product, but it was still largely combined with the pyrrhotite.

Products on 35 and 65 mesh screens.—Considerably more chalcopyrite, calcite, and other minerals had been liberated. The chalcopyrite and pyrrhotite and possibly pyrite seemed to be much more intimately combined than the other minerals.

Product on 100-mesh screen.—Most of the various minerals appeared to be free, but it was difficult to determine definitely to what extent the chalcopyrite was still combined with the other sulphides.

Products on 150, 200, and through 200 mesh screens.—To all appearances, practically all the minerals had been liberated, but the same difficulty arose in distinguishing the particles of chalcopyrite combined with pyrrhotite.

From the examination of these screen products, crushing through a 65-mesh screen, at least, would seem necessary for this ore. In all the flotation tests following, therefore, practically all of the ore has been crushed through such a screen. A screen analysis of this material for flotation testing is as follows:

TABLE 12.—*Screen analysis of material for flotation testing.*

[Head contained Cu, 4.7 per cent; Fe, 15.3 per cent; S, 10.7 per cent.]

| Size of screen opening. | | | Weights. | | Assays (per cent). | | | Percentage of total head content. | | | Cumulative per cent of total head content. | | |
|-----------------------------|--------------|---------|--------------------|------------------------|--------------------|-------|-------|-----------------------------------|-------|-------|--|-------|-------|
| Mesh. | Millimeters. | Inches. | Weight (per cent). | Cumulative (per cent). | Cu. | Fe. | S. | Cu. | Fe. | S. | Cu. | Fe. | S. |
| On.....65 | 0.208 | 0.0082 | 2.0 | | 1.1 | 14.7 | 6.6 | 0.5 | 1.9 | 1.2 | | | |
| 100 | .147 | .0058 | 14.0 | 16.0 | 4.5 | 16.6 | 11.2 | 13.4 | 15.2 | 14.7 | 13.9 | 17.1 | 15.9 |
| 150 | .104 | .0041 | 27.2 | 43.2 | 4.5 | 16.4 | 10.7 | 26.0 | 29.1 | 27.2 | 39.9 | 46.2 | 43.1 |
| 200 | .074 | .0029 | 7.2 | 50.4 | 4.2 | 15.0 | 10.3 | 6.4 | 7.1 | 6.9 | 46.3 | 53.3 | 50.0 |
| Through..200 | .074 | .0029 | 49.2 | 99.6 | 4.5 | 15.5 | 10.6 | 47.1 | 49.8 | 48.7 | 93.4 | 103.1 | 98.7 |
| Loss or error by difference | | | + .4 | | | | | + 6.6 | - 3.1 | + 1.3 | | | |
| Total... | | | 100.0 | | | | | 100.0 | 100.0 | 100.0 | | | |

Remarks.—It is interesting to note, as shown by the figures under "Percentage of total head content," that the distribution of copper and iron in the different screen products was practically the same.

When the screen products from the analysis of the crushed ore through a 65-mesh screen were examined the material on the 100-mesh screen showed much free mineral adaptable to flotation. A certain amount of combined chalcopyrite and pyrrhotite, however, could be observed. The material on the 150-mesh screen, as far as could be seen through a low-power microscope, comprised practically free mineral particles, although a few combined particles of chalcopyrite and quartz were noted. Owing to the intimate association of the chalcopyrite and pyrrhotite, there was no doubt, a small percentage of interlocked particles of these two minerals which could have been resolved with a microscope of higher power. This statement also applies to the material on and through a 200-mesh screen.

RESULTS OF FLOTATION TESTS.

When this ore was tested for differential flotation mechanical agitation machines of the Federal-Varley type (*see* fig. 1, p. 9) were used. All preliminary or small-scale tests were run in a small machine capable of treating 1,000 grams of ore. When satisfactory results were obtained with this machine the tests were repeated under the same conditions in a machine of the same type with a capacity of 4,000 grams.

Several oils and chemicals were used testing the ore for differential flotation. Certain mixtures gave good recoveries but relatively low-grade products, whereas other mixtures gave much better products, effecting a good differential separation of the chalcopyrite from the pyrrhotite, but relatively low recoveries. In testing out different mixtures or varying the quantity of a certain mixture to be used, all other conditions were kept constant. After the flotation agent was added the ore was given a preagitation for three to five minutes at a pulp dilution of 1 to 1, water to solids. Following the preagitation, water was added to raise the water level in the flotation machine so that the pulp dilution was about 3 to 1, water to solids. Agitation was then continued for 10 minutes, the froth being removed as it formed. In some tests a middling was taken off after the concentrate had been removed, more oil or flotation mixture being used to effect a higher recovery. The results from a few of the more satisfactory tests of this ore are given below:

Results of test 15.

[Head contained Cu, 4.7 per cent; Fe, 15.3 per cent.]

| Grade of products. | Assay of products. | | Recoveries. | |
|--------------------|--------------------|------------------|------------------|------------------|
| | Cu. | Fe. | Cu. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Concentrate..... | 24.6 | 31.4 | 76.3 | 30.0 |
| Middling (1)..... | 5.9 | 45.3 | 17.0 | 40.2 |
| Middling (2)..... | 1.6 | 32.7 | 2.8 | 16.9 |
| Tailing..... | .1 | 3.5 | 1.3 | 14.8 |

Flotation agents.

| | Pounds
per ton. |
|--|--------------------|
| Soda ash..... | 5.0 |
| General Naval Stores Co. No. 5 pine oil..... | .4 |
| Standard Chemical Co. No. 2 coal-tar creosote..... | .2 |

Remarks.—The differential separation of the chalcopyrite from pyrrhotite and pyrite was fairly good, but the recovery of copper was only fair. The grade of product was good in view of the fact that no attempt was made to clean the first product.

Results of test 22.

[Head contained Cu, 4.7 per cent; Fe, 15.3 per cent.]

| Grade of products. | Assay of products. | | Recoveries. | |
|--------------------|--------------------|------------------|------------------|------------------|
| | Cu. | Fe. | Cu. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Concentrate..... | 20.0 | 37.0 | 83.0 | 34.0 |
| Middling..... | 1.2 | 4.6 | 8.8 | 44.4 |
| Tailing..... | .8 | 6.3 | 11.9 | 28.6 |

Flotation agents.

| | Pounds
per ton. |
|---|--------------------|
| Sodium hydroxide..... | 2.0 |
| Mixture (1:4) of Vancouver heavy coal-tar and alcohol..... | 1.0 |
| Pensacola Tar & Turpentine Co. No. 400 wood creosote oil..... | .2 |

Remarks.—The figures from this test indicated a higher recovery of copper than in the preceding test, No. 15, but the iron content in the concentrate was a trifle higher. The product, on the whole, however, was fairly good.

Results of test 31.

[Head contained Cu, 4.7 per cent; Fe, 15.3 per cent.]

| Grade of products. | Assay of products. | | Recoveries. | |
|--------------------|--------------------|------------------|------------------|------------------|
| | Cu. | Fe. | Cu. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Concentrate..... | 25.6 | 31.5 | 80.2 | 30.0 |
| Middling..... | 4.6 | 38.4 | 13.0 | 33.2 |
| Tailing..... | .6 | 7.7 | 9.1 | 36.2 |

Flotation agents.

| | Pounds
per ton. |
|---|--------------------|
| Pensacola Tar & Turpentine Co. No. 400 wood creosote oil..... | 0.32 |
| Barrett Co. No. 2 coal-tar creosote..... | 1.4 |

Remarks.—The results indicated a product of good grade and a fair recovery of copper.

Results of test 38.

[Head contained Cu, 4.7 per cent; Fe, 15.3 per cent.]

| Grade of products. | Assay of products. | | Recoveries. | |
|--------------------|--------------------|------------------|------------------|------------------|
| | Cu. | Fe. | Cu. | Fe. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Concentrate..... | 16.3 | 39.8 | 82.9 | 62.5 |
| Middling..... | 4.5 | 33.3 | 8.2 | 18.5 |
| Tailing..... | .5 | 5.5 | 7.1 | 24.0 |

Flotation agents used.

| | Pounds
per ton. |
|---|--------------------|
| Pensacola Tar & Turpentine Co. No. 400 wood creosote oil..... | 0.2 |
| Barrett Co. No. 2 coal-tar creosote..... | .35 |

Remarks.—The results indicated a fair recovery of copper, but the iron content in the concentrate was relatively high.

CONCLUSIONS.

On the basis of the experimental work on this ore, the following conclusions seem justified:

1. Because of the intimate association of the chalcopyrite and pyrrhotite in the ore, a complete separation of the two minerals by gravity or differential flotation is impossible.

2. Crushing the ore to at least 65-mesh size is essential to effect a good separation of the sulphite minerals by differential flotation.

3. A good-grade copper concentrate and a fair recovery of copper can be effected by differential flotation with the proper flotation mixture; such a concentrate would assay about 20 per cent copper and about 30 to 35 per cent iron.

TESTS OF AN ANTIMONIAL SILVER ORE.

DESCRIPTION OF THE ORE.

The silver in this ore was in the form of polybasite, a sulphide of antimony and silver, which was finely disseminated throughout a hard, massive, quartz gangue. The identification of the polybasite was based upon a study³ of its physical properties under the microscope and blowpipe tests of carefully picked material. The results of a qualitative chemical test of a sample of this ore showed only a trace of arsenic, but a comparatively large proportion of antimony. The ore also contained a small amount of argentite, but no cerargyrite or other silver mineral was noted.

Besides the argentite and polybasite, small amounts of pyrite, stibnite, and magnetite were identified, but these minerals represented only a small percentage of the metallic content of the ore. A general assay of a sample of the ore resulted as follows:

Assay of ore with reference to silver content.

| | |
|------------------------------|------|
| Gold (ounces per ton)..... | 0.04 |
| Silver (ounces per ton)..... | 81.4 |
| Iron (per cent)..... | 1.13 |
| Sulphur (per cent)..... | .15 |
| Insoluble (per cent)..... | 97.8 |

CRUSHING THE ORE FOR FLOTATION TESTING.

The sample of ore received for testing purposes was crushed, by a small Blake crusher, to about $\frac{1}{4}$ -inch size and then screened. Most of the mineral particles seemed to be free in the material passing a 65-mesh screen, and it was therefore decided to crush the ore to that fineness for flotation testing.

A close examination of the various sizes obtained by screening the several products from flotation tests indicated that the sulphide minerals were practically freed from the gangue by the crushing, and although the tails showed quartz grains that carried inclusions of sulphides, the mineral value thus lost would be comparatively negligible when calculated on a percentage basis. Some sulphide grains

³Microscopic examination of this ore was made by R. E. Head, U. S. Bureau of Mines, Salt Lake City, Utah.

in the tailings were perfectly free from the gangue and under the microscope exhibited no physical characteristics that would hinder their floating. It is therefore assumed that they were carried out mechanically by the gangue.

RESULTS OF FLOTATION TESTS.

In testing this ore for flotation the same procedure was followed as in testing the chalcopyrite-pyrrhotite copper ore.

At first considerable difficulty was encountered in trying to effect a high recovery of the silver by flotation. It was thought that some of the silver might be in the form of a chloride, but examination of the tailings from tests showed that some of the polybasite mineral had partly oxidized or tarnished on the surface, preventing or interfering with their flotation. This impediment was overcome to a considerable extent by giving the ore or pulp a preliminary treatment with sodium sulphide, which brightened the mineral particles, and the newly formed sulphide coating on their surfaces permitted the particles to float, which incidentally increased the silver recovery.

Several oils and chemicals were used in testing this ore. Certain mixtures gave fair recoveries with relatively low-grade products, whereas other mixtures gave good-grade products with somewhat lower recoveries. Adding a small amount of sodium cyanide to the flotation mixture seemed to make the grade of the silver product higher, both with and without presulphidization of the ore pulp. In testing different mixtures, or varying the quantity of a certain mixture, all other conditions were kept constant. After the flotative agent was added the ore was given a preagitation for three to five minutes at a pulp dilution of 1 to 1, water to solids; then the pre-agitation water was added to raise the water level in the flotation machine, making the pulp dilution about 3 to 1, water to solids. Agitation was continued for 10 minutes, the froth being removed as it formed.

In tests where the ore received a sulphidizing treatment previous to flotation, the ore or pulp of about 1:1 density, water to solids, was placed in a bottle with a certain amount of sodium sulphide and the bottle agitated on rollers for any length of time desired. Then the sulphidized pulp was washed into the agitating cell of the flotation machine and floated in the usual manner.

Results of test 7.

[Head contained Au, 0.04 ounce per ton; Ag, 81.4 ounces per ton.]

| Grade of products. | Assay of products. | | Recoveries. | |
|--------------------|------------------------|------------------------|------------------|------------------|
| | Au. | Ag. | Au. | Ag. |
| | <i>Ounces per ton.</i> | <i>Ounces per ton.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Concentrate..... | 0.3 | 941.5 | 48.7 | 75.2 |
| Tailing..... | Tr. | 16.6 | | 19.0 |

Flotation agents.

Pounds per ton.

| | |
|------------------------------------|-----|
| Spencer Co. residual kelp oil..... | 0.2 |
| Vancouver imperial fuel oil..... | 1.0 |

Remarks.—The results indicated that a good grade of silver product is possible, but the recovery of silver was rather low.

Results of test 9.

[Head contained Au, 0.04 ounce per ton; Ag, 81.4 ounces per ton.]

| Grade of products. | Assay of products. | | Recoveries. | |
|--------------------|------------------------|------------------------|------------------|------------------|
| | Au. | Ag. | Au. | Ag. |
| | <i>Ounces per ton.</i> | <i>Ounces per ton.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Concentrate..... | 0.1 | 582.7 | 30.0 | 82.3 |
| Tailing..... | Tr. | 19.4 | | 20.9 |

Flotation agent.

Pounds per ton.

| | |
|-------------------------------------|-----|
| Turpentine-rosin mixture (5:1)..... | 0.6 |
|-------------------------------------|-----|

Remarks.—Results showed a rather low grade of silver product and a fair recovery. The mixture of turpentine and rosin was a strong frothing agent.

Results of test 22.

[Head contained Au, 0.04 ounce per ton; Ag, 81.4 ounces per ton.]

| Grade of products. | Assay of products. | | Recoveries. | |
|--------------------|------------------------|------------------------|------------------|------------------|
| | Au. | Ag. | Au. | Ag. |
| | <i>Ounces per ton.</i> | <i>Ounces per ton.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Concentrate..... | 0.4 | 1,235.8 | 47.5 | 73.0 |
| Tailing..... | Tr. | 24.4 | | 28.5 |

Flotation agents.

Pounds per ton.

| | |
|--|-----|
| Sodium cyanide..... | 0.5 |
| Cleveland Cliffs No. 2 wood creosote..... | 1.0 |
| Standard Chemical Co. No. 3 coal-tar creosote..... | .6 |

Remarks.—The results showed a good grade of silver product, but a rather low recovery. The addition of a small amount of sodium cyanide to the flotation mixture seemed to improve the grade.

Results of test 26.

[Head contained Au, 0.04 ounce per ton; Ag, 81.4 ounces per ton.]

| Grade of products. | Assay of products. | | Recoveries. | |
|--------------------|------------------------|------------------------|------------------|------------------|
| | Au. | Ag. | Au. | Ag. |
| | <i>Ounces per ton.</i> | <i>Ounces per ton.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Concentrate..... | 0.15 | 496.5 | 60.0 | 81.8 |
| Tailing..... | Tr. | 14.5 | | 15.3 |

Flotation agents.

| | Pounds per ton. |
|--|-----------------|
| Sodium sulphide (1 hour)..... | 0.5 |
| Cleveland Cliffs No. 2 wood creosote..... | 2.0 |
| Standard Chemical Co. No. 3 coal-tar creosote..... | .6 |

Remarks.—The results indicated that a sulphidizing treatment of the ore previous to flotation increased the silver recovery.

Results of test 30.

[Head contained Au, 0.04 ounce per ton; Ag, 81.4 ounces per ton.]

| Grade of products. | Assay of products. | | Recoveries. | |
|--------------------|------------------------|------------------------|------------------|------------------|
| | Au. | Ag. | Au. | Ag. |
| | <i>Ounces per ton.</i> | <i>Ounces per ton.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Concentrate..... | 0.2 | 507.8 | 67.5 | 85.0 |
| Tailing..... | Tr. | 13.0 | | 15.9 |

Flotation agents.

| | Pounds per ton. |
|--|-----------------|
| Sodium sulphide (1 hour)..... | 1.0 |
| Cleveland Cliffs Co. No. 2 wood creosote..... | 1.4 |
| Standard Chemical Co. No. 3 coal-tar creosote..... | .6 |

Remarks.—By doubling the amount of sodium sulphide used in the preceding test, No. 26, the recovery was increased a trifle. The grade of the product was about the same.

Results of test 37.

[Head contained Au, 0.04 ounce per ton; Ag, 81.4 ounces per ton.]

| Grade of products. | Assay of products. | | Recoveries. | |
|--------------------|------------------------|------------------------|------------------|------------------|
| | Au. | Ag. | Au. | Ag. |
| | <i>Ounces per ton.</i> | <i>Ounces per ton.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Concentrate..... | 0.2 | 601.8 | 57.5 | 85.0 |
| Tailing..... | Tr. | 13.4 | | 14.5 |

Flotation agents.

| | Pounds per ton. |
|--|-----------------|
| Sodium sulphide (2 hours)..... | 1.5 |
| Cleveland Cliffs Co. No. 2 wood creosote..... | 1.4 |
| Standard Chemical Co. No. 3 coal-tar creosote..... | 0.6 |

Remarks.—Increasing the amount of sodium sulphide and the length of time of agitation for sulphidizing gave a higher recovery than in previous tests. Also, the grade of the product was improved.

Results of test 39.

[Head contained Au, 0.04 ounce per ton; Ag, 81.4 ounces per ton.]

| Grade of products. | Assay of products. | | Recoveries. | |
|--------------------|--------------------|-----------------|-------------|-----------|
| | Au. | Ag. | Au. | Ag. |
| | Ounces per ton. | Ounces per ton. | Per cent. | Per cent. |
| Concentrate..... | 0.2 | 630.7 | 55.0 | 85.2 |
| Tailing..... | Tr. | 12.4 | | 13.5 |

Flotation agents.

| | Pounds per ton. |
|--|-----------------|
| Sodium sulphide (8 hours)..... | 1.5 |
| Cleveland Cliffs Co. No. 2 wood creosote..... | 1.4 |
| Standard Chemical Co. No. 3 coal-tar creosote..... | .6 |

Remarks.—The results indicated that lengthening the time of agitation for sulphidizing to eight hours and keeping the amount (1.5 pounds per ton) of sodium the same, gave a recovery and grade about the same as in the preceding test, No. 37.

Results of test 41.

[Head contained Au, 0.04 ounce per ton; Ag, 81.4 ounces per ton.]

| Grade of products. | Assay of products. | | Recoveries. | |
|--------------------|--------------------|-----------------|-------------|-----------|
| | Au. | Ag. | Au. | Ag. |
| | Ounces per ton. | Ounces per ton. | Per cent. | Per cent. |
| Concentrate..... | 0.16 | 551.7 | 55.0 | 92.2 |
| Tailing..... | Tr. | 12.7 | | |

Flotation agents.

| | Pounds per ton. |
|--|-----------------|
| Sodium sulphide (1 hour)..... | 2.0 |
| Cleveland Cliffs No. 2 wood creosote..... | 1.4 |
| Standard Chemical Co. No. 3 coal-tar creosote..... | .6 |

Remarks.—Sulphidization with 2 pounds of sodium sulphide seemed to give a higher recovery than the use of 1 pound. For comparison see test No. 30, on page 58. The grade of product was about the same.

Results of test 44.

[Head contained Au, 0.04 ounce per ton; Ag, 81.4 ounces per ton.]

| Grade of products. | Assay of products. | | Recoveries. | |
|--------------------|--------------------|-----------------|-------------|-----------|
| | Au. | Ag. | Au. | Ag. |
| | Ounces per ton. | Ounces per ton. | Per cent. | Per cent. |
| Concentrate..... | 0.2 | 655.5 | 55.0 | 90.2 |
| Tailing..... | Tr. | 12.3 | | 13.4 |

Flotation agents.

| | Pounds
per ton. |
|--|--------------------|
| Sodium sulphide (8 hours)..... | 2.0 |
| Cleveland Cliffs No. 2 wood creosote..... | 1.4 |
| Standard Chemical Co. coal-tar creosote..... | .6 |

Remarks.—Using 2 pounds of sodium sulphide and increasing the time of sulphidizing to eight hours seemed to improve the grade of product somewhat, but the recovery of silver remained about the same. For comparison see the preceding tests Nos. 39 and 41.

Results of test 51.

[Head contained Au, 0.04 ounce per ton; Ag, 81.4 ounces per ton.]

| Grade of products. | Assay of products. | | Recoveries. | |
|--------------------|----------------------------|----------------------------|------------------|------------------|
| | Au. | Ag. | Au. | Ag. |
| | <i>Ounces
per ton.</i> | <i>Ounces
per ton.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Concentrate..... | 0.27 | 888.1 | 51.3 | 81.5 |
| Tailing..... | Tr. | 15.7 | | 17.8 |

Flotation agents.

| | Pounds
per ton. |
|--|--------------------|
| Sodium sulphide (1 hour)..... | 2.0 |
| Cleveland Cliffs Co. No. 2 wood creosote..... | .75 |
| Standard Chemical Co. No. 3 coal-tar creosote..... | .3 |

Remarks.—Four times as much material (4,000 grams) was used as in test No. 41. (See p. 59.) The amount of sodium sulphide was the same as in that test, but only half the quantity of flotation oils was used. The results indicated a fair product but a relatively low recovery.

Results of test 52.

[Head contained Au, 0.04 ounce per ton; Ag, 81.4 ounces per ton.]

| Grade of products. | Assay of products. | | Recoveries. | |
|--------------------|----------------------------|----------------------------|------------------|------------------|
| | Au. | Ag. | Au. | Ag. |
| | <i>Ounces
per ton.</i> | <i>Ounces
per ton.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Concentrate..... | 0.25 | 819.5 | 57.5 | 93.1 |
| Tailing..... | Tr. | 5.8 | | 6.4 |

Flotation agents.

| | Pounds
per ton. |
|--|--------------------|
| Sodium sulphide (8 hours)..... | 2.0 |
| Cleveland Cliffs Co. No. 2 wood creosote..... | .75 |
| Standard Chemical Co. No. 3 coal-tar creosote..... | .3 |

Remarks.—Increasing the time of sulphidizing seemed to effect a higher recovery of the silver, but the grade of product remained about the same as for test No. 51. The results indicated a fair product and good recovery.

Results of test 53.

[Head contained Au, 0.04 ounce per ton; Ag, 81.4 ounces per ton.]

| Grade of products. | Assay of products. | | Recoveries. | |
|--------------------|--------------------------------|-----------------------------------|--------------------------|--------------------------|
| | Au. | Ag. | Au. | Ag. |
| Concentrate..... | <i>Ounces per ton.</i>
0.79 | <i>Ounces per ton.</i>
1,425.5 | <i>Per cent.</i>
49.4 | <i>Per cent.</i>
81.4 |
| Tailing..... | Tr. | 13.0 | | 15.2 |

Flotation agents.

| | Pounds per ton. |
|--|-----------------|
| Sodium sulphide (8 hours)..... | 2.0 |
| Sodium cyanide..... | .5 |
| Cleveland Cliffs Co. No. 2 wood creosote..... | .75 |
| Standard Chemical Co. No. 3 coal-tar creosote..... | .3 |

Remarks.—Comparison of these results with those of the preceding test, No. 52, seems to indicate that the addition of sodium cyanide improved the grade of the product but towered the recovery.

CONCLUSIONS.

On the basis of the experimental work on this ore the following conclusions seem justified:

1. This ore is amenable to flotation.
2. Pretreatment of the ore by sulphidization with sodium sulphide improves the silver recovery.
3. Treatment of flotation tailings on tables would seem advisable if much of the silver mineral is oxidized, and might eliminate the sulphidizing of the ore.
4. Use of sodium cyanide with the flotation agents seems to improve the grade of product.
5. It would seem advisable to make first a high recovery of silver without cyanide, then to retreat the first flotation concentrate with a small quantity of cyanide in order to improve the grade of the product.

TESTS OF A GOLD AND SILVER BEARING COPPER ORE.**DESCRIPTION OF ORE.**

The work on this ore included experiments to ascertain the flotation agents best suited to effecting a good grade of product and a high recovery of the copper as well as the gold and silver.

In general, the four samples of ore submitted for investigation contained principally copper in the form of chalcopyrite, with minor amounts of bornite and chalcocite, and pyrite in a quartzitic gangue. Other minerals identified were malachite and azurite. The quartz gangue of samples 2 and 4 was more or less coated with iron rust. Relatively large pieces of clean chalcopyrite or bornite were not com-

mon, most of the copper-bearing minerals being finely disseminated through the gangue. Chemical analyses were made of each sample with the following results:

Chemical analyses of the ore.

| | Sample
1. | Sample
2. | Sample
3. | Sample
4. |
|---|--------------|--------------|--------------|--------------|
| Cu.....per cent.. | 3.0 | 1.4 | 3.0 | 1.8 |
| Fe.....do..... | 5.7 | 3.2 | 5.4 | 3.6 |
| Pb.....do..... | Tr. | | | |
| Zn.....do..... | 2 | 4 | 1.1 | .4 |
| S.....do..... | 2.5 | 1.2 | 2.8 | 2.2 |
| Au.....ounces per ton.. | .12 | .2 | .32 | .65 |
| Ag.....do..... | 1.26 | .49 | 2.6 | 2.35 |
| CaO.....per cent.. | 6.0 | 7.4 | 7.2 | 4.3 |
| MgO.....do..... | .28 | .7 | .5 | .2 |
| Al ₂ O ₃do..... | 6.1 | 7.0 | 4.1 | 2.8 |
| SiO ₂do..... | 74.6 | 77.9 | 75.0 | 86.1 |

Examination of polished sections of this ore under the microscope indicated that it was composed chiefly of copper-bearing sulphides with chalcopyrite as the predominating mineral.⁴ The chalcopyrite was considerably fractured and was cut by numerous veins and stringers filled with chalcocite of the so-called "blue" variety. In addition to the sulphides, the ore carried copper in the form of the carbonates malachite and azurite, which were present in only small amounts in the samples examined microscopically. Small amounts of hematite and magnetite were also noted. Under the microscope the crushed ore grains, represented by samples of the concentration products, appeared free from gangue and the minerals separated from each other, nothing that would hinder successful flotation being found.

Plate VIII, *B* (p. 51), a photomicrograph of a polished surface of the ore, shows the relation of the chalcopyrite and chalcocite, the latter being the darker and occurring in fractures and veinlets in the chalcopyrite.

SCREEN ANALYSIS.

Owing to the similarity in copper and iron content, samples 1 and 3 were mixed, as well as 2 and 4. Head samples of these were assayed and the results taken as a basis for calculations in the ensuing tests. The results of these assays were as follows:

| | <i>Analyses of mixed samples.</i> | Samples
1 and 3. | Samples
2 and 4. |
|---------------------------|-----------------------------------|---------------------|---------------------|
| Copper.....per cent.. | | 3.0 | 1.7 |
| Iron.....do..... | | 5.0 | 3.1 |
| Gold.....ounces per ton.. | | .24 | .36 |
| Silver.....do..... | | 1.54 | 1.35 |

⁴ Microscopic examination made by R. E. Head, U. S. Bureau of Mines, Salt Lake City, Utah.

Samples of lots 1 and 3 were taken for a screen analysis, each screen product being assayed to determine the distribution of copper and iron. The screen analysis was made in a Tyler Ro-Tap shaking machine. Results are given in Table 2.

TABLE 13.—Screen analysis of samples 1 and 3 of the copper ore.

[Head contained Cu, 3 per cent; Fe, 5.5 per cent.]

| Size of screen opening. | | | Weights. | | Assays (per cent). | | Percentage of total head content. | | Cumulative per cent of total head content. | |
|----------------------------------|--------------|---------|-----------|----------------------|--------------------|-----|-----------------------------------|-------|--|-------|
| Mesh. | Millimeters. | Inches. | Per cent. | Cumulative per cent. | Cu. | Fe. | Cu. | Fe. | Cu. | Fe. |
| On..... | 28 | 0.589 | 0.232 | 65.2 | 2.7 | 5.5 | 58.9 | 65.3 | | |
| | 35 | .417 | .0164 | 8.8 | 74.0 | 3.0 | 5.5 | 8.7 | 67.6 | 74.0 |
| | 48 | .295 | .0116 | 6.2 | 80.2 | 3.2 | 5.2 | 6.7 | 5.8 | 74.3 |
| | 65 | .208 | .0082 | 4.7 | 84.9 | 3.3 | 5.5 | 4.7 | 4.7 | 79.0 |
| | 100 | .147 | .0058 | 3.2 | 88.1 | 3.5 | 6.1 | 3.7 | 3.5 | 82.7 |
| | 150 | .104 | .0041 | 3.0 | 91.1 | 3.7 | 5.7 | 3.3 | 3.1 | 86.0 |
| | 200 | .074 | .0029 | .8 | 91.9 | 3.5 | 6.3 | 1.0 | .9 | 87.0 |
| Through..... | 200 | .074 | .0029 | 7.2 | 99.1 | 4.2 | 6.7 | 10.7 | 8.4 | 97.7 |
| Loss or error by difference..... | | | | .9 | | | +2.3 | -0.4 | | 100.4 |
| Total Head..... | | | | 100.0 | | | 100.0 | 100.0 | | |
| | | | | | 3.0 | 5.5 | | | | |

EXAMINATION OF SCREEN PRODUCTS.

Products on 28 and 35 mesh screens.—A small amount of clean chalcopyrite could be seen, although most of the mineral was combined with quartz. A few pieces of quartz stained with oxides of copper and iron were likewise noted.

Products on 48 and 65 mesh screens.—Most of the chalcopyrite, pyrite, and quartz minerals were free, although a small number of mineral particles were still mechanically combined.

Products on 100-mesh screen.—Practically no mechanically combined mineral particles were visible. In addition to the minerals already mentioned malachite and bornite were noted. All particles seemed to have more of a flat and angular shape than those in the coarser products.

Products on 150 and 200 and through 200 mesh screens.—No combined mineral particles were seen.

This examination of the various screen products indicated the advisability of crushing the material through a 65-mesh screen for flotation, as practically no combined mineral was observed in the material on the 100-mesh screen.

RESULTS OF FLOTATION TESTS.

The same procedure was followed as in previous tests. The results from a few of the better tests on this ore are given below:

Results of test 12.

[Head contained Cu, 3 per cent; Fe, 5 per cent; Au, 0.28 ounce per ton; Ag, 1.58 ounces per ton.]

| Grade of products. | Assay of products. | | | | Recoveries. | | | |
|--------------------|--------------------|------------------|------------------------|------------------------|------------------|------------------|------------------|------------------|
| | Cu. | Fe. | Au. | Ag. | Cu. | Fe. | Au. | Ag. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Ounces per ton.</i> | <i>Ounces per ton.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Concentrate..... | 21.4 | 20.2 | 1.24 | 11.9 | 83.4 | 47.3 | 51.8 | 87.4 |
| Tailing..... | .8 | 2.9 | | | 21.0 | 50.5 | | |

Flotation agents.

| | Pounds per ton. |
|---|-----------------|
| Sodium hydroxide..... | 2.0 |
| Mixture (1:4) of Vancouver (dark) coal-tar creosote and wood alcohol..... | 1.5 |
| General Naval Stores Co. No. 5 s. d. pine oil..... | 0.08 |

Remarks.—A fair grade of copper concentrate and a good recovery of the copper and silver were obtained, but the gold recovery was low.

Results of test 14.

[Head contained Cu, 3 per cent; Fe, 5 per cent; Au, 0.28 ounce per ton; Ag, 1.58 ounces per ton.]

| Grade of products. | Assay of products. | | | | Recoveries. | | | |
|--------------------|--------------------|------------------|------------------------|------------------------|------------------|------------------|------------------|------------------|
| | Cu. | Fe. | Au. | Ag. | Cu. | Fe. | Au. | Ag. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Ounces per ton.</i> | <i>Ounces per ton.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Concentrate..... | 18.8 | 19.4 | 1.24 | 10.5 | 87.8 | 54.4 | 62.2 | 93.0 |
| Tailing..... | .5 | 2.9 | | .2 | 14.3 | 50.0 | | 10.8 |

Flotation agents.

| | Pounds per ton. |
|---|-----------------|
| Barrett Co. No. 633 coal-tar creosote..... | 0.8 |
| Pensacola Tar & Turpentine Co. No. 400 wood creosote oil..... | .6 |

Remarks.—The grade of the copper concentrate was fair and the recovery of the copper and silver was good, but the recovery of gold was low.

Results of test 16.

[Head contained Cu, 3 per cent; Fe, 5 per cent; Au, 0.28 ounce per ton; Ag, 1.58 ounces per ton.]

| Grade of products. | Assay of products. | | | | Recoveries. | | | |
|--------------------|--------------------|------------------|------------------------|------------------------|------------------|------------------|------------------|------------------|
| | Cu. | Fe. | Au. | Ag. | Cu. | Fe. | Au. | Ag. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Ounces per ton.</i> | <i>Ounces per ton.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Concentrate..... | 23.2 | 24.9 | 1.72 | 12.6 | 77.5 | 50.0 | 61.0 | 79.0 |
| Middling..... | 7.4 | 9.1 | .4 | 4.9 | 10.4 | 5.2 | 5.4 | 11.4 |
| Tailing..... | .2 | 2.9 | Tr. | .16 | 5.7 | 50.0 | | 8.4 |

Flotation agents.

| | Pounds
per ton. |
|---|--------------------|
| Sodium hydroxide..... | 1 |
| Mixture (1:4) of Vancouver coal tar (heavy) and wood alcohol..... | 1 |

Remarks.—The grade of the copper concentrate was good, but the recovery of copper was only fair.

Results of test 17.

[Head contained Cu, 3 per cent; Fe, 5 per cent; Au, 0.28 ounce per ton; Ag, 1.58 ounces per ton.]

| Grade of products. | Assay of products. | | | | Recoveries. | | | |
|--------------------|--------------------|------------------|------------------------|------------------------|------------------|------------------|------------------------|------------------------|
| | Cu. | Fe. | Au. | Ag. | Cu. | Fe. | Au. | Ag. |
| | <i>Per cent.</i> | <i>Per cent.</i> | <i>Ounces per ton.</i> | <i>Ounces per ton.</i> | <i>Per cent.</i> | <i>Per cent.</i> | <i>Ounces per ton.</i> | <i>Ounces per ton.</i> |
| Concentrate..... | 20.2 | 20.9 | 1.64 | 9.1 | 83.4 | 52.0 | 71.4 | 71.5 |
| Middling..... | 2.9 | 6.6 | .20 | 2.0 | 4.3 | 5.8 | 3.6 | 5.1 |
| Tailing..... | .4 | 3.0 | Tr. | .05 | 11.0 | 49.5 | | 26.0 |

Flotation agents.

| | Pounds
per ton. |
|---|--------------------|
| Sodium hydroxide..... | 1.0 |
| Imperial fuel oil..... | 1.0 |
| Pensacola Tar & Turpentine Co. No. 400 wood creosote oil..... | .8 |

Remarks.—The grade of the copper concentrate and the recoveries of copper, gold, and silver were fair.

CONCLUSIONS.

On the basis of the experimental work on this ore the following conclusions seem justified:

1. This copper ore, on the basis of the tests on the sample, is amenable to flotation.
2. Copper concentrate of good grade with high recoveries of the copper and silver are possible by flotation.
3. The silver and gold seem to follow the copper.
4. Mechanical agitation machines will give good results.
5. A pulp density of 3 of water to 1 of ore seems to be best.
6. An alkaline pulp responds more readily to flotation than an acid pulp.

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TECHNICAL PAPER 106. Asphyxiation from blast-furnace gas, by F. H. Willcox. 1916. 79 pp., 8 pls., 11 figs.

TECHNICAL PAPER 135. Bibliography of recent literature on flotation of ores, January to June, 1916, compiled by D. A. Lyon, O. C. Ralston, F. B. Laney, and R. S. Lewis. 1917. 20 pp.

TECHNICAL PAPER 136. Safe practice at blast furnaces, a manual for foremen and men, by F. H. Willcox. 1916. 73 pp., 1 pl., 43 figs.

TECHNICAL PAPER 143. The ores of copper, lead, gold, and silver, by C. H. Fulton. 1916. 41 pp.

TECHNICAL PAPER 149. Answer to questions on the flotation of ores, by O. C. Ralston. 1917. 30 pp.

TECHNICAL PAPER 152. The inflammability of aluminum dust, by Alan Leighton. 1918. 15 pp.

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TECHNICAL PAPER 156. Carbon monoxide poisoning in the steel industry, by J. A. Watkins. 1917. 19 pp., 1 fig.

TECHNICAL PAPER 157. A method for measuring the viscosity of blast-furnace slag at high temperatures, by A. L. Feild. 1916. 29 pp., 1 pl., 7 figs.

TECHNICAL PAPER 176. Bibliography of recent literature on flotation of ores, July 1 to December 31, 1916, by D. A. Lyon, O. C. Ralston, F. B. Laney, and R. S. Lewis. 1917. 27 pp.

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DEPARTMENT OF THE INTERIOR

HUBERT WORK, SECRETARY

BUREAU OF MINES

H. FOSTER BAIN, DIRECTOR

**THE ELECTROTHERMIC METALLURGY
OF ZINC**

BY

B. M. O'HARRA

**THIS BULLETIN REPRESENTS WORK DONE UNDER A COOPERATIVE
AGREEMENT BETWEEN THE BUREAU OF MINES, DEPART-
MENT OF THE INTERIOR, AND THE MISSOURI
SCHOOL OF MINES AND METALLURGY**



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THE ELECTROTHERMIC METALLURGY OF ZINC.

By B. M. O'HARRA.

INTRODUCTION.

Zinc smelting is frequently termed a backward art. The term is hardly true, for great progress has been made in recent years in the design and in the thermal efficiency of the retort furnace, in the quality of retorts, in the recovery of zinc, and in the ability to treat more impure and complex ores. The fact remains, however, that the peculiar physical and chemical properties of zinc have delayed such immense advances—large smelting units, high recovery, low-unit costs, and ability to treat low-grade ores—as have been made in the metallurgy of lead and copper.

In current retort-smelting practice, the ores, if carbonates, are calcined to remove carbon dioxide; or, if sulphides, are roasted to remove sulphur. The sulphur must be eliminated as completely as possible, usually to less than 1 per cent. This means that the roast must be continued for a long time at a high temperature, with resulting high cost and small capacity per furnace. The roasted ore is mixed with about 50 per cent of its weight of reducer, which may be fine coke, anthracite, or nonbituminous coal. This mixture is charged into horizontal fire-clay retorts. The retorts used in this country are about 8 inches in internal diameter by 4 feet in length and hold 50 or 60 pounds of ore. The diameter of the retort is limited by the time required for the heat to penetrate to the center of the charge; the length by the strength of the retort, which at the high temperature employed can not support the weight of the charge if the span is more than about 50 inches. The retorts are heated externally by coal, natural gas, or producer gas to a temperature of 1,200° C. or more. The zinc oxide is reduced to zinc, which is volatile at this temperature, and passes into a conical fire-clay condenser attached to one end of the retort, where it condenses to liquid zinc and is removed. Twenty-four hours are required for distillation, and because of the low heat conductivity of the retort walls and of the ore charge itself, the heat efficiency even of the most modern regenerative furnaces is only about 12 per cent. The retorts have a short life, 30 to 60 days, at the high temperature required; zinc

losses are heavy, usually 10 per cent or more; and the proportion of impurities in the retort charge must be carefully controlled to prevent the rapid corrosion of retorts.

For these reasons, much work has been done in an effort to find cheaper and more efficient methods of recovering zinc from its ores. Various methods have been proposed for smelting in the blast furnace. Some of these contemplated the production of zinc vapor and its condensation to liquid; others attempted to produce liquid zinc directly by smelting under pressure. It is now realized that fundamental difficulties render these proposals impracticable.

The electric furnace for smelting offers obvious advantages in the way of the efficient utilization of energy, large units, easy attainment of high temperatures, and the possibility of treating complex ores.

E. H. and A. H. Cowles first suggested a design for an electric furnace for the reduction of zinc ores and patented their process in 1885. Their attention was soon directed to other uses for their furnace and nothing further of importance was done in the electric smelting of zinc until about 1900. Several investigators then started work upon various processes and had more or less success. The chief difficulty they encountered was the impossibility of obtaining more than a small amount of the zinc as liquid metal because most of the zinc vapor condensed as blue powder. Interest in the electrothermic metallurgy of zinc increased, many patents on processes and furnace design were taken out, and the literature on the subject grew, until in 1914 and 1915 the problem seemed to be approaching solution. About this time the hydrometallurgy of zinc, the development of which had been lagging somewhat behind that of the electrothermic metallurgy, sprang into the limelight and a commercial process was soon perfected. This rapid development, perhaps aided by the war premium on the high-grade spelter produced by electrolytic deposition, distracted attention from electric smelting to some extent, though progress continued.

With the perfection and standardization of the electrolytic process has come a better realization of its limitations, especially the necessity for large-scale operation and the difficulty of obtaining high extraction. Because of these limitations the electrothermic metallurgy of zinc still has a field of its own and in fact promises to become a strong competitor of both hydrometallurgy and retort smelting except under conditions highly favorable to the latter methods.

Because of the imperfections of the retort and electrolytic processes and the promise that the electric furnace offers in overcoming these imperfections, the United States Bureau of Mines, in cooperation with the Missouri School of Mines and Metallurgy, has undertaken a study of the electrothermic metallurgy of zinc.

LITERATURE ON ELECTROTHERMIC METALLURGY OF ZINC.

As a first step in this investigation the literature was reviewed thoroughly. This literature is voluminous, but consists mostly of patents and of articles in various periodicals and publications of technical societies. No adequate compilation of the literature exists, so it is difficult for one not having much time at his disposal to acquire a comprehensive knowledge of what has been accomplished. In order that the results of the preliminary study may be available in convenient form for the industry in general, and especially for those who wish to undertake further investigations on their own account, a short review of the work done and the results obtained by previous investigators is presented.

For the purpose of this review, the literature has been drawn on so freely that proper acknowledgment of most of the sources is not possible. The reader is therefore referred for further information to the selected bibliography which is appended, or to the complete "Bibliography on the Electrothermic Metallurgy of Zinc," Bulletin of the Missouri School of Mines and Metallurgy, volume 6, No. 2.

TYPES OF ELECTRIC FURNACES.

The various furnaces that have been used for smelting zinc ores might be classified as direct-resistance, indirect-resistance, radiating-arc, and buried-arc types; as continuous and intermittent types; or as slagging and dry distillation types. The various processes may be divided into reduction processes, reducing the roasted ore with carbon, as in retort practice; and precipitation processes, in which the raw sulphide ore is treated with copper, iron, or lime and carbon. It has seemed better, rather than follow any of these classifications in describing the different furnaces and methods, to discuss them in the approximate chronological order of their development, beginning with the Cowles furnace, thus giving something of a historical view of the evolution of the whole subject.

ACKNOWLEDGMENTS.

The preparation of this bulletin was undertaken at the suggestion of Dr. C. H. Fulton, director of the Missouri School of Mines and Metallurgy and consulting engineer for the Bureau of Mines, and the writer is greatly indebted to him for his constant advice and helpful criticisms. Especial acknowledgment is made to Mr. F. T. Snyder for information concerning his work in British Columbia and at Chicago; to the Canadian Department of Mines for the use of W. R. Ingalls' unpublished report on the electric zinc smelting experiments of the Canadian Government; and to Mr. O. C. Ralston, of the Bureau of Mines, for information as to costs of power in the western United States.

COWLES BROTHERS' PROCESS.

The first electric furnace for the smelting of zinc ores was that of E. H. and A. H. Cowles, who patented their process in 1885. This furnace consisted of a cylindrical retort *A* (fig. 1) made of silica or other nonconducting material, and similar in form to the ordinary Belgian retort, surrounded by heat-insulating material, *B*. The rear end of the retort cylinder was closed by a carbon plate *C*, which formed the positive electrode, and the outer end by an inverted graphite crucible *D*, which formed the negative electrode and served as a condenser for the zinc vapor. The mouth of the crucible was

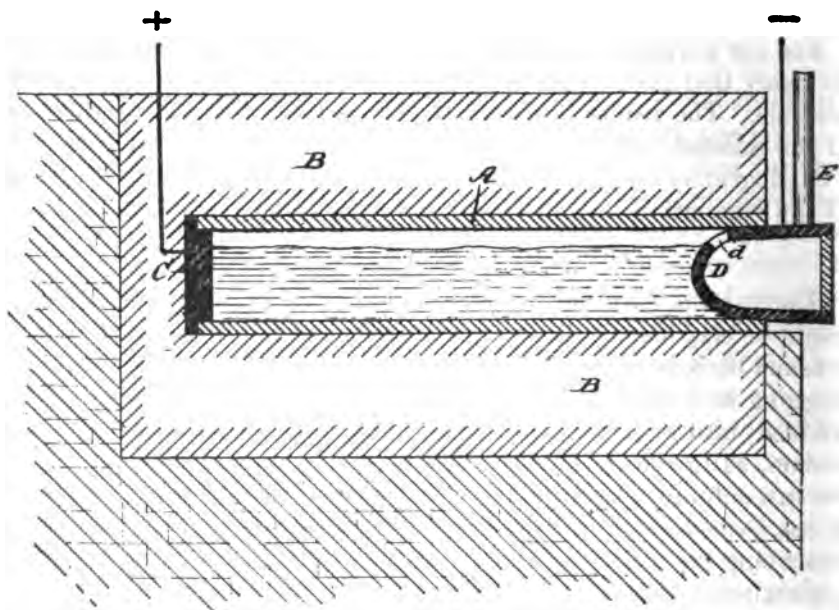


FIGURE 1.—Cowles Brothers' furnace (U. S. Patent 819795): *A*, Cylindrical retort; *B*, heat-insulating material; *C*, carbon plate, forming positive electrode; *D*, inverted graphite crucible, forming negative electrode; *d*, opening in upper side of crucible inside retort; *E*, pipe for escape of gases.

closed with a luting of clay, or otherwise, and the opening *d* in the upper side of the crucible inside the retort formed a passage for the zinc vapor from the retort into the condenser. The pipe *E* permitted the escape of gases from the condenser.

In operation, the roasted zinc ore was mixed with granular carbon, to serve as reduction material and to carry the current, and charged into the retort. The plug *D* was removed for the purpose. The retort was charged nearly full and a small space left along the top. The current was passed through the charge, heating it to distillation temperature; the zinc vapor was then condensed in the graphite

crucible and recovered. A modification was also described which provided more convenient means of charging and discharging the retort.

No further attempt was made to develop this furnace for smelting zinc ores, although the principle was used extensively in the production of aluminum and other difficultly reducible metals.

THE DE LAVAL PROCESS AND ELECTRIC ZINC SMELTING IN SCANDINAVIA.

EARLY HISTORY.

About 1898, C. G. P. de Laval became interested in the electric smelting of zinc ores and a few years later obtained patents on a furnace for this purpose. His first furnaces were of the continuous-smelting arc type shown in Figure 2.

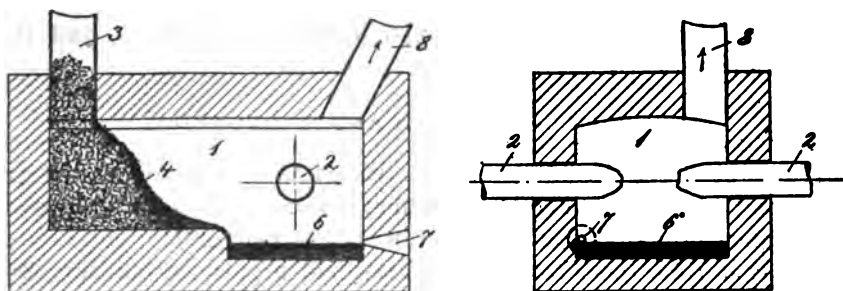


FIGURE 2.—De Laval radiating-arc furnace (U. S. Patent 736611): 1, Furnace chamber; 2, electric arc; 3, charge conduit; 4, surface of the pile of ore; 6, basin; 7, tap hole; 8, outlet for zinc vapor.

The pulverized roasted ore mixed with carbon, or raw sulphide ore mixed with metallic iron, was charged through the conduit 3 at one side of the rectangular furnace, so that it formed a pile sloping toward the source of heat. The surface of the pile 4 was heated by radiation from the electric arc 2, the zinc being reduced and volatilized. The residue melted and flowed down the slope, continually exposing a fresh surface to the action of the heat, and was collected in the basin 6, from which it could be tapped out through the tap hole 7.

The zinc vapor passed through the outlet 8 to a condenser. Later this furnace was modified by feeding the ore in at the back with a screw, thus avoiding the disturbance and evolution of dust from introducing a cold charge on top of the heated surface of the pile. The outlet for the zinc vapor in this improved furnace was placed between the pile of ore mixture and the heating arc to avoid superheating the vapor, which occurred with the original arrangement.

A company was organized in 1903 to exploit de Laval's patents and G. D. L. brand spelter from these works began to appear on the European market in 1904. This was a very pure spelter, containing less than 0.1 per cent total impurities.¹ (This metal was doubtless from the redistillation of scrap zinc or crude spelter, as the metal produced in smelting ores by the de Laval process is apt to be high in lead.) Plants were operated at Trollhattan, Sweden, and Sarpsborg, Norway. Metallurgical results as to condensation and metal recovery from ore were very poor and for several years most of the spelter produced came from the smelting of drosses and scrap zinc; however, by reason of the very cheap power available the company was enabled to continue operation in the face of these metallurgical difficulties and to continue experiments on the smelting of ores. The radiating-arc furnaces were later replaced by a buried-arc or arc-resistance type, but a full description of this type of furnace was never published. Between 1900 and 1911 the works at Sarpsborg and Trollhattan changed hands several times, and the operations do not seem to have been profitable.

REPORT OF MOULDEN AND HARBOARD.

In 1911 J. C. Moulden and F. W. Harboard reported on the process for an English company which had been formed to take over and enlarge the plants at Trollhattan and Sarpsborg.²

At that time the Trollhattan works had a furnace building 300 by 52 feet, equipped with 11 smelting furnaces of the arc-resistance type, 2 refining furnaces, and several arc furnaces of the old type which were not in operation. The works at Sarpsborg had 3 arc-type furnaces and 4 refining ones. The furnaces of the resistance type were of 350 horsepower each and had one large vertical electrode $2\frac{1}{2}$ square feet in cross section, passing through the roof, the other electrode being a carbon block embedded in the bottom of the furnace. The charge was introduced through an opening in the roof, the later furnaces being charged on one side only of the electrode. A furnace was being erected at that time with a continuous side feed which was expected to be an improvement. The capacity of each furnace was about 3 metric tons, and about 2.8 metric tons of ore were actually smelted per 24 hours.

Three-phase current was supplied to the works at 9,640 volts and transformed to a single-phase 100-volt current for use at the furnaces. The ore was charged with suitable flux and reduction material into the furnace, where most of the zinc and some of the lead

¹ Offerhaus, C., Zinc made in the electric furnace: *Electrochem. and Met. Eng.*, vol. 8, 1909, p. 54.

² *Engineering and Mining Journal*, Zinc smelting at Trollhattan; summary of a report by F. W. Harboard: Vol. 93, Feb. 10, 1912, pp. 314-315.

were volatilized and condensed. A large share of this condensed as blue powder and zinc oxide, containing about 54 per cent zinc and 20 per cent lead, which was mixed with fresh ore and recharged. When blue powder was resmelted, a greater proportion of liquid zinc was condensed than when ore alone was smelted.

Most of the lead that escaped volatilization was reduced to metal, carrying most of the silver. Some lead, silver, and zinc with the copper formed a matte, and the rest of the residue fused to a liquid slag. These fused products were tapped from the furnace through a tap hole near the bottom.

Mr. Harboard and his assistants made an extended test on Broken Hill slime containing about 36 per cent zinc, 24 per cent lead, and 30 ounces of silver per ton. Four furnaces worked on ore-powder mixture and three on ore mixture; they took the following charges: Ore-powder mixture—roasted Broken Hill slime, 100 kg.; blue powder, 200 kg.; coke dust, 25 kg.; lime, 5 kg. Ore mixture—Broken Hill slime, 300 kg.; calamine, 10 kg.; coke, 75 kg. At the end of the second day, the ore furnaces were making sufficient powder to supply the powder furnaces (which had been started on powder from the works stock) and no more powder was taken from stock. The furnaces were tapped every four hours for crude zinc and powder, and for slag, matte, and lead about every eight hours. During the early part of the run, the slags contained much zinc and lead, mainly because of imperfect separation of matte. The slags were too basic, from improper proportions of flux and reducing agent. Later on the slags improved, though two furnaces working on the same charge frequently gave entirely different slags, probably because of irregular mixing of the charge outside the furnace.

Improper mechanical facilities for changing electrodes were a serious cause of delay.

During 27.58 days of the experimental run, the seven furnaces smelted 518 metric tons of roasted Broken Hill slime, 19 tons of calamine, and 22.5 tons of blue powder from stock. They produced 160.8 tons of crude zinc and 36 tons of powder and at the end of the run had on hand 13.4 tons more of blue powder than at the beginning, the remainder of the powder having been resmelted. The crude metal obtained from this smelting averaged about 79 per cent zinc, 20 per cent lead, and 0.6 per cent iron. It was afterwards refined, giving 112.4 tons of spelter, 99.9 per cent pure, and 24.7 tons of lead. The lead tapped with the slag was remelted to remove the slag and yielded 41 tons of marketable bullion containing 141 ounces of silver per ton. There was also 17 tons of "leak lead," assaying 27 ounces of silver per ton, and 9 tons of skimmings containing zinc, lead, and silver.

The total input of metals was 204.04 tons of zinc, 128.35 tons of lead, and 15,750 ounces of silver, and the extraction in the form of metals was 130.46 tons of zinc, 94.94 tons of lead, and 7,230 ounces of silver, or a yield of 64 per cent, 73.99 per cent, and 45.9 per cent, respectively. Including the metal content of the powder, these became 73.4 per cent, 79.3 per cent, and 49.5 per cent. A considerable part of the losses of silver and lead were due to absorption by the furnace masonry.

Mr. Harboard considered that side charging would give much better results than charging through the roof. He was of the opinion that the recovery of metals could be much improved, but said that the difficulty of condensing the zinc vapor to liquid zinc was a great drawback.

The loss of zinc in refining 111,794 kg. of crude zinc at Trollhattan was 5.7 per cent. The average results of refining commercial spelter at Sarpsborg were stated to be as follows: Fine zinc produced, 83.9 per cent; powder, 9.8 per cent (corresponding to 8.8 per cent of metals); residues (lead, etc.), 2.2 per cent; charge in furnace, 1.1 per cent; impregnation in brickwork of furnaces and condensers, 0.5 per cent; unaccounted for 3.5 per cent.

A 20-day test at Sarpsborg smelting ore in the old arc-type furnaces gave results similar to those obtained at Trollhattan, but the power consumption was 70 per cent greater. The power used in the experimental run at Trollhattan averaged 2,078 kw. h. per ton of ores melted, including the power used for resmelting the blue powder but not that used for refining crude zinc, the cost of power being 41s. 3d. (\$10) per kilowatt year. The electrode consumption at Trollhattan was 31.51 kg. per metric ton (1,000 kg.) of ore smelted, and at Sarpsborg 40.57 kg. per metric ton of ore.

DEVELOPMENT IN RECENT YEARS.

In 1911 the production of spelter in Sweden and Norway was 6,575 long tons; in 1912, 8,000 tons; and in 1913, 17,000 tons. Part of this came from resmelting of dross and scrap, and part from the smelting of ores.³

A report of the directors of the Hydraulic Power and Smelting Co., issued in 1913,⁴ stated that out of seventeen new 1,000-horsepower furnaces and eight 500-horsepower furnaces planned for Trollhattan, thirteen of the former and all of the latter had been completed. The capacity of the zinc-refining works at Sarpsborg had been increased from 8,000 to 10,000 tons per year and the output

³ Ingalls, W. R., Electric zinc smelting: Trans. Am. Electrochem. Soc., vol. 25, 1914, p. 169.

⁴ Engineering and Mining Journal, Scandinavian electric zinc smelting: Vol. 96, Nov. 29, 1913, p. 1030.

contracted for at a profit. The smelting of ordinary roasted zinc ores still did not give a profit, but much better results were being obtained.

In 1921, W. R. Ingalls^{*} reported that the plants at Trollhattan and Sarpsborg and one or two smaller ones were in operation and that a new plant of an annual capacity of 60,000 metric tons of ore was being built at Glomfjord.

The Trollhattan metallurgists have given up the attempt to obtain liquid spelter from the first smelting operation, but have found that blue powder can be melted economically by heating it to the melting point of zinc and subjecting it to a rubbing action.

The present method of operation is to run the smelting furnace at a very high temperature so that both lead and zinc are completely volatilized. These are condensed as blue powder, which is then melted to a very leady spelter. This metal can be redistilled to yield pure spelter or it may be refined by simple gravity refining to yield an ordinary grade of spelter. Ore-smelting furnaces have been built and successfully operated with a capacity of 12,000 kg. of ore per 24 hours.

DE LAVAL'S CYCLONE FURNACE.

De Laval also obtained patents on a furnace which operated upon an entirely different principle. This furnace is illustrated in Figure 3.

The finely divided ore, flux, and reducer, 2, were fed into a circular furnace, 1, projected tangentially by a gas, such as CO, entering at 5,

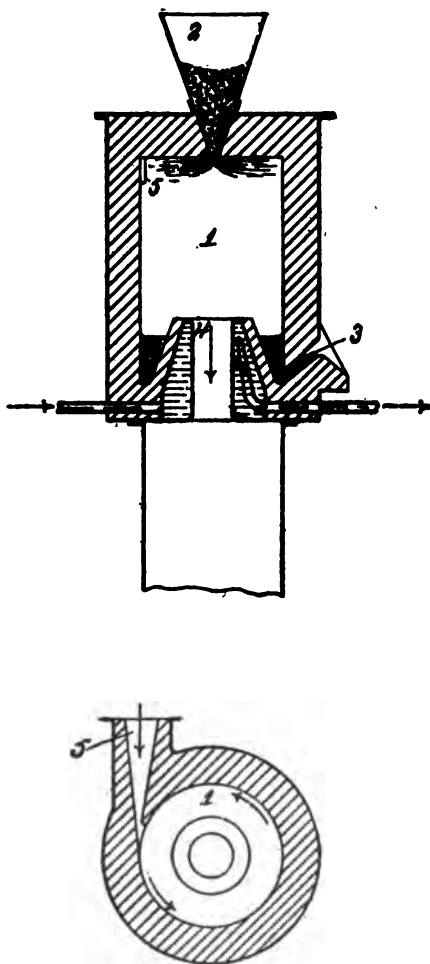


FIGURE 3.—De Laval's "cyclone" furnace (U. S. Patent 852440): 1, Circular furnace; 2, finely divided ore, flux, and reducer; 3, passage through which residue flowed out; 4, exit for zinc vapor and gases; 5, entrance for gas.

^{*} Ingalls, W. R., *Electrometallurgy of zinc*: Trans. Am. Electrochem. Soc., vol. 40, 1921, p. 245.

and were heated to a high temperature. The zinc was reduced and vaporized; the residue was projected against the periphery of the furnace, melted, and ran down the sides to an annular pool at the bottom and flowed out at 3; while the zinc vapor and gases passed out of the furnace by means of the opening 4 in the center. This furnace was tested in England and in Sweden, but a successful operation is not on record.

THE FURNACE OF CASORETTI AND BERTANI.

In 1900 Carlo Casoretti and Francesco Bertani described a continuous furnace for zinc ores (fig. 4) which combined fuel and electrical heating. According to their specifications the fine roasted ore was first mixed with coke and fluxes and briquetted with glucose solution. The bricks were then dried in the regenerator *r* of the smelting furnace and recrushed. The prepared ore was fed into the upper muffle *p* and worked down into the lower muffle *q*, both of which were heated by the coal fire *f*. After being heated in the lower muffle nearly to distillation temperature it was raked with the rake *p'* into the smelting chamber. This was heated by an electric current between the carbon electrodes *f'*. The zinc vapors and gases passed through the passage *v* into the condenser *c*, where the zinc was condensed and collected into suitable receivers. The remaining gases passed through the conduit *v* to the space beneath the muffle *q*, where they were burned. The slag was tapped from the bottom of the electric crucible into the chamber beneath, where it preheated the air for combustion of the CO gas from the condenser.

This furnace was expected to smelt ore with a power consumption of 2 horsepower hours per kilogram of zinc, plus a consumption of 15 per cent of coal.

STANSFIELD'S EXPERIMENTS.

Figure 5 is a sketch of a furnace in which Prof. Alfred Stansfield conducted some experiments in 1900 with a view to smelting Broken Hill lead-zinc ores to recover lead, zinc, and precious metals in the same operation. The charge was fed down the shaft C into the smelting chamber AB, where it was heated by the resistance of the slag bath to the passage of current between the two electrodes. Slag, bullion, and zinc vapor were produced and zinc condensed in the chambers F to K. Some molten zinc was obtained, but the greater part was condensed as blue powder.

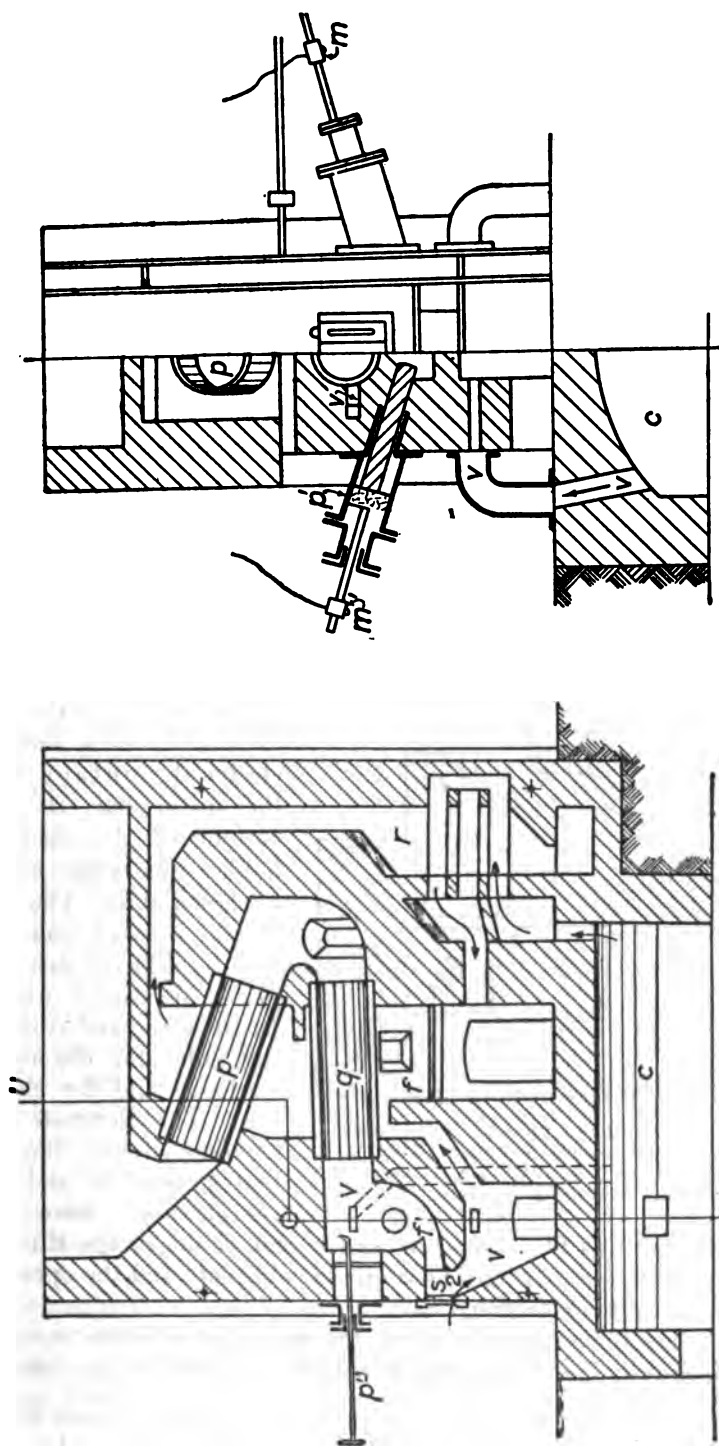


FIGURE 4.—Combined fuel and electric furnace of Casoretti and Bertani (British Patent 472, 1900) : *c*, Condenser ; *f*, electrodes ; *f'*, electrodes ; *f*, coal fire ; *p*, upper muffle ; *p'*, rake ; *q*, lower muffle ; *r*, regenerator ; *v*, passage for zinc vapor and gases ; *v'* conduit for gases from condenser.

SALGUES FURNACE.

In 1903, A. Salgues described two or three furnaces for the electrothermic treatment of zinc ores.⁶ One of these is shown in Figure 6. This is one of the earliest adaptations to zinc smelting of the buried-

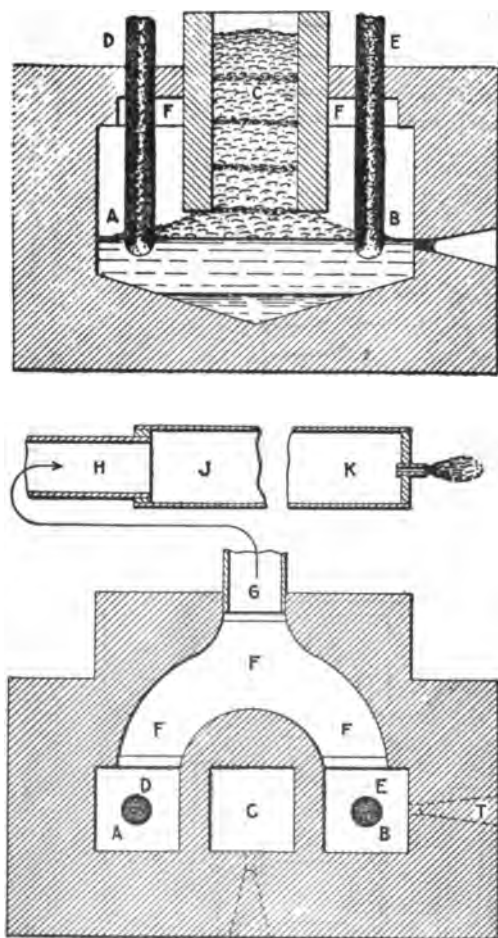


FIGURE 5.—Stansfield zinc furnace (from Stansfield, Alfred, *The electric furnace*, 1914, p. 318): A, B, Smelting chamber; C, shaft; D, E, electrodes; F, G, H, J, K, condensing chambers.

arc type of furnaces, variations of which have been used by so many of the later experimenters.

The smelting chamber, built of fire brick, inside an iron jacket cooled by water sprinklers K, was in two parts—A and B—to facilitate cleaning and repairing. It was provided with a tap hole, D, a flue, C, for the passage of zinc vapor to the condenser, and two charging and poking holes, E. The lower electrode G was embedded in the base of the furnace, making electrical contact with the metal bar H. The bottom of the electrode holder was water cooled as shown at L. The roof of the furnace was covered by a heavy cast-iron plate, M, cooled with a jet of water and containing holes for the upper electrode and for charging. The electrode was surrounded by the asbestos ring N and the charge holes were covered

with the lids O. Any zinc vapor that could escape through remaining crevices was condensed on the cold plate and the openings were sealed by the condensed zinc.

The charge, consisting of roasted zinc ore mixed with the necessary carbon for reduction, was introduced through the charge holes E

⁶ Salgues, A., [*The electrometallurgy of zinc*]: Bull. Soc. Ing. Civ. France, 1903, p. 174; Stansfield, Alfred, *The electric furnace*, 1914, p. 315.

and surrounded the upper adjustable electrode. The furnace operated continuously, producing zinc vapor, lead bullion, matte, and slag. Salgues recognized the necessity for prereduction of the charge if good condensation of liquid spelter was to be obtained.

Salgues's experiments were carried on at Champagne, Ariegne, France, with a modified carbide furnace of 100-kw. capacity. Operating on 40 to 45 per cent zinc ores fed into the furnace cold, he obtained a yield of 5 kg. of zinc per kilowatt day, or 4,355 kw. h. per ton of ore.

JOHNSON PROCESS.

EARLY EXPERIMENTS.

Woolsey McA. Johnson began work on the electric smelting of zinc ores in 1903, while he was metallurgist for the Lanyon Zinc Co. His first efforts followed closely the metallurgy of the retort process, as the Cowles Brothers had done in 1885, but on a larger scale.

His furnace for this purpose (see fig. 7) was a long, box-shaped retort, 1, built of fire brick or other suitable refractory, with an opening, 4, at one end for the passage of the zinc vapor and gases to the condenser. Graphite blocks, 5 and 6, set on opposite sides of the furnace at each end, with their faces flush with the inside of the furnace walls, were connected by means of graphite or metal rods, 7, projecting through the walls to the external electric circuit. Loose, granular coke, 12, or similar material was piled in each end of the chamber between the graphite blocks. The two piles of coke thus formed the electrodes between which the current passed, and the charge itself became the resistor in which the heat was developed. The floor of the furnace was first covered with a protective layer of refractory material, 11, acid or basic, according to the charge to be smelted. Over

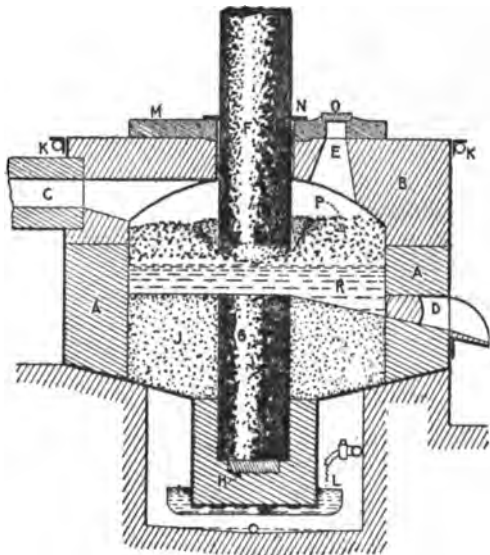


FIGURE 6.—Salgues's zinc furnace (from Stansfield, Alfred, *The electric furnace*, 1914, p. 315): A, B, Walls of smelting chamber; C, flue; D, tap hole; E, charging and poking holes; F, upper electrode; G, lower electrode; H, metal bar; K, water sprinklers; L, water for cooling bottom of electrode holder; M, cast-iron plate; N, asbestos ring; O, lids; P, charge; R, molten metal and slag.

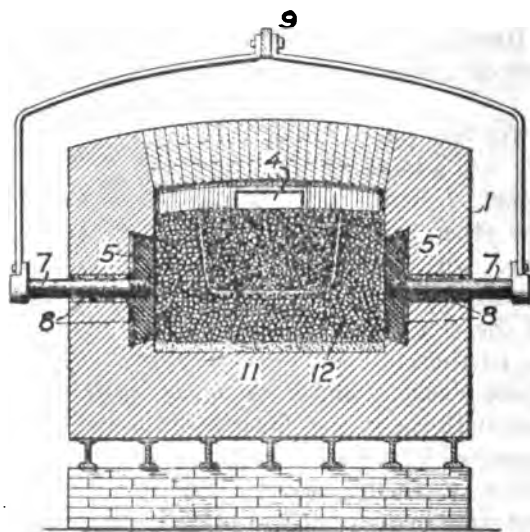
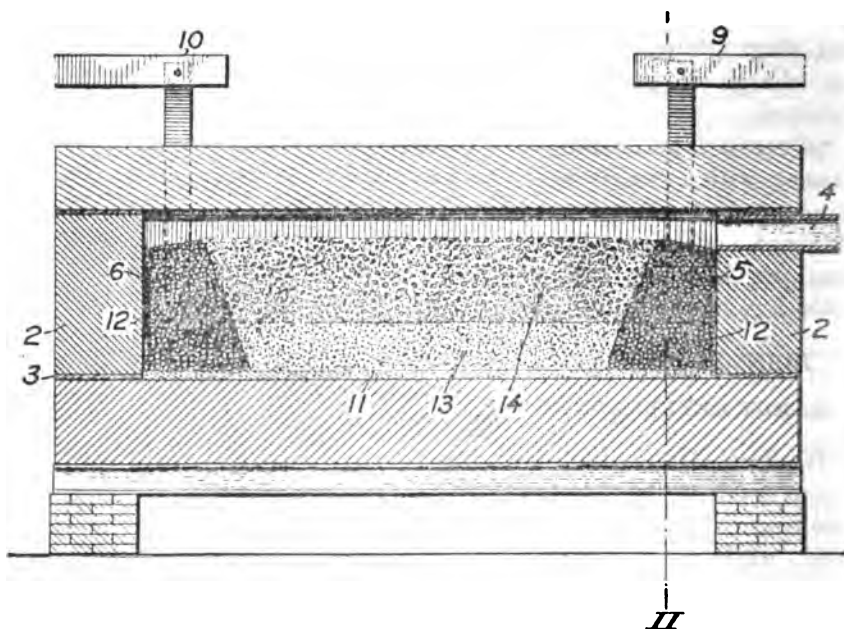


FIGURE 7.—Early furnace of W. McA. Johnson (U. S. Patent 814050): 1, Retort; 2, furnace walls; 4, opening for passage of zinc vapor and gases to condenser; 5, 6, graphite blocks; 7, graphite rods; 8, insulation; 9, 10, electric connection; 11, refractory material; 12, coke; 13, mixture of high-grade ore and coke; 14, mixture of low-grade ore and coke.

this refractory material and extending upward against the side walls was placed a portion, 13, of the charge, consisting of high-grade roasted ore mixed with sufficient high-resistance coke to act as reducer and to carry the electric current. The space above and within this high-grade charge was filled with a mixture of low-grade ore and coke, 14. In this way the high-grade ore protected the furnace walls from the corrosive action of the low-grade ore. As described in the patent, it was necessary to remove one or both end walls to rake out the residue and introduce a new charge. These walls were made of single blocks, or doors, which could be easily removed and could be luted in when the furnace was in operation. Other openings could be arranged if desired.

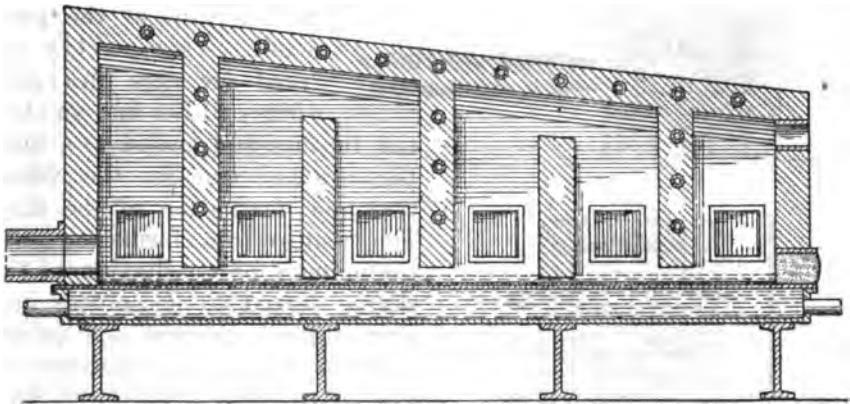


FIGURE 8.—An early Johnson condenser (U. S. Patent 851520).

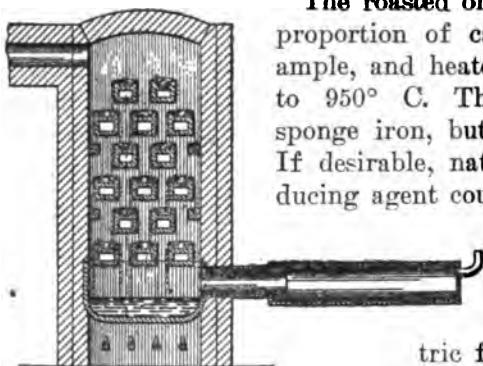
CONDENSATION DIFFICULTIES.

Johnson had the usual difficulties with condensation and much of his effort in the following few years was turned toward solving these difficulties. He tried condensers of several designs and obtained patents on them. Two are shown in Figures 8 and 9. In both of these arrangement was made to conduct the vapor through a tortuous passage with artificially cooled walls, and to bring it into contact with a body of molten zinc. In the condenser shown in Figure 9 there was a continual shower of molten zinc overflowing from the receptacles A. This shower was kept up, when necessary, by pumping molten zinc from the sump to the top of the condenser. In practice this was found to be less satisfactory than the stationary bath of zinc.

PREREDUCTION OF THE CHARGE.

The desirability of a prereduction of the charge, both as a help in obtaining a proper condensation of zinc and as an economy of electrical power, was early realized.

The earlier method of carrying this out is described in United States patent 868345.⁷ The ore, after suitable crushing, was roasted at a low temperature, about 850° to 900° C., in any suitable roaster. It was not necessary to reduce the sulphur below about 3 per cent, so that if the roasting was done in a muffle-type roaster a rich SO₂ gas, very desirable for sulphuric acid making, could be produced.



The roasted ore was mixed with a suitable proportion of carbon, 35 per cent, for example, and heated to a temperature of 850° to 950° C. This reduced iron oxide to sponge iron, but left zinc oxide unaffected. If desirable, natural gas or some other reducing agent could be substituted for carbon

in this step of the procedure. This partly reduced ore was charged while still hot into an elec-

tric furnace and heated to a temperature sufficient for the reduction and distillation of the zinc. The sponge iron formed in the preliminary part reduction of the charge served to decompose any zinc sulphide present and to reduce zinc oxide; the importance of this latter reaction depended upon the amount of iron in the ore.

By following this procedure as outlined, the CO₂ and CO produced in the reduction of iron are eliminated in the preliminary stage; consequently in the final stage a rich zinc vapor is formed which is more easily condensed to liquid metal.

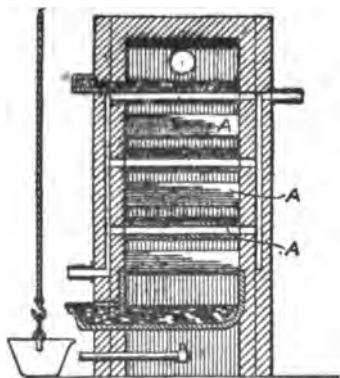


FIGURE 9.—Another early Johnson condenser (U. S. Patent 902534): A, Receptacles for molten zinc.

TYPE OF FURNACE USED IN LATER EXPERIMENTS.

While the condensation problem was being worked out, various types of furnace were experimented with, until that used in the later larger scale tests was developed. This final type, as first described, is illustrated in Figure 10. It was of the continuous-smelting, buried-arc class, similar to the Scandinavian and Salgues furnaces, producing spelter, bullion, matte, and slag in the same operation.

⁷ Johnson, W. McA., Process of treating ferruginous blende, U. S. Patent 868345, Oct. 15, 1907.

The electrodes 3, projecting vertically through the roof of the furnace 1 were connected to one terminal of the external electric circuit while the electrode 5, embedded in the hearth of the furnace, was connected to the opposite terminal. The charge was introduced through the hopper 7. Metal and matte were removed through the tap hole 9 and slag through the slag tap 4. The zinc vapor and gases passed by means of the conduit 12 to the column of coke 13 which was so arranged that it could be heated to incandescence, preferably by electrical means, and thence to the condenser 21.

In the operation of this furnace the ore was charged into the furnace with only sufficient carbon to reduce the valuable metallic constituents, leaving no excess. Proper fluxes were added to form a slag fusible at $1,100^{\circ}$ to $1,175^{\circ}$ C. Under these conditions a con-

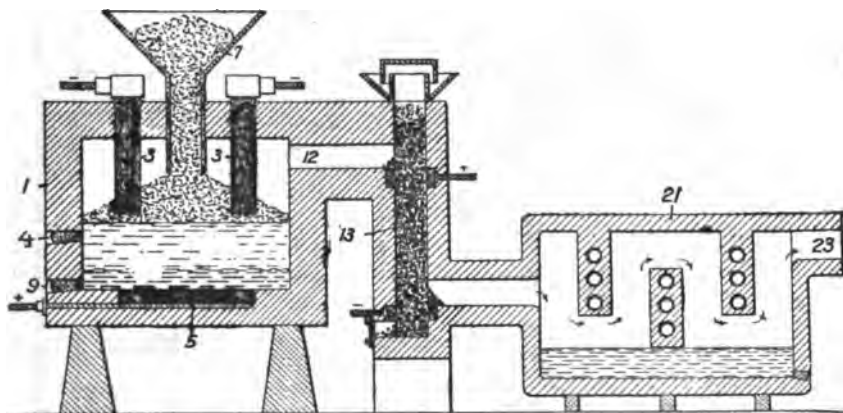


FIGURE 10.—Johnson continuous zinc furnace (U. S. Patent Reissue 13208) : 1, Furnace; 3, electrodes; 4, slag tap; 5, electrode; 7, hopper; 9, tap hole; 12, passage to condenser; 13, coke; 21, condenser.

siderable proportion of CO_2 was formed in the smelting furnace, and the hot-coke column was introduced to reduce the CO_2 to CO and prevent oxidation of the zinc vapor.

Figure 11 shows the construction of a later form of this furnace. It was found possible at a later stage of the experiments to do away with the coke column.

EFFECTS OF SULPHUR ON THE CONDENSATION OF ZINC.

In this later work, Johnson found that sulphur in the electric furnace, either as free sulphur, carbon disulphide, sulphur dioxide, or sulphur trioxide, tended to sulphidize the zinc vapors and form blue powder. He then proposed to modify his procedure to avoid this trouble.* He so conducted the roasting operation at low tem-

* Johnson, W. McA., Electric reduction of zinc ores, U. S. Patent 1150271, Aug. 17, 1915.

perature as to reduce the sulphur to the desired minimum and to decompose all sulphates other than those of lead, calcium, and barium. The roasted ore was then mixed with reducing agents, say 15 to 20 per cent coal, and fluxes, and treated in the preheater in such a way as

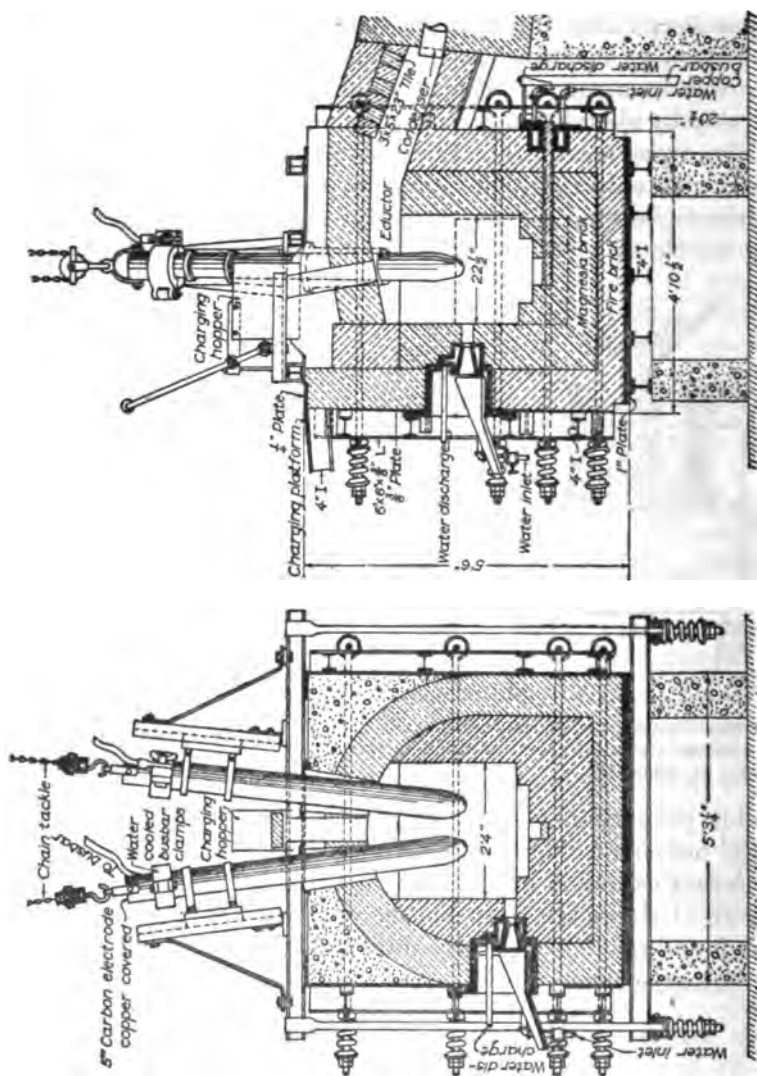


FIGURE 11.—A later form of the Johnson continuous furnace (from Engineering and Mining Journal, vol. 96, 1913, p. 965).

to reduce the sulphates present to sulphides and to reduce iron oxides and copper oxides to metal. Any muffle-type roasting furnace could be used as a preheater. The charge to the electric furnace was so proportioned that the percentage of iron and copper present should be from five to seven times that of sulphur. These metals decomposed any sulphides present in the charge, forming copper and iron

subsulphides. Because of the excess of metallic copper and iron present, no sulphur vapors could be set free and sulphidization of the zinc vapor was prevented.

RESULTS OF TESTS OF JOHNSON'S PROCESS.

By 1912, Johnson's process had begun to give very promising results. During 1912, 1913, and 1914 he made continuous runs in furnaces producing several hundred pounds of zinc per day, and published the results of several of these, which were more or less satisfactory. In 1914, he read a paper before the Canadian Mining Institute, giving a description of the process and the furnace as developed at that time, and the metallurgical results of a typical furnace run. The following description is taken from that paper.⁹

The process, as applied to zinciferous copper-lead ores, is briefly as follows:

(1) Rough roasting, to expel a large part of the sulphur until it does not exceed, say, from 4 per cent to 6 per cent. This roasting is an ordinary operation and can be done in general practice at a cost of less than \$1 per ton. If the original ore or concentrate is high enough in sulphur, it can be roasted by sulphuric acid manufacturers at little or no charge to the metallurgist and the cost of this preliminary treatment would then be reduced to that of freight and handling only.

(2) Preheating and prereducing the ore with admixture of soft coal, so as to reduce much of the iron oxide to metallic iron (sponge or pyrophoric iron), which is necessary in the subsequent treatment in the electric furnace. More than half of the heat required for the chemical reactions is supplied very cheaply in this preheater.

(3) Final reduction to metals in the continuous zinc furnace to decompose the sulphides of zinc and lead, forming (a) metallic zinc which is volatilized and collected in the condenser, (b) copper matte, and (c) lead bullion which are collected in the bottom of the furnace, and (d) molten slag which floats on the top of the matte, is the most important step in the process.

The present Johnson continuous electric zinc furnace, as installed at Hartford, has an exterior form of a cube about 5 feet on the edge, resting on a heavy iron plate supported by four pliers. The bricks of the furnace are bound together by 4-inch I beams and tie-rods having heavy steel springs at the ends to compensate for expansion and contraction. The outer construction of fire brick surrounds the interior hearth or crucible of magnesite brick. This crucible, in which the smelting reactions occur, has interior dimensions of 35 inches by 40 inches by 30 inches in height. At the bottom of the crucible is a water-cooled tap hole for the molten matte and lead bullion, and at a proper distance from the bottom is another water-cooled tap hole for the removal of the molten slag. Four copper-covered carbon electrodes 5 inches in diameter are attached to the bus bars by water-cooled clamps. These electrodes, supported by chain tackles, project through the arched roof of the furnace into the crucible. Hoppers for the introduction of ore, flux, and reducing material are situated at the top of the furnace near the electrodes. At one side near

⁹Johnson, W. McA., The commercial aspect of electric zinc-lead smelting: Trans. Canadian Min. Inst., vol. 17, 1914, pp. 107-129.

the top is a brick-lined flue to carry the vaporized zinc and furnace gases to the condenser in which the zinc is collected in molten metallic form, and the escaping gas, chiefly carbon monoxide, is burned in a steady flame as it escapes through small orifices into the air. * * *

The ores to be treated in the electric furnace are crushed to proper size, roasted to proper content of sulphur, and blended to give a mixture of the following range of composition: Zinc, 15 to 40 per cent; lead, 5 to 30 per cent; copper, 1 to 5 per cent; iron, 10 to 20 per cent; sulphur, 3 to 7 per cent; silica, 10 to 30 per cent; alumina, 3 to 5 per cent; and lime, 3 to 7 per cent. A type mixture would contain: Zinc, 30 per cent; lead, 10 per cent; copper, 2 per cent; iron, 15 per cent; sulphur, 4 per cent; silica, 15 per cent; and lime, 6 per cent. This blended material is then mixed with about 15 per cent of soft coal and carefully charged into a muffle preheater of the Hasenclever type, fired with soft coal, and carefully deoxidized at a final temperature of from 850° C. to 1,040° C., which is just below the reduction temperature of zinc oxide. The deoxidized material is then fed continuously into the electric furnace, which is of the buried-arc type. The electrodes are submerged in the slag and the charge. Care must be taken that there is no local heating, called "hot spots," or local cooling, called "cold spots" or "freezing," of the slag in the fusion zone, which should have a temperature of from 1,225° C. to 1,250° C.

In the preheating operation the following chemical reactions occur:

- (1) $\text{CaSO}_4 + 2\text{C} + \text{H}_2 = \text{CaS} + \text{CO} + \text{CO}_2 + \text{H}_2\text{O}.$
- (2) $\text{BaSO}_4 + 2\text{C} + \text{H}_2 = \text{BaS} + \text{CO} + \text{CO}_2 + \text{H}_2\text{O}.$
- (3) $\text{ZnSO}_4 = \text{ZnO} + \text{SO}_2 = \text{ZnO} + \text{SO}_3 + \text{O}.$
- (4) $2\text{Fe}_2\text{O}_3 + 3\text{C} + 2\text{H}_2 = 4\text{Fe} + 2\text{CO} + \text{CO}_2 + 2\text{H}_2\text{O}.$
- (5) $4\text{CuO} + 2\text{C} + \text{H}_2 = 4\text{Cu} + \text{CO} + \text{CO}_2 + \text{H}_2\text{O}.$

In the above reactions, the carbon is derived from solid carbon, carbon monoxide, and hydrocarbons. The hydrogen comes from the hydrocarbons. The relative percentages of CO and CO₂ in the furnace gases will depend on the physicochemical conditions of the charge. There are also other chemical reactions which need not be given here.

From 90 to 99 per cent of the Fe₂O₃ is reduced to the metallic state. ,

The charge, after the desired reactions have taken place in the preheater, is fed through the hoppers of the electrical furnace direct to the smelting zone in which the following chemical reactions occur, under the influence of heat generated by the passage of the electric current:

| | Temperature at which
reactions begin, °C. |
|--|--|
| 6. $\text{ZnO} + \text{C} = \text{Zn} + \text{CO}$ | 1,040 |
| 7. $\text{ZnS} + \text{Fe} = \text{Zn} + \text{FeS}$ | 1,175 |
| 8. $\text{ZnS} + 2\text{Cu} = \text{Zn} + \text{Cu}_2\text{S}$ | 1,150 |
| 9. $\text{CO}_2 + \text{Fe} = \text{FeO} + \text{CO}$ | 1,000 |
| 10. $\text{ZnO} + \text{Fe} = \text{FeO} + \text{Zn}$ | 1,200 |
| 11. $3\text{Fe} + \text{SO}_2 = \text{FeS} + 2\text{FeO}$ | 1,200 |
| 12. $2\text{ZnS} + \text{C} = 2\text{Zn} + \text{CS}_2$ | 1,350 |
| 13. $2\text{Fe} + \text{CS}_2 = 2\text{FeS} + \text{C}$ | 1,100 |
| 14. $\text{ZnO} + \text{CO} = \text{Zn} + \text{CO}_2$ | 900 |

As a result of these reactions, the sulphur combines first with the reduced copper and then with reduced iron to form copper matte, which collects some silver and gold and settles to the crucible. Similarly the reduced lead, in molten metallic form, collects the rest of the silver and gold, forming a base bullion, which settles in the crucible and collects below the molten matte. The silica, lime, iron oxide (and alumina and magnesia if present) combine to form the slag, which being specifically lighter, floats on top of the matte.

The zinc, which is reduced to the metallic form, is immediately vaporized at the temperature of the furnace and passes to the condenser at the side, where it is condensed and collected in a molten condition. The slag is deoxidized and desulphurized and the matte is made so metallic that no sulphur vapor is given off at the temperature of the fusion zone. Under this condition the gas from the furnace is so pure that when it reaches the condenser the zinc vapor changes directly from the gaseous condition to the liquid condition, and molten zinc is formed with little or no blue powder.

The condenser consists of elongated iron vessels lined with fire brick having movable covers to facilitate cleaning, and capable of being heated or cooled. If sufficient service be provided and enough time allowed, the zinc vapor will condense satisfactorily to the molten metal. The uniform manner in which the chemical reactions occur in the smelting furnace is shown by the steadiness of the flame of burning carbon monoxide as it escapes into the outer air from the prolongs of the condensing vessels. This flame has the appearance of that of a Bunsen burner in a laboratory.

Johnson emphasized the fact that the field of his furnace lay especially in the treatment of complex ores of zinc, lead, copper, and precious metals, which could not be treated by the retort process, or could only be treated at a higher cost than the purer high-grade ores. In the electric furnace smelting of preheated ore, the chief consumption of power was in the reduction of zinc oxide. The lower-grade ores could, therefore, be treated more cheaply per ton of ore than high-grade ores; and the lead, copper, and precious metals could be recovered at the same time as the zinc.

Table 1¹⁰ gives the metallurgical balance sheet of run No. 1 in Johnson's electric furnace No. 26, January 3 to February 13, 1914.

TABLE 1.—*Metallurgical balance sheet of run No. 1, in Johnson electric furnace No. 26, January 3 to February 13, 1914.*^a

(This balance sheet is from the mixing floor to the electric furnace products.)

| ZINC. | | | |
|--------------------|-----------------|----------------------|-----------|
| | Charge chiefly. | Quantity,
pounds. | Per cent. |
| Colorado ore | ----- | 27,847 | 31.6 |
| N. H. ore | ----- | 47,248 | 42.4 |
| Y. P. slimes (raw) | ----- | 5,430 | 33.7 |
| | | | |
| | | | 22,192 |

^a The energy consumption per ton of charge was 1,490 kw. h. Average voltage, 25; average amperes, 2,800.

Theoretical energy consumption with efficient prereducing, per short ton:

(Zn. 29 per cent) calcined zinc ore 264 kw. h.

(Zn. 44 per cent) calcined zinc ore 565 kw. h.

(Zn. 70 per cent) calcined zinc ore 900 kw. h.

Electrode carbon consumed in 41 days, 185 pounds, or 4.5 pounds per day; per ton of ore treated, 6.05 pounds.

¹⁰ Johnson, W. McA., The commercial aspect of electric zinc-lead smelting: Trans. Canadian Min. Inst., vol. 17, 1914, pp. 120-122.

| Accounted for. | Quantity,
pounds. | Per cent. |
|------------------|----------------------|-----------|
| Spelter _____ | 17,245 | 95.2 |
| Flue dust _____ | 2,562 | 6.2 |
| Matte _____ | 3,478 | 9.1 |
| Lead dross _____ | 177 | 4.0 |
| Slag _____ | 45,850 | 6.9 |

In furnace bottom not yet recovered _____

Spelter recovery^b _____

Zinc recovered in condenser _____

Total zinc recovery _____

Condensation factor^b _____

LEAD.

| Charge chiefly. | Quantity,
pounds. | Per cent. |
|--------------------------|----------------------|-----------|
| Colorado ore _____ | 27,847 | 12.77 |
| N. H. ore _____ | 27,248 | 6.15 |
| Y. P. slimes (raw) _____ | 5,430 | 8.63 |

| Accounted for. | Quantity,
pounds. | Ounces
per ton. | Metal
content. |
|---|----------------------|--------------------|-------------------|
| Bullion _____ | | | 2,437.0 |
| Matte _____ | 3,478 | 9.07 | 315.3 |
| Lead dross _____ | 177 | 60.0 | 108.2 |
| Slag _____ | 45,850 | 0.153 | 70.4 |
| Spelter _____ | 17,245 | 2.03 | 351.5 |
| Flue dust _____ | 2,562 | 14.5 | 371.0 |
| In furnace bottom not yet recovered _____ | | | 2,042.6 |
| | | | 5,694.0 |

COPPER.

| Charge chiefly. | Quantity,
pounds. | Per
cent. | M
con
tent |
|--------------------------|----------------------|--------------|------------------|
| Colorado ore _____ | 27,847 | 1.69 | 47 |
| N. H. ore _____ | 27,248 | 0.64 | 171 |
| Y. P. slimes (raw) _____ | 5,430 | 0.37 | 30 |
| | | | 674 |

^b "Spelter recovery" is calculated in the usual commercial way—i. e., on the basis of 100 per cent of zinc in the spelter. The "condensation factor" is similarly calculated.

| Accounted for. | Quantity,
pounds. | Per cent. | Metal
content,
pounds. | Distribu-
tion of
metal,
per cent. |
|-------------------------------------|----------------------|-----------|------------------------------|---|
| Iron | 3,478 | 12.62 | 439.0 | 65.2 |
| Lead | 45,850 | 0.239 | 109.3 | 16.2 |
| Flue | 17,245 | 0.05 | 8.6 | 1.3 |
| Slag | 2,594 | 0.07 | 1.8 | 0.3 |
| In furnace bottom not yet recovered | | | 116.0 | 17.0 |
| | | | <hr/> 674.7 | <hr/> 100.0 |

SILVER.

| Charge chiefly. | Quantity,
pounds. | Ounces
per ton. | Metal
content,
ounces. |
|------------------|----------------------|--------------------|------------------------------|
| do (brown ore) | 13,288 | 22.0 | 146.0 |
| do (black ore) | 12,664 | 9.0 | 61.5 |
| No. 1 | 16,427 | 6.2 | 51.0 |
| No. 2 | 4,054 | 2.6 | 5.3 |
| Slimes (reduced) | 1,831 | 4.69 | 4.3 |
| Mix | 6,290 | 5.35 | 16.8 |
| | | | <hr/> 284.9 |

| Accounted for. | Quantity,
pounds. | Per cent. | Metal
content,
pounds. | Distribu-
tion of
metal,
per cent. |
|-------------------------------------|----------------------|-----------|------------------------------|---|
| Iron | 2,437 | 116.5 | 142.9 | 50.7 |
| Lead | 3,478 | 21.8 | 37.9 | 13.5 |
| Lead dross | 177 | 87.0 | 7.7 | 2.7 |
| Flue | 17,245 | 0.45 | 3.9 | 1.4 |
| Slag | 2,594 | 0.45 | 0.6 | 0.3 |
| In furnace bottom not yet recovered | 45,850 | 0.78 | 17.9 | 6.4 |
| | | | <hr/> 74.0 | <hr/> 30.0 |
| | | | <hr/> 284.9 | <hr/> 100.0 |

GOLD.

| Charge chiefly. | Quantity,
pounds. | Ounces
per ton. | Metal
content. |
|----------------------|----------------------|--------------------|-------------------|
| Colorado (brown ore) | 13,288 | 0.04 | 0.268 |
| Colorado (black ore) | 12,664 | 0.44 | 3.008 |
| H. No. 1 | 16,427 | --- | --- |
| H. No. 2 | 4,054 | --- | --- |
| P. Slimes (reduced) | 1,831 | 0.05 | 0.046 |
| Mix | 6,290 | --- | --- |
| | | | <hr/> 3.322 |

| Accounted for. | Quantity,
pounds. | Ounces
per ton. | Metal
content. | Distribu-
tion of
metal,
per cent. |
|-------------------------------------|----------------------|--------------------|-------------------|---|
| Flue | 2,437 | 1.07 | 1.359 | 45.0 |
| Lead | 3,478 | 0.099 | 0.173 | 5.8 |
| Lead dross | 177 | 3.0 | 0.266 | 8.9 |
| Spelter | 17,245 | --- | --- | --- |
| Flue dust | 2,594 | --- | --- | --- |
| Slag | 45,850 | --- | --- | --- |
| In furnace bottom not yet recovered | | | 1.520 | 40.0 |
| | | | <hr/> 3.818 | <hr/> 100 |

| Accounted for. | Quantity,
pounds. | Per cent. | Metal
content,
pounds. | Distribu-
tion of
metal,
per cent. |
|---|----------------------|-----------|------------------------------|---|
| Spelter | 17,245 | 98.0 | 16,900 | 76.3 |
| Flue dust | 2,594 | 68.2 | 1,769 | 8.0 |
| Matte | 3,478 | 9.1 | 316 | 1.4 |
| Lead dross | 177 | 4.0 | 7 | — |
| Slag | 45,850 | 6.9 | 3,156 | 14.2 |
| | | | <hr/> 22,148 | <hr/> 99.8 |
| In furnace bottom not yet recovered | | — | 44 | 0.2 |
| | | | <hr/> 22,192 | <hr/> 100.0 |
| | | | | Per cent. |
| Spelter recovery ^b | | | | 77.9 |
| Zinc recovered in condenser | | | | 8.2 |
| | | | | <hr/> |
| Total zinc recovery | | | | 86.1 |
| Condensation factor ^b | | | | 77.87 |

LEAD.

| Charge chiefly. | Quantity,
pounds. | Per cent. | Metal
content,
pounds. |
|-------------------------|----------------------|-----------|------------------------------|
| Colorado ore..... | 27,847 | 12.75 | 3,550 |
| N. H. ore..... | 27,248 | 6.18 | 1,671 |
| Y. P. slimes (raw)..... | 5,430 | 8.63 | 467 |
| | | | <hr/> 5,694 |

| Accounted for. | Quantity,
pounds. | Ounces
per ton. | Metal
content. | Distribu-
tion of
metal,
per cent. |
|--|----------------------|--------------------|-------------------|---|
| Bullion | ----- | ----- | 2,437.0 | 42.8 |
| Matte | 3,478 | 9.07 | 315.3 | 5.5 |
| Lead dross..... | 177 | 60.0 | 106.2 | 1.9 |
| Slag | 45,850 | 0.153 | 70.4 | 1.2 |
| Spelter | 17,245 | 2.03 | 351.5 | 6.1 |
| Flue dust..... | 2,562 | 14.5 | 371.0 | 6.5 |
| In furnace bottom not yet re-
covered | ----- | ----- | 2,042.6 | 36.0 |
| | | | <hr/> 5,694.0 | <hr/> 100.0 |

COPPER.

| Charge chiefly. | Quantity,
pounds. | Per
cent. | Metal
content,
pounds. |
|--------------------------|----------------------|--------------|------------------------------|
| Colorado ore | 27,847 | 1.69 | 470.0 |
| N. H. ore | 27,248 | 0.64 | 174.4 |
| Y. P. slimes (raw) | 5,430 | 0.57 | 30.3 |
| | | | <hr/> 674.7 |

^b "Spelter recovery" is calculated in the usual commercial way—i. e., on the basis of 100 per cent of zinc in the spelter. The "condensation factor" is similarly calculated.

| Accounted for. | Quantity,
pounds. | Per cent. | Metal
content,
pounds. | Distribu-
tion of
metal,
per cent. |
|--|----------------------|-----------|------------------------------|---|
| Matte ----- | 3, 478 | 12. 62 | 439. 0 | 65. 2 |
| Slag ----- | 45, 850 | 0. 239 | 109. 3 | 16. 2 |
| Spelter ----- | 17, 245 | 0. 05 | 8. 6 | 1. 3 |
| Flue dust ----- | 2, 594 | 0. 07 | 1. 8 | 0. 3 |
| In furnace bottom not yet re-
covered ----- | | | 116. 0 | 17. 0 |
| | | | <hr/> 674. 7 | <hr/> 100. 0 |

SILVER.

| Charge chiefly. | Quantity,
pounds. | Ounces
per ton. | Metal
content,
ounces. |
|----------------------------|----------------------|--------------------|------------------------------|
| Colorado (brown ore) ----- | 13, 288 | 22. 0 | 146. 0 |
| Colorado (black ore) ----- | 12, 664 | 9. 0 | 61. 5 |
| N. H. No. 1 ----- | 16, 427 | 6. 2 | 51. 0 |
| N. H. No. 2 ----- | 4, 054 | 2. 6 | 5. 3 |
| Slimes (reduced) ----- | 1, 831 | 4. 69 | 4. 3 |
| Old mix ----- | 6, 290 | 5. 35 | 16. 8 |
| | | | <hr/> 284. 9 |

| Accounted for. | Quantity,
pounds. | Per cent. | Metal
content,
pounds. | Distribu-
tion of
metal,
per cent. |
|---|----------------------|-----------|------------------------------|---|
| Bullion ----- | 2, 437 | 116. 5 | 142. 9 | 50. 7 |
| Matte ----- | 3, 478 | 21. 8 | 37. 9 | 13. 5 |
| Lead dross ----- | 177 | 87. 0 | 7. 7 | 2. 7 |
| Spelter ----- | 17, 245 | 0. 45 | 3. 9 | 1. 4 |
| Flue dust ----- | 2, 594 | 0. 45 | 0. 6 | 0. 3 |
| Slag ----- | 45, 850 | 0. 78 | 17. 9 | 6. 4 |
| In furnace bottom not yet recovered ----- | | | 74. 0 | 30. 0 |
| | | | <hr/> 284. 9 | <hr/> 100. 0 |

GOLD.

| Charge chiefly. | Quantity,
pounds. | Ounces
per ton. | Metal
content. |
|------------------------------|----------------------|--------------------|-------------------|
| Colorado (brown ore) ----- | 13, 288 | 0. 04 | 0. 268 |
| Colorado (black ore) ----- | 12, 664 | 0. 44 | 3. 008 |
| N. H. No. 1 ----- | 16, 427 | --- | --- |
| N. H. No. 2 ----- | 4, 054 | --- | --- |
| Y. P. Slimes (reduced) ----- | 1, 831 | 0. 05 | 0. 046 |
| Old mix ----- | 6, 290 | --- | --- |
| | | | <hr/> 3. 322 |

| Accounted for. | Quantity,
pounds. | Ounces
per ton. | Metal
content. | Distribu-
tion of
metal,
per cent. |
|---|----------------------|--------------------|-------------------|---|
| Bullion ----- | 2, 437 | 1. 07 | 1. 359 | 45. 0 |
| Matte ----- | 3, 478 | 0. 099 | 0. 173 | 5. 8 |
| Lead dross ----- | 177 | 3. 0 | 0. 266 | 8. 9 |
| Spelter ----- | 17, 245 | --- | --- | --- |
| Flue dust ----- | 2, 594 | --- | --- | --- |
| Slag ----- | 45, 850 | --- | --- | --- |
| In furnace bottom not yet recovered ----- | | | 1. 520 | 40. 3 |
| | | | <hr/> 3. 318 | <hr/> 100. 0 |

LATER DEVELOPMENTS.

After Johnson had convinced himself that the various stages of his procedure were properly correlated he proposed various methods of combining the furnaces used for the different steps into a self-contained unit to carry out the complete process cheaply and efficiently. One of these combinations is shown in Figure 12, in which *A* is a fuel-fired muffle furnace, *B* the electric smelting furnace, and *C* the zinc condenser. The previously roasted ore, mixed with reducer and flux, is fed in at the top of the muffle furnace and rabbled from

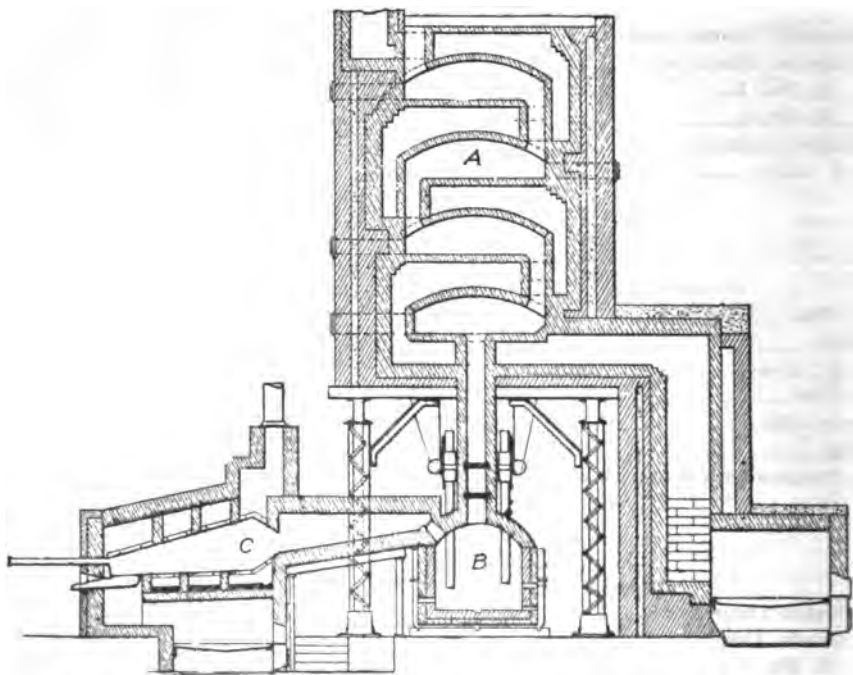


FIGURE 12.—Johnson continuous zinc smelter (U. S. Patent 1244504) : *A*, Fuel-fired muffle furnace; *B*, electric smelting furnace; *C*, zinc condenser.

one hearth to another until the prereduction is complete. It then goes from the lower hearth through a chute directly into the electric furnace below, in which it is smelted and the zinc vapor led into the condenser as shown.

In 1914 Johnson was reported to be planning a 10-ton plant at Keokuk,¹¹ and in 1917 the major rights in his patents were sold to the American Smelting & Refining Co.,¹² who were to try them out at one of their western plants, but no results of either of these trials have been published.

¹¹ Bartlett, R. L., Zinc: Mineral Industry, vol. 23, 1914, p. 798.

¹² Metallurgical and Chemical Engineering, vol. 17, 1917, p. 566.

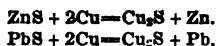
A patent¹³ was granted in 1915 to Johnson on a proposal to combine his process with the usual retort process. According to this proposal, the zinc ore was treated first in fuel-fired retorts for recovery of the easily reduced portion of the zinc, then the residue was treated in an electric furnace for the recovery of the remainder of the zinc and the lead, copper, gold, and silver. In the usual retort furnace, a large share of the zinc is recovered rapidly at a low temperature, without excessive damage to the retort, and without the necessity for a large excess of reduction carbon. The retort furnace is therefore an efficient agent for this stage of the process. The continuous electric furnace, on the other hand, was particularly adapted for the reduction of low-grade ores containing other valuable metals than zinc, such as the product of the preceding step would be. Theoretically this proposal is sensible; whether it would work out in practice would depend upon the cost of the double handling of the material which would be necessitated.

IMBERT-THOMSON-FITZGERALD PROCESS.

THE ORIGINAL IMBERT PROCESS.

The work of Imbert, Thompson, and Fitzgerald originated in methods proposed by A. H. Imbert for carrying out the precipitation process of treating zinc blende.

Imbert's first proposal¹⁴ was to treat zinc-lead sulphide ores with metallic copper in externally heated graphite crucibles or retorts similar to those employed for treating zinc-lead-silver drosses from the desilverization of lead bullion. Zinc blende, which by itself is fusible only at very high temperatures, was claimed to melt readily in the presence of copper, and the zinc and lead sulphides were decomposed by the reactions:



The lead sank to the bottom of the retort and the zinc was distilled off and condensed by appropriate means. The copper was recovered by converting the copper matte according to the usual method and used again.

Later the use of copper was abandoned and metallic iron or a ferro-alloy substituted, but since in the ordinary mixture of ore and iron the reaction proceeds very slowly and only at high temperatures, Imbert proposed to dissolve the zinc sulphide in a molten bath and treat the resulting solution with iron. He mentioned various sub-

¹³ Johnson, W. McA., and Hale, E. W., Duplex smelting process, U. S. Patent 1165371, Dec. 12, 1915.

¹⁴ Imbert, A. H., Process of extracting metals from their sulphides, U. S. Patent 807271, Dec. 12, 1905.

stances¹⁵ which would form suitable fluid baths for dissolving the zinc sulphide, among which were mixtures of an alkaline earth oxide and a metallic oxide, such as CaO and FeO, a mixture of 3 parts of iron sulphide with 1 part of iron oxide (this would dissolve 6 parts of zinc sulphide) and later iron oxide (Fe_2O_3) alone, in the proportion of 1 equivalent of iron oxide to 2 equivalents of zinc sulphide. Lead blast-furnace slag also was suggested as a dissolvent. Oxidized zinc ores, or mixtures of oxidized ore with sulphide ores, could be dissolved in a bath of iron oxide and sulphide and treated with iron in the same manner as zinc sulphide.¹⁶

THOMSON-FITZGERALD RADIANT RESISTOR FURNACE.

Imbert soon found that a fuel-fired retort was not suitable for carrying out his process. At this stage John Thomson and F. A. J. Fitzgerald started to design an electric furnace which could be used for the purpose, and from their experiments there developed a great variety of carbon-resistor furnaces, only a few representative ones of which will be described here.

The first group of these furnaces were of the radiant-resistor type, in which the carbon resistor was placed in the furnace chamber just below the arch, and radiated heat downward to the charge.

The first furnace built¹⁷ had an oblong reaction chamber which was roofed over by the resistor. This was built up of carbon rods laid transversely across the top of the reaction chamber, the ends of the bottom row of rods resting on ledges of magnesite brick. The resistor was in contact with terminals at either end of the furnace. The smelting chamber of this furnace would hold 550 pounds of iron.

A three-day test with his furnace gave a radiation loss of 18 kw. with the temperature of the bath at $1,300^\circ \text{C}$. The full working capacity of the furnace was 40 kw., so that the efficiency was about 55 per cent. At the end of this test, the temperature of the furnace was gradually raised to determine at what point the furnace would finally give way. It was found that failure occurred first by the fusion of the magnesite bricks which supported the resistor rods. The arch did not suffer as might have been expected, as the pressure on the lower carbon rods caused less resistance and consequently larger current and higher temperature in the lower part of the re-

¹⁵ Imbert, A. H., Process of treating zinc and lead sulphide ores, U. S. Patent 875578, Dec. 31, 1907; Metallurgical treatment of sulphurous ores by the precipitation process, U. S. Patent 875579, June 6, 1907; Process of reducing zinc ores, U. S. Patent 927857, July 13, 1909.

¹⁶ Imbert, A. H., Metallurgical process of treating oxidized zinc ores by the precipitation process, U. S. Patent 875580, Dec. 31, 1907.

¹⁷ Fitzgerald, Francis A., A new electric-resistance furnace: Trans. Am. Electrochem. Soc., vol. 19, 1911, pp. 273-284.

sistor and a more moderate temperature in the upper part near the arch.

Later another furnace of similar type of 150 kw. capacity was built. In order to obtain the necessary resistance without making the furnace too long, the furnace was built with both terminals at one end, the resistor being in two sections connected with each other at the opposite end of the furnace. One side of each section of the resistor was supported on a ledge running along one side of the furnace, while the other sides were supported on longitudinal arches spanning the furnace chamber.

The inside dimensions of the reaction chamber were 55 inches square by 10 inches deep. The resistor was built of seven layers of rods three-fourths inch in diameter by 20 inches long. Each section of the resistor was about 45 inches long. This furnace gave an efficiency of about 78 per cent at 1,250° C. and 72 per cent at 1,450° C.

It was found that the temperature reached in the resistor was too near the working limit of the supporting refractories, so several forms of self-supporting resistor were designed. Some of these were simple arches built up of carbon wedges; others were arches built up of interlocking corrugated carbon plates. A furnace could be arranged to be heated partly by fuel and partly by electricity by building a false arch just under the resistor, thus inclosing it in a separate chamber in which a reducing atmosphere could be maintained to prevent oxidation of the graphite.

The resistance in these resistors is largely in the contacts between the plates or rods and is much greater than that of the graphite alone.

Another interesting adaptation of the above design was for a revolving furnace with a central resistor rod which might be built up of interlocking plates¹⁸ or might take the form of a carborundum tube filled with granular carbon.¹⁹

The Imbert process, using Thomson-Fitzgerald furnaces, was experimented with for several years in Hohenlohehütte, Upper Silesia, and in Australia, but was finally abandoned.²⁰

OTHER THOMSON-FITZGERALD FURNACES.

A later group of furnaces²¹ designed by Thomson for the reduction of zinc ores by carbon were heated by a resistor of granular

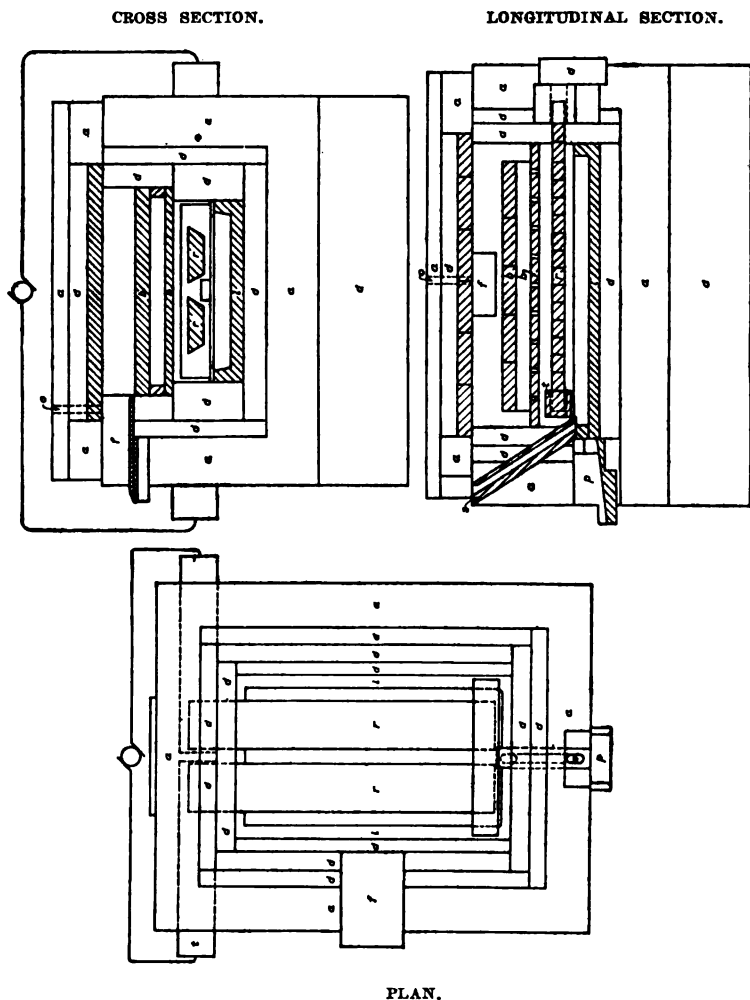
¹⁸ Thomson, John, Electric furnace, U. S. Patent 950880, Mar. 1, 1910.

¹⁹ Thomson, John, and Fitzgerald, F. A. J., Furnace, U. S. Patent 950878, Mar. 1, 1910.

²⁰ Ingalls, W. R., Zinc: Mineral Industry, vol. 21, 1912, p. 895.

²¹ Thomson, John, Electric zinc furnace with integral condenser, U. S. Patents 1080862, 1080863, 1080865, 1080866, Dec. 9, 1913; 1086414, 1086415, 1086417, Feb. 10, 1914; 1090427, 1090428, Mar. 17, 1914. Electric zinc furnace with integral compound resistor and compound condensers, U. S. Patent 1086416, Feb. 10, 1914. Universal electric zinc furnace with integral condenser, U. S. Patent 1086418, Feb. 10, 1914. Universal electric furnace combined with means for condensing zinc, U. S. Patent 1090429, Mar. 17, 1914.

carbon inclosed in a grating of refractory rods or coarse pieces of carbon. The resistor was surrounded by and embedded in the ore charge and the zinc vapors passed through the granular carbon of the resistor into a condenser which was built as an integral part of the furnace.



PLAN.

FIGURE 13.—Thomson-Fitzgerald radiant resistor furnace for the redistillation of spelter and scrap zinc—plan, cross section, and longitudinal section (from Fitzgerald, F. A. J., Radiant resistor furnace, Trans. Am. Electrochem. Soc., vol. 36, 1919, pp. 350-352): *a*, Wall of heat-insulating brick; *b*, diaphragms; *d*, inner walls of fire brick; *f*, opening to condenser; *l*, hearth; *o*, hole for pyrometer tube; *p*, tapping spout; *r*, zigzag resistors; *s*, inclined tube for charging molten zinc.

During the war the price of high-grade zinc increased so much that there was incentive for experiments on the redistillation of low-grade spelter or scrap zinc, and Thomson and Fitzgerald adapted their radiant-resistor furnace to this use. The design of a furnace

modified for this purpose is shown in Figure 13 (plan, vertical section, and transverse section). The furnace was about $6\frac{1}{2}$ feet long. The inner walls were of fire brick *d*, outside of which was a wall of heat-insulating brick *a*. The premelted zinc was introduced into the hearth *l* through the inclined tube *s*, the lower end of which extended below the surface of the molten bath. The furnace was heated by zigzag resistors *r*, which were made from graphite slabs with slots cut in them to increase the path of the current. In the later forms of this resistor the slots are filled with alundum cement, thus materially increasing its strength. Above the resistor were two diaphragms *b*. The lower consisted of transverse plates separated by a slight space, and the upper was solid but did not extend to the end of the furnace. The zinc vapor passed between the plates of the lower diaphragm, around the end of the upper one, and thence out of the opening *f* to the condenser. The roof of the furnace was of carbon slabs covered with fire brick and insulating brick. The hole *o* admitted a pyrometer tube. The condenser was a flat box 100 by 50 by 7 cm., with a series of transverse baffle plates. It was heated with a charcoal fire at the start, but as soon as distillation was well started no external heating was necessary, and a good condensation to liquid zinc was obtained.

Experimental work on the redistillation of zinc was carried on for several months with this furnace, with considerable success. The results of a 37-hour run were as follows:²²

Results of 37-hour run with Thomson-Fitzgerald radiant-resistor furnace.

| | | |
|---|--------------------|-------|
| Average power used during distillation..... | kilowatt..... | 55.2 |
| Total energy for distillation..... | kilowatt hour..... | 2,051 |
| Minimum temperature of zinc entering condenser..... | °C..... | 1,040 |
| Average temperature of zinc entering condenser..... | do..... | 1,225 |
| Average temperature of zinc in premelter..... | do..... | 550 |
| Weight of zinc condensed (3,494 pounds)..... | kilograms..... | 1,558 |
| Weight of zinc condensed per hour (95 pounds)..... | do..... | 43 |
| Energy per kilogram of zinc..... | kilowatt hour..... | 1.29 |
| Energy per pound of zinc..... | do..... | .59 |

The total energy consumption per pound of zinc, including pre-melting, during the last 9 hours of the run, was 0.62 kw. h.

Recently Thomson has patented a furnace²³ for the smelting of zinc ore in which the resistor, made of a carbon plate cut into zigzag shape, extends through the center of the furnace, heating partly by conduction and partly by radiation, the ore charge lying in piles on either side of it.

²² Fitzgerald, F. A. J., Radiant resistor furnace: Trans. Am. Electrochem. Soc., vol. 36, 1919, pp. 349-355.

²³ Thomson, John, Electric furnace for reducing oxidized zinc concentrates by carbon, U. S. Patent 1321683, Nov. 11, 1919.

THIERRY FURNACES.

A series of patents²⁴ have been issued to C. V. Thierry alone, and to C. V. and M. Thierry, on a group of furnaces similar to the granular carbon-resistor furnaces of John Thomson. Thierry's object was the reduction of a concentrated zinc oxide product with carbon, so no provision was necessary for taking care of a solid or liquid residue, except to provide for an occasional cleaning out of the furnace.

One of these furnaces is shown in cross section in Figure 14. In this furnace, *a* is the reaction chamber on the bottom of which lies

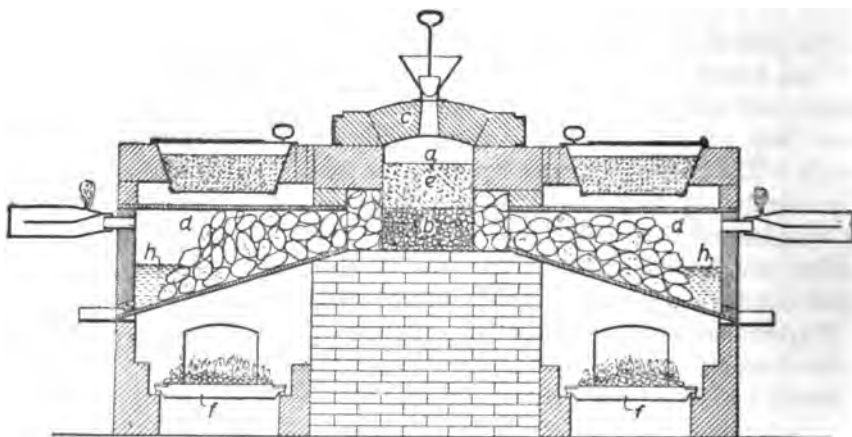


FIGURE 14.—Thierry furnace (U. S. Patent 1030350, June 25, 1912): *a*, Reaction chamber; *b*, carbon resistor; *c*, cover; *d*, condensers; *e*, charge; *f*, grates; *h*, molten zinc.

the granular carbon resistor *b*, making contact with the electrodes at opposite ends of the furnace. The chamber is closed by the cover *c*. The resistor preferably having been previously heated to smelting temperature, the charge *e* of zinc oxide and reducer is introduced through the hoppers onto the surface of the bed of carbon. The zinc vapor produced by the reduction of the zinc oxide passes through the carbon of the resistor and into the condensers *d*, which also contain granular carbon. The condensers are arranged to be preheated by means of the grates *f*. This furnace is claimed to have given good yields of metallic zinc.²⁵

THE TAYLOR SHAFT FURNACE.

Edward R. Taylor has designed a series of shaft furnaces for smelting ores. One of these intended for the smelting of zinc ores

²⁴ Thierry, C. V., Metallurgy of zinc, U. S. Patents 1030349, 1030350, June 25, 1912. Electric zinc furnace with integral condenser, U. S. Patents 1110359, Sept. 15, 1914; 1122063, 1122064, Dec. 29, 1914.

²⁵ Ingalls, W. B., Zinc: Mineral Industry, vol. 21, 1912, pp. 894, 895.

was described by him in a paper read before the American Electrochemical Society in 1907.²⁶

The following description is quoted from this paper:

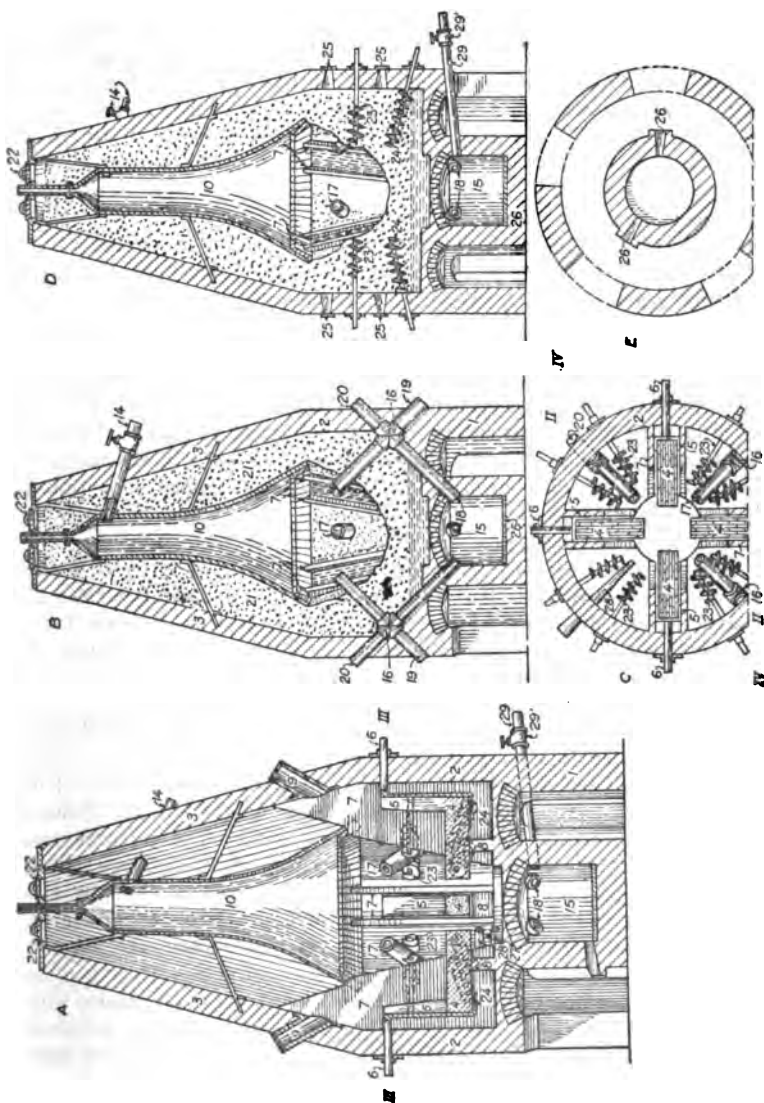


FIGURE 15.—The Taylor shaft furnace (Taylor, Edward R., A closed electric furnace for reducing and distilling metals from their ores, Trans. Am. Electrochem. Soc., vol. 11, 1907, p. 296) : 1, Base; 2, reducing chamber; 3, shaft; 4, horizontal electrodes; 5, 6, metallic conductors; 7, partitions inclosing electrode; 8, piers; 9, charging aperture; 10, hood; 14, pipe; 15, chamber; 16, conduits; 17, 18, 19, 20, legs; 21, insulating material; 22, apertures; 23, 24, screws; 25, 26, 27, 28, tap holes; 29, openings.

The furnace structure (fig. 15) comprises a base 1, reducing chamber 2, and shaft 3, the latter for the introduction of the charge and the removal of certain of the gases formed in the operation.

* Taylor, Edward R., A closed electric furnace for reducing and distilling metals from their ores: Trans. Am. Electrochem. Soc., vol 11, 1907, pp. 295-301.

The base may be formed of a series of masonry arches supporting the superstructure and inclosing a collecting chamber 15. Within the furnace and above the base are horizontal electrodes 4 of carbon supported by masonry piers 8 and provided with metallic conductors 5 and 6. On either side of each electrode are vertical retaining walls, and near the upper portion is a charging aperture 9 provided with a suitable closure. In the upper portion, and supported in part by these walls, is a hood, or bell 10, flaring outward and downward, and serving for the collection and withdrawal of some of the gaseous products of the reaction. This hood may be constructed of iron or steel for moderate temperature work, or suitable refractory material for operations demanding higher temperatures, or exposed metal portions may be protected with refractory material. 14 represents a pipe for the removal of the lighter gases, such as carbon monoxide (CO).

Within the base of the furnace is the chamber 15, referred to before, which serves for the collection of the volatile metal portions resulting from the reaction. This chamber connects with the furnace proper by means of the conduits 16, which may be made of fire-clay tubes or graphitic carbon, or may be built within the masonry of the furnace. These conduits are provided with legs 17 extending into the interior of the furnace at a point somewhat above the electrodes (or so they will not be covered with the charge passing through the furnace) and with legs 18 extending into the collecting chamber. Legs 19 and 20, in prolongation of legs 17 and 18, pass through the walls of the furnace, and are provided with suitable closures. These extensions afford opportunity for inspection and cleaning of the conduits. In some cases it may be desirable to withdraw some of the furnace products through the legs 19.

Adjacent to these conduits, and preferably arranged on two or more levels of the furnace, are a plurality of screws or equivalent mechanical devices 23 and 24, constructed and arranged to continuously, or from time to time, force portions of the furnace charge into the reaction zone. As best shown in Figure 15, *C*, a plurality of these screws are arranged radially around the periphery of the furnace, and in operation serve to force the material into the field of reaction between the electrodes. As shown in Figure 15, *D*, the same screws are mounted in two or more superposed rows, the screws 23 in the upper row being bladed at their inner ends only, while the screws 24 of the lower row are bladed from their inner ends nearly or quite to the furnace walls, and inclined upwardly toward the reaction zone. The purpose and effect of this construction is to insure that portions of the charge which descend along the periphery of the furnace, past the screws 23, shall be conveyed by the screws 24 to the field of reaction through the surrounding charge.

Ores adapted to being worked in a furnace of this construction are fed into the furnace at the top apertures 22 provided for that purpose. They may be previously mixed with carbon or other reducing material, or, if necessary, with suitable fluxes. Broken carbons may be fed upon the electrodes as required through openings 9. Several years in the use of broken carbons in the reinforcement of the main carbon electrode has amply demonstrated their great usefulness, both for regulating the current and prolonging the life of the main carbon electrode. For an expenditure of \$175 for the main carbons we have been enabled to produce 2,000,000 pounds of bisulphide of carbon in the electric furnace. In much electric furnace work the cost of carbons is a large item in the cost of the finished product, which is here shown to be almost infinitesimal, certainly not one of the larger of the fixed charges of running a plant.

As the ore is moved forward into the reaction zone it is replaced by fresh portions from the periphery of the furnace. Any slag or nonvolatile reduction products may be drawn off from time to time through the tap holes 27 and 28.

Provision is made for the lighter gases to pass through the hopper 10 and the pipe 14, while the heavier volatile metals may pass more readily through 16 to the receiving chamber 15, entering by the pipes 18, and such gases as are not condensed under such circumstances out through the pipe 29. By controlling the openings 14 and 29, the direction and movement of these gases and vapors may be directed and controlled.

The metal collecting in 15 may be tapped off through the tap holes 26.

COTE AND PIERRON PROCESS.

EARLY EXPERIMENTS.

E. F. Cote and P. R. Pierron at first directed their attention largely to the direct treatment of raw zinc sulphide ores with metallic iron in an electric furnace. In this way they hoped to do away with some of the difficulties of condensing zinc vapor in the presence of the large amounts of carbon monoxide and considerable proportions of CO₂ formed when the ore is first roasted and then reduced with carbon. They found, however, that due to the moisture and carbonates in the ore and the air introduced into the furnace with the charge, considerable quantities of harmful gases were still produced. They therefore introduced a column of incandescent carbon between the furnace and the condenser to reduce the carbon dioxide and water vapor to carbon monoxide and hydrogen.

Their various patents are mainly on different designs for combining the electric furnace and the column of carbon in practical form. In some of these the carbon was heated electrically and in others it was so managed that the heat of the gases going into it would keep it at the necessary temperature.

FLEURVILLE'S DESCRIPTION OF THE PROCESS IN 1908.

In 1908, Eugene Fleurville published a comprehensive account of the development of the Cote and Pierron process and a description of the furnace then in use.²⁷ Figure 16 is a diagram of the furnace. The bottom and side walls of the crucible *a* were made of graphite, incased in sheet metal, and were connected to one pole of the electric circuit. There was a projection *b* in the center of the furnace bottom, directly under the electrode *c*, so that the current flowed between this projection and the electrode. The arch *d* was of refractory brick. The holes *e* served for the introduction of the charge and could be closed by refractory stoppers. The zinc vapor passed through the outlet *g* and passage *h* to the condenser *j*, which was a vertical chamber filled with pieces of carbon *i*. The condenser was tapped through the hole *o*. Air could be admitted at *k* and drawn into the stack *m*, allowing the carbon in the upper part of the cham-

²⁷ Fleurville, Eugene, *Electric zinc smelting*: *Electrochem. and Met. Ind.*, vol. 7, November, 1909, pp. 468-472.

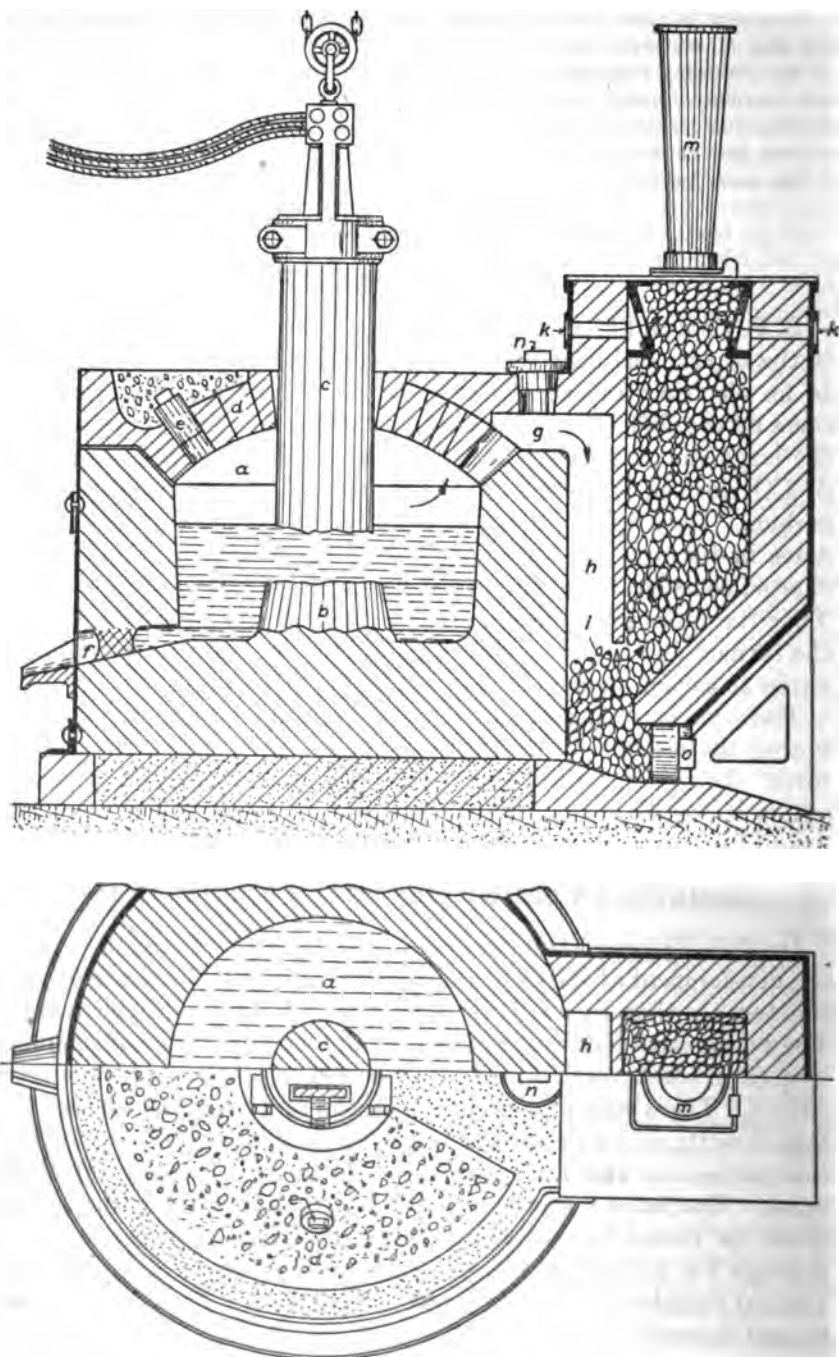


FIGURE 16.—Cote and Pierron furnace (French Patent 385018, 3d addition, Nov. 9, 1908) : *a*, Crucible; *b*, projection from bottom electrode; *c*, electrode; *d*, arch; *e*, charge holes; *f*, spout; *g*, outlet; *h*, passage for zinc vapors; *i*, carbon; *j*, condenser; *k*, opening for admission of air; *m*, stack; *n*, opening for cleaning channels; *o*, hole for drawing zinc.

ber to be heated to a red heat. The downward passage of the carbon was regulated so that the heat of the gases coming into the condenser, with the heat liberated by the condensation of the zinc, would keep the condenser at the proper temperature for the reduction of "zinc mist" and condensation of the zinc. The opening *n* was provided for cleaning the channels *g* and *h* in case of clogging. The condenser could have been heated electrically if necessary.

The charge was first dried on the roof of the furnace. The previous charge having been tapped out, the upper electrode was lowered into contact with the projection *b*. The charge was then introduced through the holes *e*, the holes again closed, and a new charge placed on top of the furnace. The electrodes being in contact with each other, practically no current passed through the charge, which was heated only by contact with the electrodes. The lead was precipitated at this stage and tapped out. The door *o* at the bottom of the condenser was left open during this time and the pieces of carbon allowed to slide down enough to clean the condenser. At the same time, the carbon was heated to the proper temperature by the gases produced in the furnace. After the lead had all been precipitated and removed from the furnace the tap hole and condenser door were closed and the electrode *c* gradually raised so that the charge was heated by resistance and arc heating. The zinc liberated and the iron sulphide formed settled into the annular ring around the lower electrode projection. Finally, by simple arc heating, the last of the zinc was set free and the slag was melted and tapped out, leaving the furnace ready for the next charge.

The iron sulphide was to be sold to sulphuric acid manufacturers, partly compensating for the cost of the metallic iron used.

The hot zinc vapor going into the condenser maintained the lower part of its column of carbon at a sufficient temperature to reduce any zinc oxide in the vapor. As the vapor passed up into the cooler regions, it was condensed to liquid zinc and trickled to the bottom where it was tapped off. Any dust or foreign material tending to clog the condenser was removed by raking out some of the carbon from time to time and fresh carbon was introduced at the top to replace it. The small amount of carbon monoxide passed out through the stack *m*. Incomplete condensation of gas would be shown by escaping fumes of zinc oxide; then the temperature of the condenser could be lowered by increasing the speed of the passage of the carbon down the condenser or by cutting down the power to the furnace crucible.

The first experiments with this process on a semicommercial scale were made in 1906 at Lyons, France. The furnace consisted of a cylindrical crucible of 40 cm. inside diameter and 45 cm. high, built

up of magnesia brick held together by a metal casing. The electrodes were placed vertically along the central axis of the crucible and charge hopper. The furnace worked somewhat on the same principles as the one above described. The experiments with this furnace were intended chiefly to work out the chemical problems of the precipitation reactions, power consumption, metal losses, and electrode consumption. No particular effort was made to obtain liquid zinc.

The Société des Fonderies Electriques was formed in 1907 to continue the experiments on a larger scale. The plan was first to study the suitability of various types of furnaces for the process and then to determine the cost of operation and commercial possibilities of a 500-horsepower plant. It was intended to treat low-grade zinc ores for the recovery of zinc white, leaving the problem of condensation of spelter till later. After successful completion of this step, a 1,200 to 1,500 horsepower plant was to be built for the making of zinc white. As experience was acquired in the handling of large volumes of zinc vapor, the production of liquid spelter was to be increased until practically all of the zinc was recovered as metal.

At the time Fleurville's article was written, the first step of this program had been successfully completed. Furnaces of many types were tried, the one finally adopted being of the type described above. The author notes that the manufacture of zinc oxide involves its own problems which may or may not be less difficult than the problems of zinc condensation.

Following are the results of a typical furnace run :

Total time of series of experiments, 240 hours.

Weight of ore smelted during that time, 14,560 kilograms.

Average zinc content, 43.6 per cent.

Weight of iron mixed with ore, 6,235 kilograms.

Weight of fluxes added to the charge, 3,480 kilograms.

Mean current, 4,300 amperes; voltage, 40; power factor, 0.80.

Weight of zinc oxide obtained, 6,730 kilograms.

Metallic zinc condensed before passage to the combustion apparatus, 182 kilograms.

Zinc content in the slag and iron sulphide, 2.7 per cent.

Consumption of electrodes, 193 kilograms.

Electric energy cost \$13 per horsepower year.

Electrodes cost \$10.40 per 100 kilograms.

Iron cost, \$12 per ton; lime, \$3 per 1,000 kilograms.

The workmen were paid an average of 60 cents per day.

Under these conditions the cost of 4,912 kilograms of oxide and metallic zinc was \$479, not including general expenses but including maintenance of the furnaces. Adding the cost of packing and loading into cars brought the total cost to \$526. The oxide was sold for \$640.

For the production of commercial zinc white, the use of iron was later given up because some of the iron volatilized, discoloring the zinc oxide. Instead of iron, lime and carbon were used. These react with the blende according to the reaction $\text{ZnS} + \text{CaO} + \text{C} = \text{CO} + \text{CaS} + \text{Zn}$. The results for a furnace run using this method of reduction are given and are similar to those obtained when iron was used. The zinc oxide produced analyzed 98.6 to 99 per cent ZnO , 0.4 per cent moisture, 0.2 to 0.3 per cent iron, and 0.5 to 0.7 per cent silica, lime, and other impurities. The ore was free of other metals which would contaminate the oxide.

TEST REPORTED IN 1914.

In 1914 results were reported of a test in which the zinc was obtained as spelter.²⁸ In 72 hours the furnace smelted 9,560 kg. of blende, assaying 34.8 per cent zinc, mixed with 2,970 kg. of iron. The electric energy used was 16,285 kw. h. There were produced 2,800 kg. of spelter, assaying 99.06 per cent zinc. The iron sulphide and slag weighed 9,730 kg. and ran 1.85 per cent zinc. Electrode consumption was 110 kg. The daily capacity of the furnace was placed at 3,200 kg. of ore at 1,700 kw. h. per metric ton (1,574 kw. h. per ton of 2,000 pounds). Two men were required to operate the furnace. The cost per metric ton of 34 per cent to 38 per cent ore was \$9.52, or \$8.63 per ton of 2,000 pounds.

COMPOUND SMELTING FURNACE AND CONDENSER.

In 1917 G. Flusin reported²⁹ that a plant was under construction at Maurienne, France, which was to have four 500-horsepower furnaces with a daily capacity of 4 metric tons of ore per furnace.

The process as used at that time consisted of the treatment of raw blende, or mixed sulphides with lime and carbon in a "compound furnace." This was really a combination of two furnaces. The charge was smelted in the first furnace, of the arc-resistance type. The second furnace, of the indirect-resistance type, received the zinc in the form of droplets and zinc dust directly from the first furnace, and with the addition of a small amount of heat energy, revolatilized it as pure zinc vapor which was condensed to liquid spelter. The power consumption was estimated at 2,000 to 2,200 kw. h. per metric ton of 35 per cent zinc ore. Seventy to 75 per cent of the power was used in smelting and 25 to 30 per cent in redistillation. The slag assayed 1.5 per cent zinc and the spelter 99.93 per cent. The total

²⁸ Engineering and Mining Journal, The Cote-Pierron electric zinc-smelting process: Vol. 97, Apr. 11, 1914, p. 756.

²⁹ Flusin, G. [Recent development of the Cote and Pierron electric zinc-smelting process]: Bull. Tech. Suisse Romande, vol. 23, No. 17, 1917, p. 233; abstract, Met. and Chem. Eng., vol. 18, Jan. 1, 1918, p. 17.

loss of zinc was less than 10 per cent. Electrode consumption was 12 kg. per metric ton.

The compound furnace mentioned by Flusin is evidently of the type described in Cote and Pierron's United States Patent 1,184,520 of May 23, 1916. Figures 17 and 18 illustrate this furnace. The smelting furnace, Figure 17, is similar to the earlier one described by Fleurville. The vapors from this pass into the chamber *h* of the

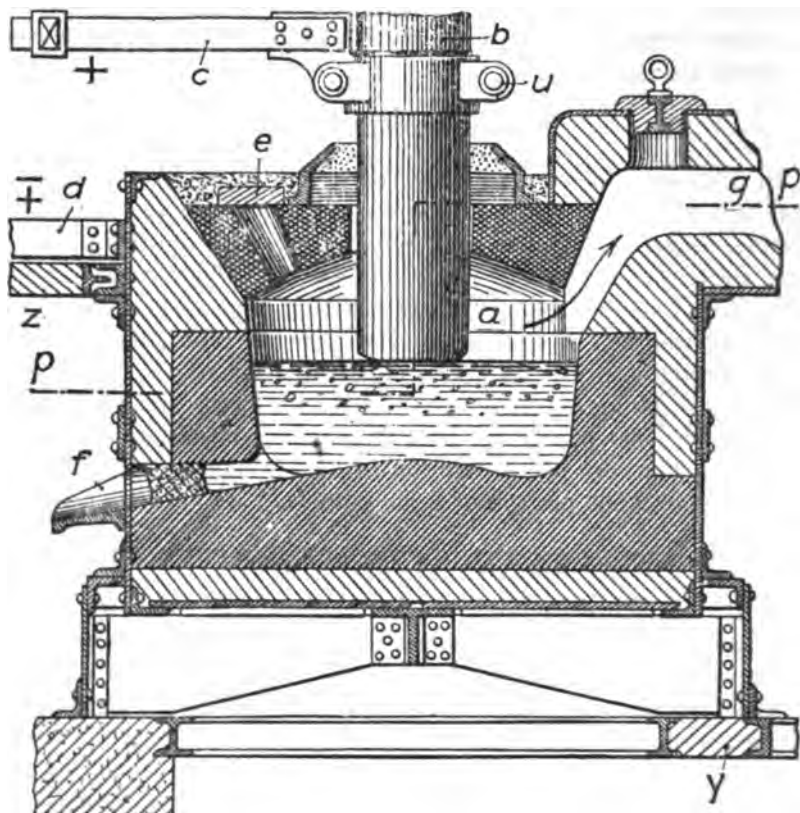


FIGURE 17.—Cote and Pierron smelting furnace (U. S. Patent 1184520, May 23, 1916) : *a*, Chamber; *b*, electrode; *c*, *d*, busbars; *e*, charge hole; *f*, spout; *g*, passage for zinc vapor.

refining furnace, Figure 18. Here most of the zinc condenses as dust and droplets and falls upon the bed of carbon below. Any zinc vapor remaining is condensed in the carbon column *j*. The column of carbon below the condensing chamber is caused to move downward, carrying the zinc powder with it, by removing carbon through doors at the bottom and replacing it with the carbon raked out of the column *j*, which in turn is replaced by a fresh supply. As the zinc powder is carried below the narrowed region *k* of the column, it is heated by an electric current between the graphite ring *l* and the electrode *s* and revolatilized. The pure zinc vapor thus produced

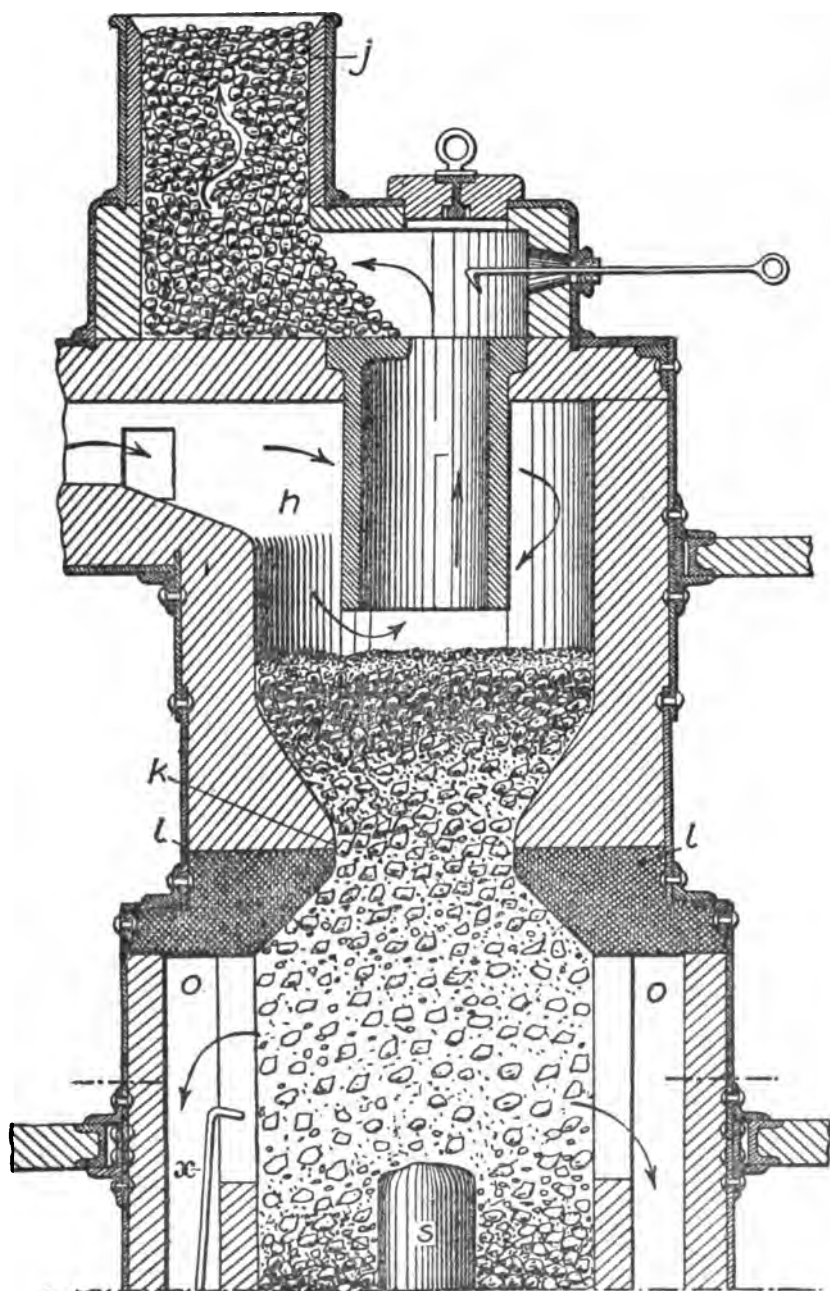


FIGURE 18.—Cote and Pierron refining furnace (U. S. Patent 1184520, May 23, 1916):
h, Condensing chamber; *j*, carbon column; *k*, narrow region of column; *l*, graphite
 ring; *o*, annular ring serving as condenser for refined zinc vapor; *s*, electrode.

passes through the slits and into the annular ring *o*. Here it is condensed to liquid metal and is collected at the bottom of the condenser compartments. Any liquid zinc which passes down the carbon column without being revolatilized is collected at the bottom of the column.

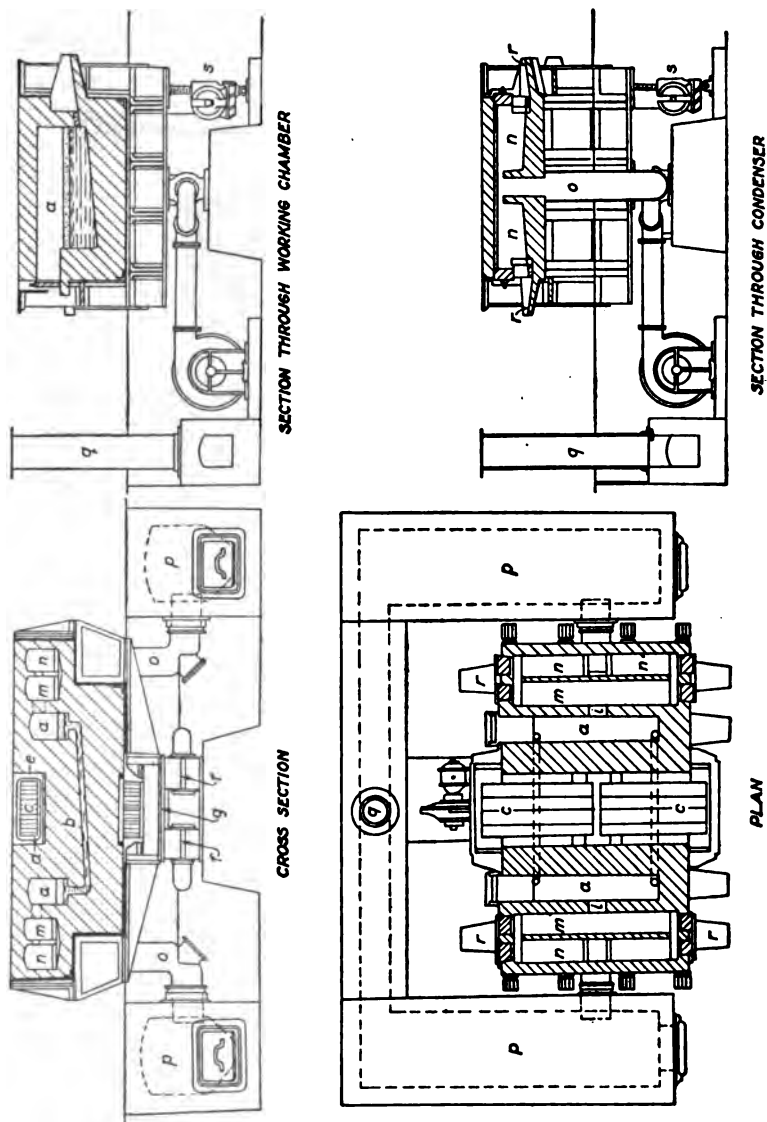


FIGURE 19.—Gin induction furnace (British Patent 28688, 1906) : *a*, Working chambers; *b* conduits; *c*, double magnetic circuit; *d*, insulated copper coils; *e*, water-cooling channels; *f*, trunnions; *t*, passages to condensing chambers; *m*, *n*, condensing chambers; *o*, conduit for gases from condenser; *p*, dust chamber; *q*, stack; *r*, spout; *s*, jackscrew.

GIN INDUCTION FURNACE.

One of the early furnaces suggested for the electrothermic production of zinc was the induction furnace of Gustave Gin shown in Figure 19. The two parallel channels *a* formed the working cham-

bers of the furnace, and with the conduits *b* of circular cross-section made up the secondary circuit of a transformer. The two insulated copper coils *d*, connected in series, wound on the upper horizontal branches of the double magnetic circuit *c* formed the primary circuit of the transformer. The magnetic core and primary winding were protected from the hot furnace walls by air cooling and by the water-cooled channels *e*.

The whole furnace was contained in an iron shell resting upon two iron trunnions *f*. The lining was of burnt dolomite tamped in with a pitch binder.

A bath of iron or other metal in the working channels and connecting conduits served to carry the secondary current and produced the heat for smelting the charge which was spread upon its surface. The iron could take part in the reduction of the zinc, if desired, or if roasted ore and carbon were charged the iron would simply serve as resistor. The zinc vapor and gases produced passed through the channel *i* into the condensing chambers *m* and *n*, and thence out through the conduit *o* to the dust chamber *p* and stack *q*. The furnace could be tipped by the motor-driven jackscrew *s* to facilitate the tapping operation, pouring at *r*.

WORK OF F. T. SNYDER AND THE CANADA ZINC CO.

DESCRIPTION OF SOME OF THE SNYDER FURNACES.

From 1906 to 1909, patents were granted to Frederick T. Snyder on several furnaces and processes for smelting zinc ores electrically.³⁰

These included an induction furnace, two vertical shaft furnaces, and several forms of slag-resistance furnaces. Snyder appreciated the beneficial effect of prereduction of the charge and of a concentrated zinc vapor in obtaining condensation to liquid zinc, and took these into consideration in his design.

Two of these furnaces, one of vertical-shaft type and one of the slag-resistance type, are illustrated in Figures 20 and 21.

The shaft furnace shown in Figure 20 had a fire-brick base, *a*, inclosed in an iron shell, *a'*, held in place by lugs, *p*, surmounted by the water-jacketed rectangular furnace shaft *b*, which was left open at the top to receive the charge and permit the escape of uncondensed vapors. The crucible was to be kept full of molten metal, matte, and slag. A longitudinal row of three electrodes, *c*, projected vertically downward into the furnace far enough for the ends to dip into the

³⁰ Snyder, F. T., Metallurgical process, U. S. Patents 814810, Mar. 13, 1906; 859185, July 2, 1907. Electric furnace, U. S. Patents 825359, July 10, 1906; 859136, 859137, July 2, 1907; 942110, Dec. 7, 1909. Process of treating ores, U. S. Patent 834644, Oct. 30, 1906. Smelting process, U. S. Patents 859132, 859134, July 2, 1907. Smelting furnace, U. S. Patent 859133, July 2, 1907. Process of treating zinc ores, U. S. Patent 933133, Sept. 7, 1909.

molten conducting material in the crucible. These electrodes were star connected to a three-phase, alternating current, the neutral point being connected to the metal in the crucible of the furnace.

The hopper *d* was arranged to feed powdered coke immediately around the electrodes, the main body of the charge being fed between the outer edges of this hopper and the side walls of the furnace. The consumption of electrodes was thus considerably reduced.

In operating the furnace the top of the charge was maintained at a temperature low enough to condense zinc vapor and prevent

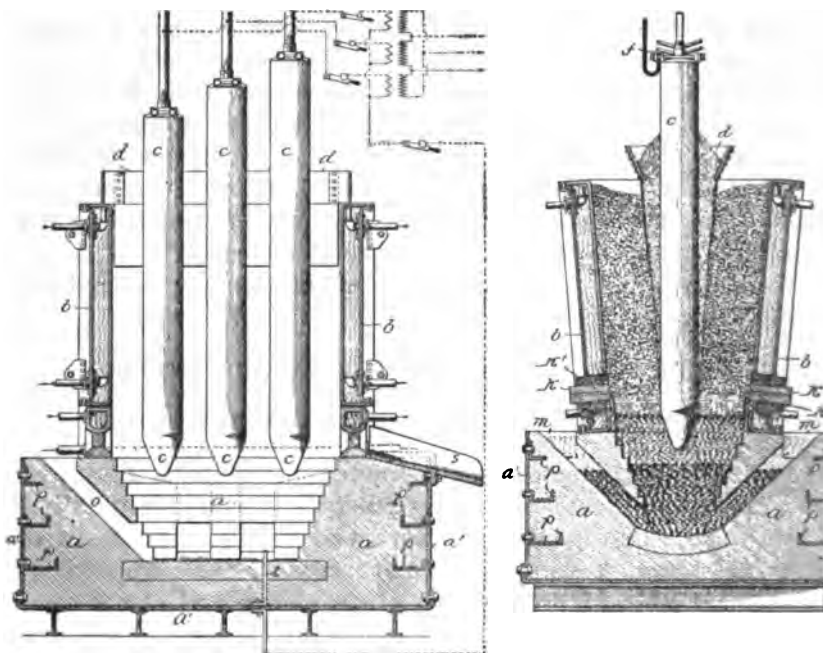


FIGURE 20.—Snyder shaft furnace (U. S. Patent 942110, Dec. 7, 1909): *a*, Fire-brick base; *a'*, iron shell; *b*, furnace shaft; *c*, electrodes; *d*, hopper; *k*, drains for liquid zinc; *k'*, water jackets; *m*, zinc wells; *o*, lead well; *p*, lugs; *s*, slag tap.

escape. As the charge traveled down the shaft it was gradually heated and the easily reducible oxides were reduced to metal. The gases from this preliminary reduction, which would otherwise dilute the zinc vapor formed later, were thus eliminated before the charge reached the temperature for the reduction of zinc oxide. When the charge reached the smelting zone the zinc oxide was reduced, the resulting CO gas escaped from the top of the furnace while the zinc vapor condensed in the cooler portions of the charge and was returned with it to the smelting zone. In this way the charge was progressively enriched until a large part of the zinc condensed as liquid metal against the water jackets. It then flowed down the sides of the furnace and escaped through the drains *k* provided for this purpose.

purpose. The shelf, or crust, of slag formed around the sides of the furnace just below the drains by the cooling effect of the water jackets *k'* assisted in diverting the liquid zinc into the drains.

The remainder of the charge, after the zinc was distilled off, melted to slag, matte, and bullion and settled into the crucible. The slag and matte were drawn off through the slag tap *s* at the top of the crucible while the lead was ladled out of the lead well *o*, which communicated with the bottom of the crucible.

The liquid zinc that drained from the tubes *k* was collected in well *m* in the base of the furnace, which could be connected to the lower part of the crucible. These wells served to permit the refin-

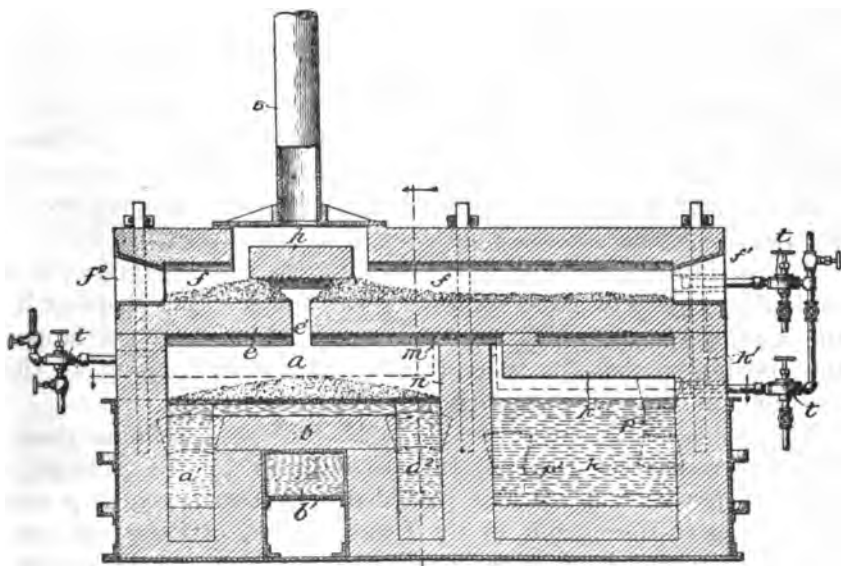


FIGURE 21.—Snyder slag-resistance furnace (U. S. Patent 859137, July 2, 1907): *a*, Furnace chamber; *a'*, *a''*, chambers filled with molten lead; *b*, bridge wall; *b'*, water jacket; *e*, roof of smelting chamber; *e'*, passage between preheating chamber and furnace; *f*, preheating chamber; *h*, cover; *k*, condenser; *k'*, flues; *k''*, roof; *m*, opening to condenser; *n*, partition; *p*, *p'*, passages for lead and refined spelter; *s*, stack; *t*, oil burners.

ing of the zinc, as the lead contained could settle out and join the main body of the same metal which was contained in the crucible.

In the resistance furnace illustrated in Figure 21, *a* represents the furnace chamber proper and *e* its roof. This was divided by the bridge wall *b*, cooled by a circulation of water in the water jacket *b'*. The two chambers *a'* and *a''* formed by this bridge wall were filled with molten lead and each chamber communicated with the outside of the furnace by two lead wells, on opposite sides of the furnace. The electrodes were dipped into the lead wells on one side of the furnace while the lead wells on the other side permitted the metal

to be ladled out as fast as it was produced, keeping the metal inside the chambers at the desired level. The top of the bridge wall was covered with a layer of slag connecting the two chambers containing lead and providing a passage between them for the electric current. The charge was so proportioned that this slag should have a high electrical resistance, a low dissolving power for zinc oxide, and a formation temperature between the temperatures of volatilization of zinc and lead (between $1,000^{\circ}$ and $1,100^{\circ}$ C.). A high-lime slag of about the composition 30 per cent CaO , 30 per cent FeO , and 40 per cent SiO_2 was mentioned as being suitable.

The heat required for smelting was produced by the passage of the current through this layer of slag.

On top of the slag rested a layer of incandescent coke and on this the charge proper. Since the formation temperature of the slag was regulated to slightly over $1,000^{\circ}$ C. it was assumed that as long as a sufficient amount of charge was present any excess of heat supplied after this temperature was reached would be used up in reducing zinc oxide and forming slag, so that the temperature of the furnace would not rise high enough to volatilize much lead.

The preheating chamber *f* for roasting the ore and giving it a preliminary reduction was connected with the smelting chamber by the opening *e'* over which was the cover *h*. This cover overlapped the opening so that the latter could be sealed by heaping up the materials in the preheating chamber around it.

The condenser *k*, large enough to be used as a refining chamber, was connected with the furnace chamber by the passage *m* over the partition *n*. It was provided with two passages, *p* and *p*¹, leading to the outside of the furnace. One of these was connected with the bottom of the condenser and permitted the removal of any lead which separated from the spelter, while the other, opening into the condenser near the top, provided for the removal of the refined spelter. The gases remaining after the condensation of the zinc vapor were led through the flues *k'* to the preheating chamber where the CO was burned to furnish heat for the preliminary reduction. The gases from the preheating chamber passed out through the stack *s*. Oil burners *t* were provided for heating up the cold furnace and melting the slag when starting after having been shut down for some time. These burners were shut off and the openings plugged with clay while the regular smelting operation was being carried on.

In operating the furnace, the crushed and, if necessary, partly roasted ore, was fed into the preheating chamber and the roasting completed. The proper proportions of reduction material and fluxes

were then added, and after the reduction had been carried as far as possible without volatilizing the zinc, the charge was introduced into the smelting chamber through the opening *e'* and the opening sealed. The zinc was there reduced and volatilized and entered the condenser with the CO gas produced. The lead, gold, and silver settled to the bottom of the electrode chamber as bullion and were drawn off

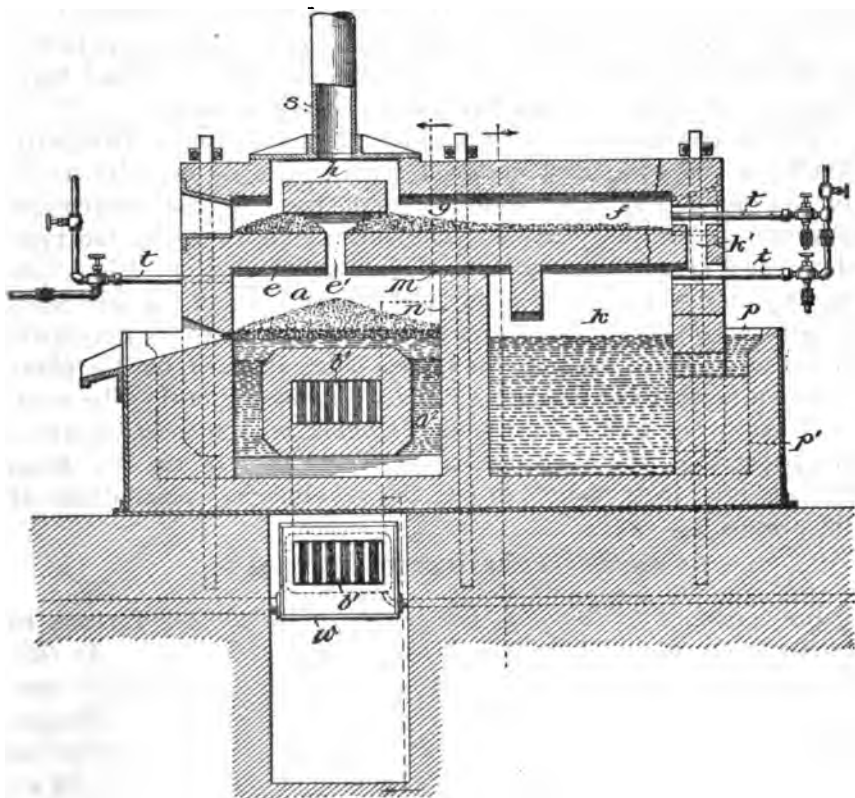


FIGURE 22.—Snyder induction furnace (U. S. Patent 859137, July 2, 1907): *a*, Furnace chamber; *a'*, chambers filled with molten lead; *b'*, transformer core; *e*, roof of smelting chamber; *e'*, opening between preheating chamber and furnace; *f*, preheating chamber; *h*, cover; *k*, condenser; *k'*, flues; *m*, passage to condenser; *n*, partition; *p*, zinc well; *s*, stack; *t*, oil burners; *w*, primary winding.

through the lead wells, and the other constituents of the ore formed slag which was tapped out as necessary, always leaving enough to carry the heating current.

The zinc and any lead volatilized were condensed and separated in the condenser, while the CO gas passed on into the upper chamber and was burned to preheat the succeeding charge.

Figure 22 shows the same furnace arranged as an induction furnace. The transformer core is represented by *b'*, and *w* is the pri-

mary winding. The secondary circuit is formed by the molten metal and slag contained in the smelting chamber, this material extending completely around the core.

EXPERIMENTS AT VANCOUVER AND NELSON, BRITISH COLUMBIA.

Snyder began experimental work with small furnaces at Vancouver, British Columbia, in 1905. Several types of furnace, 50 to 150 kw. size, were tried during 1905 and 1906, leading up to the shaft style of furnace, somewhat similar in shape to a lead blast furnace, which was later used at Nelson, British Columbia.

The Canada Zinc Co. erected a 10-ton plant at Nelson in 1908, with buildings and equipment designed for an ultimate capacity of 30 tons of ore per day, financial aid being obtained from the provincial government for the purpose. The furnace used was of the type shown in Figure 20 (see p. 42). This plant started operations late in 1908 on ores averaging about 40 per cent zinc, 10 per cent lead, $1\frac{1}{2}$ per cent copper and 12 ounces silver per ton,²¹ and very optimistic reports were given out as to the success being attained, but the plant was shut down about a year later and work discontinued. The work at Nelson was a series of mechanical troubles with furnace construction, operation, and electrode breakage, and none of the runs seem to have been long enough to test the metallurgical possibilities of the process.

LATER EXPERIMENTS AT CHICAGO.

Just previous to the war, Snyder designed another furnace to overcome the faults of those used at Nelson (see Fig. 23). In this furnace a bath of lead, 6, forming the bottom electrode, makes contact with the bus bar 9 through the passage 7 and block 8. The tap hole is shown at 10. Above the furnace body 1 is a frame 12, in which is mounted the roof 13. The upper electrode 14 is held by an arm which can be raised and lowered. The roof is provided with a plurality of openings around its periphery for introduction of the charge, and is arranged so that a considerable supply of ore can be stored upon it. Below each of the charging ports 17 is a reciprocating feeder, 18, to push the charge into the smelting chamber as it gradually works down through the opening.

The zinc, volatilized in the smelting chamber, condensed in the incoming charge and trickled down into the troughs 21, from which it was conducted to a collecting trough, 22, and withdrawn through the outlet 23. When such fine ore was being smelted as to prevent the condensed zinc from percolating through it, the ore and coke

²¹ Electrochemical and Metallurgical Industry, *Electric zinc in Canada*: Vol. 7, 1909, p. 189.

were charged separately as shown, the zinc finding an outlet through the coarser coke.

A 1,500-kw. furnace of this type was built in Chicago and ran for a short time on zinc ores. Plate I, A, is a photograph of this fur-

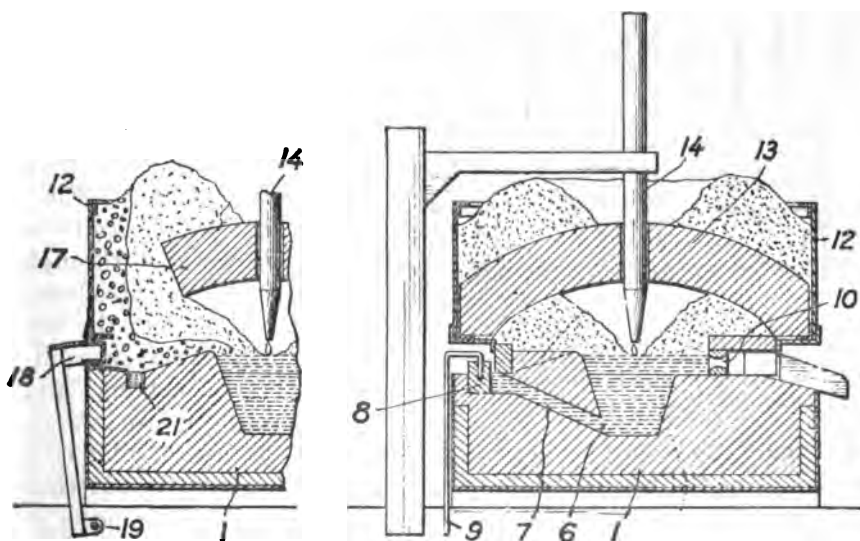
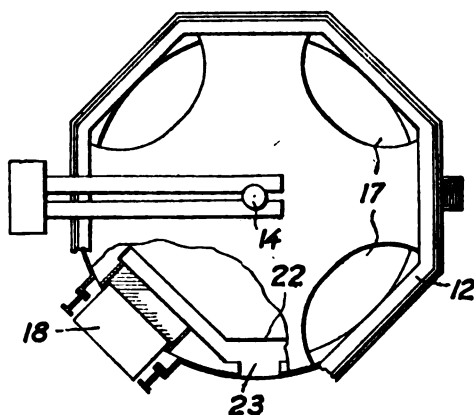


FIGURE 23.—Snyder's Chicago furnace (U. S. Patent 1254079, 1918): 1, Furnace body; 6, bath of lead; 7, passage; 8, block; 9, bus bar; 10, tap hole; 12, frame; 13, roof; 14, upper electrode; 17, charging ports; 18, reciprocating feeder; 21, troughs; 22, collecting trough; 23, outlet.

nace. Mr. Snyder states that these tests were successful, but that after a short run the furnace was converted to other uses and the zinc work dropped. He states the power consumption at about 900 kw. h. per ton of ore, the furnace capacity being about 35 tons a day. The electrode consumption was about 10 pounds per ton of ore, and

furnace labor six-man shifts per 35 tons. No exact data as to metal recoveries are available, but the slags produced were low in metal values.

LOUVRIER FURNACES.

François Louvrier has suggested several types of electric shaft furnace adapted to the smelting of zinc ores. One of his early furnaces is shown in Figure 24. The charge was fed continuously down the shafts *a* into the smelting region around the electrodes *b*, slag being removed at *c*. The zinc vapor was led into the condenser *d* between the two smelting chambers and condensed.

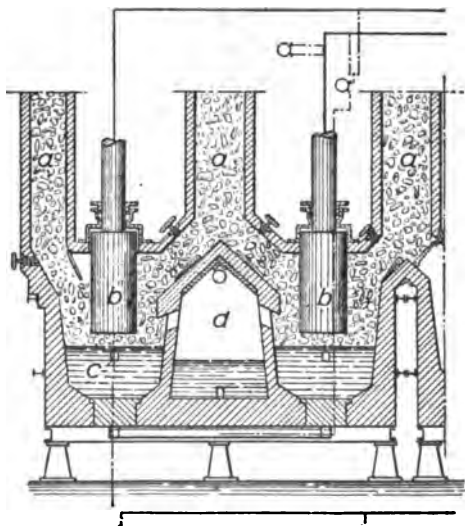


FIGURE 24.—Early Louvrier furnace (U. S. Patent 989169, Apr. 11, 1911): *a*, Shafts; *b*, electrodes; *c*, opening for removal of slag; *d*, condenser.

Later Louvrier concluded that the large formation of blue powder in electric smelting was due to the intense localized heat employed, and altered his furnace to a single boshed column, with a plurality of electrodes entering the walls of the furnace bosh.

A recent form of his furnace is shown in Figure 25. In this the smelting chamber 2 is of larger diameter than the charge column 3. Electrodes, 12, which can be raised and lowered through stuffing boxes, are introduced at intervals through the top of this smelting chamber outside the feed column, and other electrodes are shown

at 8 and 9, those at 9 being always in contact with the annular ring of metal in the bottom of the crucible. The heating current thus traverses the charge in the crucible in all directions and heats it uniformly.

The upper column of the furnace is double walled, the two walls forming the annular chamber 15. This is provided with a series of shelves 16. Powdered coal is fed through this chamber from the supply bin 17 into the smelting chamber, some of it being always retained on the shelves provided. The zinc vapors produced in smelting are led off over this powdered coal and out through the flue 19 to the condenser. Powdered coal can also be fed around the vertical electrodes, thus protecting them somewhat from wear and oxidation.

THE QUENEAU FURNACE.

During the several years beginning about 1908, A. L. J. Queneau worked with various forms of a rotary electrical furnace, one modification of which is presented in Figure 26. The external steel shell *e* was provided with rings *d*, which rested upon the flanged rollers *q*, and with a band gear *h*, driven by a variable-speed motor so that the whole furnace could be rotated or oscillated. The shell was closed at one end by the metal plate *a'* having the hub projection *s* through which the charge was introduced. This end plate was so fastened to the shell as to be electrically insulated from it and made contact with one lead of the external electric circuit by means of the brush *r*.

The opposite end of the shell was closed with the plate *a*, with a recess, *g*, for cooling with air or water spray. This plate was electrically connected by the flexible connection *b* to the cylindrical shell and thence by the brush *m* to the other lead of the electric circuit. The shell was lined first with a layer of fire brick and inside of that with a layer of magnesite or chrome brick resistant to the slag produced in the furnace.

The end courses *n* were made of blocks of a conducting mixture of graphite, dead burned magnesite, and a carbonaceous binder, baked to drive off hydrocarbons. These were backed up by the graphite end linings *o*.

The conductor that carried the current inside the furnace was a layer of molten iron alloyed with about 12 per cent of phosphorus. This alloy has a higher resistance than pure iron and remains fairly constant in composition. If desired, the slag formed in the furnace could be used alone as resistor, in which case, because of its higher resistance, a higher voltage and smaller current could be used.

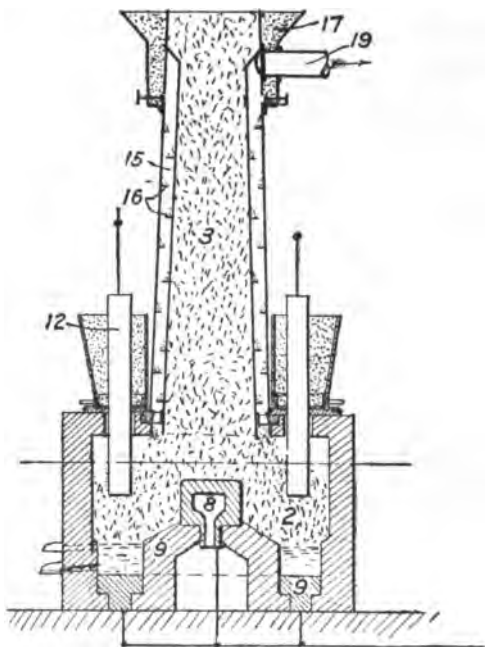


FIGURE 25.—Louvrier furnace of 1920 (U. S. Patent 1342636, June 8, 1920): 2, Smelting chamber; 3, charge column; 8, 9, 12, electrodes; 15, annular chamber; 16, series of shelves; 17, supply bin for powdered coal; 19, flue.

The current thus passed from the lead *r* to the plate *a'*, thence through the graphite *o* and conducting courses *n* to the fluid conductor *t*; thence through the courses *n*, and graphite *o*, at the opposite end to the plate *a* and shell *e* to the opposite lead *m*.

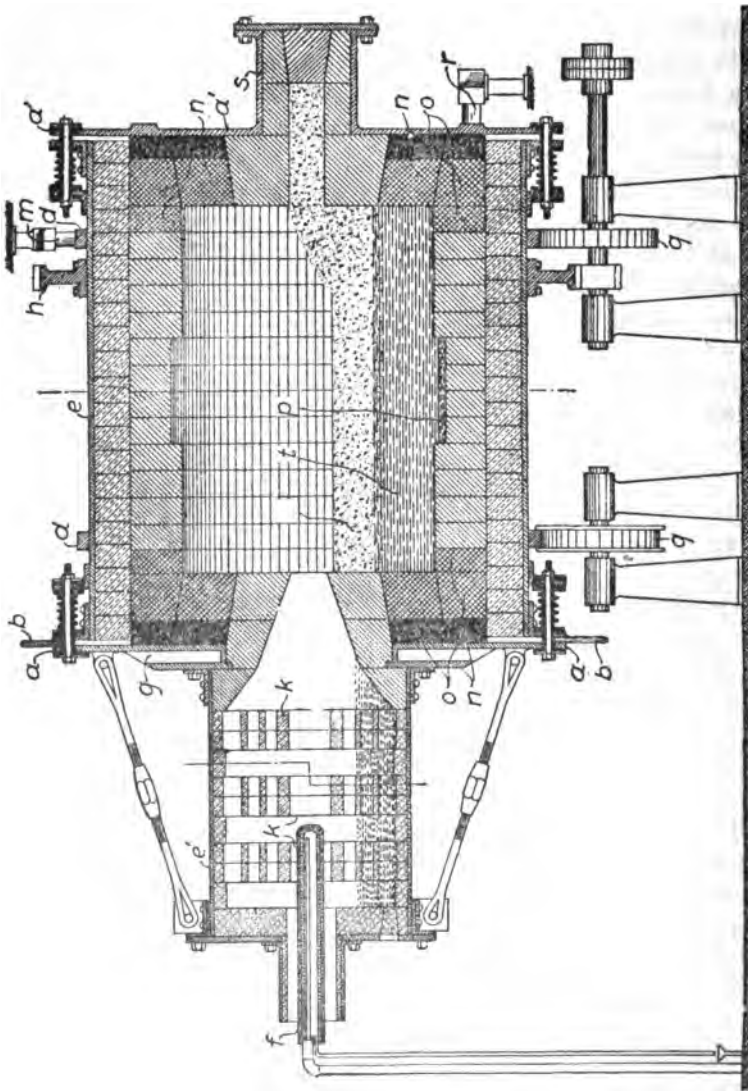


FIGURE 26.—The Queneau rotary furnace (U. S. Patent 1006877, Oct. 24, 1911): *a, a'*, End plate; *b*, flexible connection; *d, d'*, rings; *e*, steel shell; *e'*, condenser shell; *f*, air or water cooling device; *g*, cooling recess; *h*, band gear; *k*, partitions; *m*, brush; *n*, conducting end courses; *o*, graphite and linings; *p*, well for heavy metals; *q*, flanged rollers; *r*, brush; *s*, hub projection; *t*, fluid conductor.

At *p* the diameter of the furnace chamber was increased somewhat to provide a well for any heavy metal contained in the charge.

The condenser shell *e'* was bolted to the end plate of the furnace shell. It was lined with refractory brick and was provided with the perforated partitions *k* to provide condenser surface, and with the air or water cooling device *f*.

Queneau also designed a stationary furnace of the buried-arc or slag-resistance type, which was charged at the side, on the same principle as a copper reverberatory smelting furnace. In this furnace the condenser consisted of a member of short tubes embedded in the end walls of the furnace.

ELECTROTHERMIC ZINC SMELTING EXPERIMENTS OF THE CANADIAN GOVERNMENT.

PURPOSE OF THE INVESTIGATION.

After the closing down of the plant of the Canada Zinc Co. at Nelson, British Columbia, the zinc producers of East and West Kootenay, British Columbia, appealed to the Canadian Government to conduct an investigation for the purpose of developing some process by which the zinc ores of that region could be profitably treated. This question of the development of the zinc industry of British Columbia had been a much agitated question for several years, as the distance to a market for zinc ores made the profitable mining of such ores in the Province difficult.

In response to this appeal, \$50,000 was voted in 1910 by the Dominion Government for the purpose asked. W. R. Ingalls was placed in charge of the investigation and was instructed "to investigate and carry through an investigation for the discovery or development of some method for the economical treatment of the mixed zinc-sulphide ores of Canada, in the production of metallic zinc or a marketable zinc product."

EXPERIMENTS AT MCGILL UNIVERSITY.³²

After a careful survey of the field it was decided that electric smelting offered the most promise, and laboratory experiments were inaugurated at McGill University, first under the immediate direction of Dr. Alfred Stansfield and later of Edward Dedolph. It was realized from the beginning that if electric smelting were to be successful with power costs as in Canada, a large yield of spelter must be obtained in the first smelting with only a small amount of blue powder to be resmelted. To prove that zinc vapor produced in an electric furnace can be successfully condensed to liquid metal under favorable conditions the first experiments consisted of simply volatilizing spelter by electrical heat and recondensing it. Then a series of tests were made with several types of furnace reducing fairly pure zinc oxide. Results from these tests proving favorable, work on roasted zinc concentrates was started.

A prolonged series of small-scale experiments was then carried out during the years 1911 and 1912. Many designs of furnace were

³² Most of the data on experiments at McGill and Nelson are taken from W. R. Ingalls' unpublished report to the Canadian Department of Mines.

used, with various methods of charging, arrangements of electrodes, and condensation apparatus. The detrimental effect of CO_2 was realized and for some time an incandescent coke column between the furnace and condenser was considered necessary. This was, however, subject to clogging and the temperature difficult to regulate, and was later found to be unnecessary if the ore charge were preheated before being charged to the smelting furnace.

The detrimental effect of dust carried into the condenser from the furnace was for a long time not realized in its full importance, but after steps were taken to diminish dusting the condensation results were much improved.

The type of furnace finally decided upon in the small-scale experiments was of the slag-resistance type. The charge was fed through an open shaft and rested upon the slag bath. A brick wall separated the charge shaft from the vaporizing part of the furnace, affording conditions that minimized the dusting of the charge. A furnace of this type, having a capacity of 200 to 250 pounds per 24 hours, gave in a series of runs very encouraging results and it was decided to build a larger furnace at Nelson, British Columbia.

EXPERIMENTS AT NELSON, BRITISH COLUMBIA.

During 1913 a furnace, with a preheater and other accessories, designed for a capacity of 2,000 pounds per 24 hours, was erected at Nelson, in the old plant of the Canada Zinc Co. In this plant the charge of roasted ore and reducer was to be preheated in retorts, measuring 7 by 11 by 62 inches inside dimensions, for 4 hours. Eight of these retorts were sufficient to preheat 2,240 pounds per 24 hours. The preheated charge was pulled from these retorts into heavy pots and transferred to the smelting furnace.

The furnace was of the final type used in the small experiments, as above described, but was so planned that details of construction, such as the charge-feeding device and electrode arrangement could be easily altered if further experiments proved such alteration necessary. The condenser consisted of five common wrought-iron pipes, 5 inches in diameter, heated by a grate fire.

In the first test of this furnace the masonry walls were quickly destroyed, and it was decided that no kind of masonry would be likely to endure the severe conditions of electric zinc smelting. Accordingly a furnace with water-jacketed walls was built. Thenceforth the work at Nelson consisted of efforts to work out a satisfactory furnace construction and method of operation. These efforts were rewarded by a gradual improvement in results, but the original appropriation was used up before the metallurgical difficulties were entirely overcome.

Ingalls, as a result of the work at Nelson, advised that a 1-ton furnace could never be a commercial success, and that it was im-

possible to estimate the cost of developing a larger furnace and overcoming the difficulties that he expected to encounter in going from a 1-ton furnace to one of perhaps 10 tons capacity per day. Furthermore, W. McA. Johnson, who had been working along the same lines, was as far or farther advanced toward a successful process, and was planning to build a 10-ton plant; therefore, Ingalls advised that the work be dropped pending Johnson's further experiments.

PETERSON'S EXPERIMENTS.

During 1912 and 1913, some interesting work was done by Peter E. Peterson at Butte, Mont., on behalf of the Butte & Superior Co.,

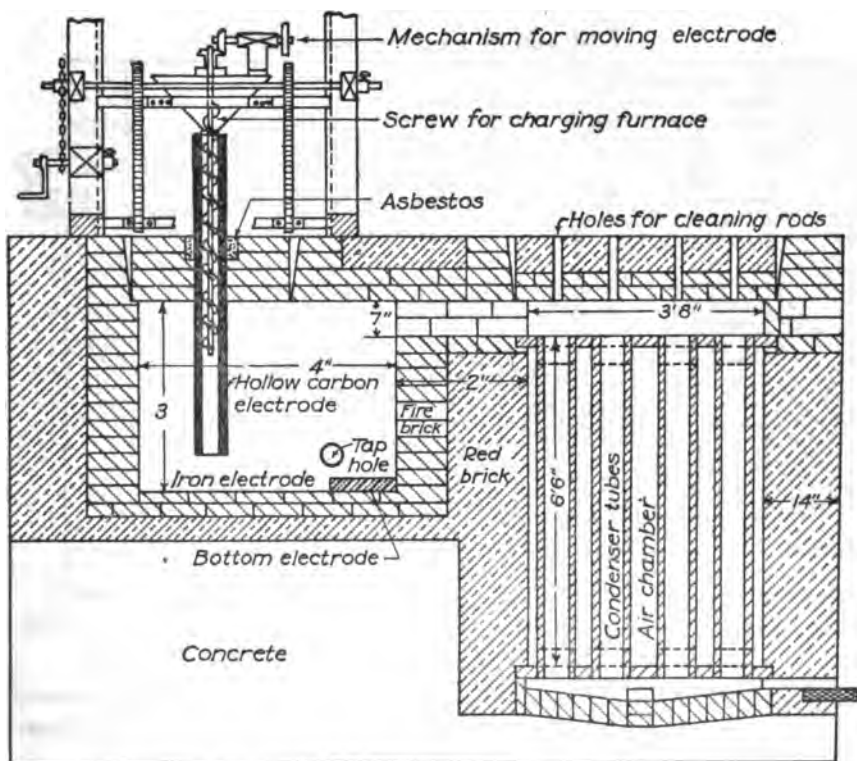


FIGURE 27.—Peterson furnace with hollow electrode (Trans. Am. Electrochem. Soc., vol. 24, 1913, p. 220).

using the buried-arc or slag-resistance type of furnace similar to the Trollhattan and Johnson furnaces, but with novelties in the way of charge feeding devices and condensers.³³

In the first experiments, the precipitation method of treating raw blende with iron or lime and carbon was used. Later this was abandoned in favor of the usual reduction of roasted ore with carbon.

³³ Peterson, P. E., The electric furnace for zinc smelting: Trans. Am. Electrochem. Soc., vol. 24, 1913, pp. 215-239.

One of the earlier furnaces used in the larger-scale experiments is shown in Fig. 27. This furnace was designed for a capacity of 1 ton per 24 hours. The bottom electrode was an iron plate or graphite rod, and the vertical one was a hollow carbon or graphite tube. The charge was introduced into the furnace through this hollow electrode by a screw feeding device. This furnace gave a low power consumption—1,100 to 1,300 kw. h. per ton of 50 per cent zinc concentrate—but was mechanically unsatisfactory.

The condenser consisted of eight clay tubes surrounded by cold-air passages. Another condenser used with this type of furnace

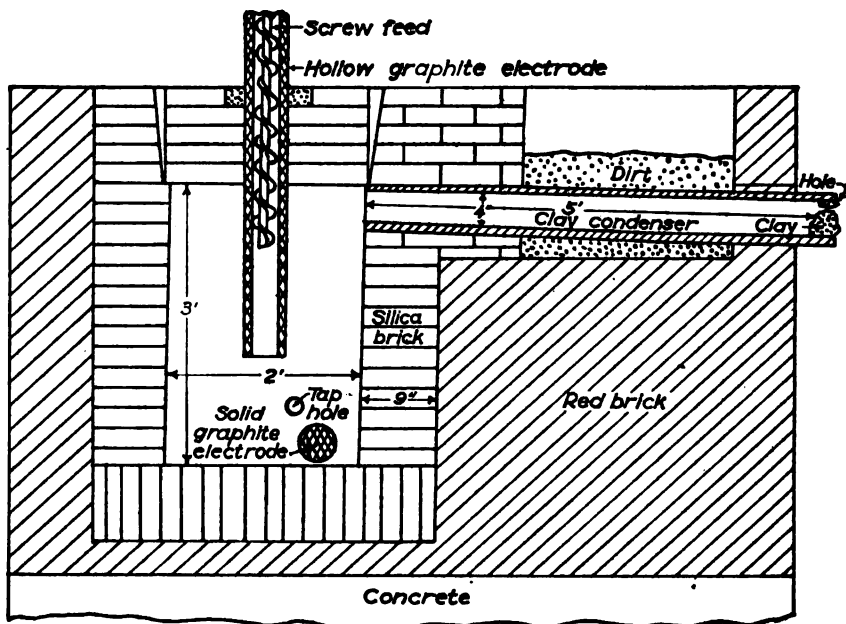


FIGURE 28.—Hollow electrode furnace with multiple horizontal-tube condenser (Trans. Am. Electrochem. Soc., vol. 24, 1913, p. 225).

consisted of several horizontal clay or graphite tubes, arranged as in Figure 28, which could be covered with a variable depth of heat-insulating material.

The first condenser could not be kept hot enough for good condensation; it consequently produced mostly blue powder and was decided to be too complicated for practical use. The horizontal tubes in the second condenser gave some spelter but did not have capacity enough to condense all the zinc vapor produced. A later run was made with this latter condenser, in which the furnace was run on barren material until the condenser reached 859°C . The zinc charge was then introduced and fed at such a slow rate that very little zinc escaped from the end of the condenser tubes. Dur-

ing 12 hours of smelting in this way, 86.4 per cent of the zinc charged to the furnace was recovered as liquid zinc, but the power consumption was 4,000 kw. h. per ton of 50 per cent zinc concentrate. The principle of this condenser was evidently good, but its capacity was not proportioned properly to that of the furnace.

It was decided that the horizontal bottom electrode was unsatisfactory, so it was given up and a furnace using two nearly vertical electrodes was tried. One of these furnaces is shown in Figure 29.

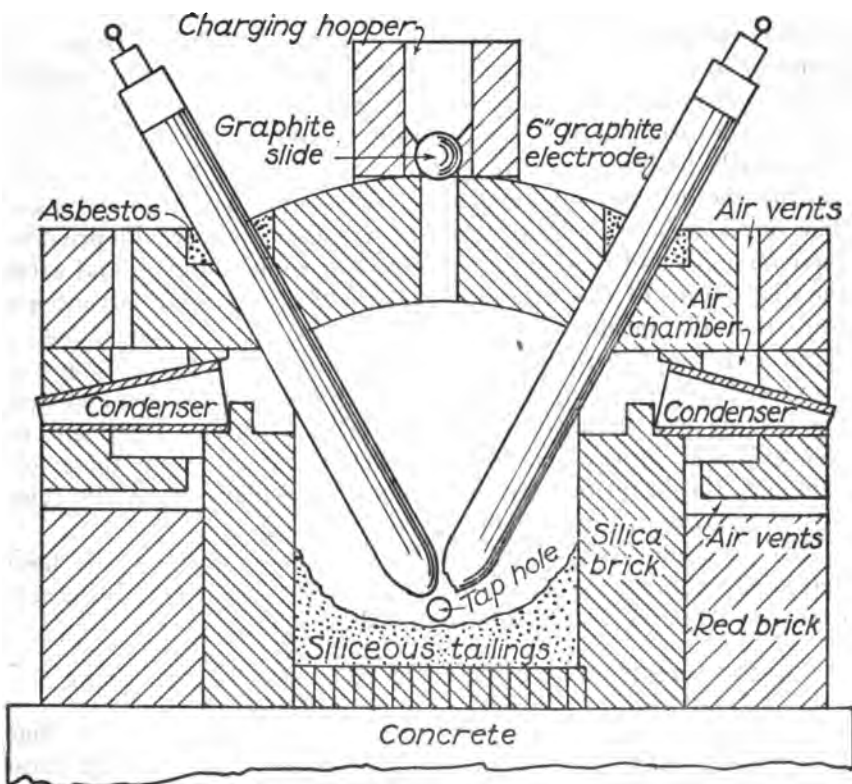


FIGURE 29.—Peterson buried-arc furnace (Trans. Am. Electrochem. Soc., vol. 24, 1913, p. 225).

The smelting operation in this furnace gave no trouble during an actual running time of 30 days, but the same condensation difficulties arose.

In mixing the charge, the plan was to produce slags having such a high formation temperature that the zinc would be volatilized before slagging of the charge began. It was found possible to reduce the zinc content of the slag to any desired figure. Coke was used as a reducer. Sufficient sulphur was left in the roasted concentrates and enough copper ore was added to the charge to pro-

duce some matte to serve as a collector for the precious metals. The company decided that the work was not of commercial promise and abandoned it in 1913.

FULTON ELECTRO-RESISTANCE BRIQUET FURNACE.

THE INDESTRUCTIBLE BRIQUET.

In 1914, Charles H. Fulton proposed,²⁴ as an improvement in the retort distillation of zinc ores, to mix the roasted ore with an excess of fine coke and sufficient hot coal tar, pitch or similar carbonaceous material to act as a binder, and to form this mixture into briquets under a pressure of 500 to 1,000 pounds per square inch, finally baking the briquets at a temperature of 450° to 500° C. to drive off volatile hydrocarbons from the pitch, leaving a coke network uniting the original coke and ore particles into a continuous mass. The novel feature of these briquets was that they would preserve their strength and their original form and volume during and after the distillation of the zinc from them. The particular advantages were that the intimate mixture of the ore with carbon caused a rapid and complete reduction of the zinc, and that the residue was prevented from forming a slag to corrode the retort walls, and that the residue was in ideal condition for blast-furnace treatment to recover lead, copper, and precious metals. Aside from these advantages, the briquets were much more convenient to handle than a loose mixture of ore and coke or coal.

The particular proportions of ore, coke, and pitch to be used depended upon the character of the ore, a typical mixture being 100 parts ore, 60 parts coke, and 18 to 20 parts pitch.

APPLICATION OF THE BRIQUET TO ELECTRIC SMELTING.

The strength and stability of these briquets, with the fact that they would conduct electricity, suggested the feasibility of an electric furnace in which the baked briquets themselves would serve as the resistor.

Experimental work with such a furnace was begun in Cleveland, Ohio, in 1914, and in 1916 a plant was erected in East St. Louis, Ill., for the purpose of working out the technical details of the process on a commercial scale. Experiments were continued at this plant, with very promising results, until January, 1918, when the unsettled labor and material markets, due to the war, forced the suspension of the work.

²⁴ Fulton, C. H., Recovery of zinc, U. S. Patent 1193680, Aug. 8, 1916.

EXPERIMENTS AT EAST ST. LOUIS.

The work at East St. Louis was very completely discussed by Fulton in a paper read before the American Institute of Mining and Metallurgical Engineers in 1919.⁵⁵ Most of the following description is taken from this paper.

BRIQUETS.

The briquets used in most of the work at East St. Louis were cylindrical, 9.25 inches in diameter and 21 inches long, weighing about 90 pounds, and containing 50 pounds of ore. They were formed by mixing the coke and ore, then heating and mixing with the melted pitch in a pug mill and pressing. The mold had a temperature of 75° to 90° C. for the best results and the pressure used varied from 500 to 1,000 pounds per square inch.

The size of the ore used in the briquets, within reasonable limits, has little effect either on the strength of the briquet or on the extraction of the zinc; ore which passes a 200-mesh screen gives practically the same result as that which passes a 10-mesh screen and 86 per cent of which remains on a 35-mesh screen. The size of the coke has a strong influence upon the strength of the briquet, very fine coke giving a much stronger briquet than comparatively coarse coke. With very fine coke and ore, however, it is necessary to use more of the pitch binder, thus increasing the cost of the briquet somewhat.

The coke used must be enough to reduce all the reducible oxides of the ore and leave an amount sufficient to give the briquet the requisite stability and electrical conductivity. Ordinarily the briquet after the distillation of the zinc should have at least 40 to 50 per cent of the weight of the original briquet, depending somewhat upon the nature of the residue. When the ore gives an infusible granular residue the briquet has more strength after distillation than when this residue tends to fuse into globules. An ore that leaves a residue of reduced metal has a higher electrical conductivity than the average, whereas an ore with a lime gangue may form calcium carbide, thus giving the briquet a high resistance. From 50 to 75 per cent of coke is usually required. This is more than is used in the retort process, but the distilled briquets can be crushed and used a second time as coke, or may be used as a high-ash fuel.

The usual proportion of pitch required is 10 to 15 per cent, more being required for fine ore and coke than for coarse. More pitch than this would be squeezed out in pressing the briquet, while less than this does not give a satisfactory briquet. A high-melting

⁵⁵ Fulton, C. H., Electric resistance furnace of large capacity for zinc ores: *Trans. Am. Inst. Min. and Met. Eng.*, vol. 64, 1920, p. 188.

pitch (melting point 170° to 200° F.) giving upon distillation 55 to 60 per cent of good, firm coke is the best binder. The pitch may be partly replaced by coking coal, but with less satisfactory results.

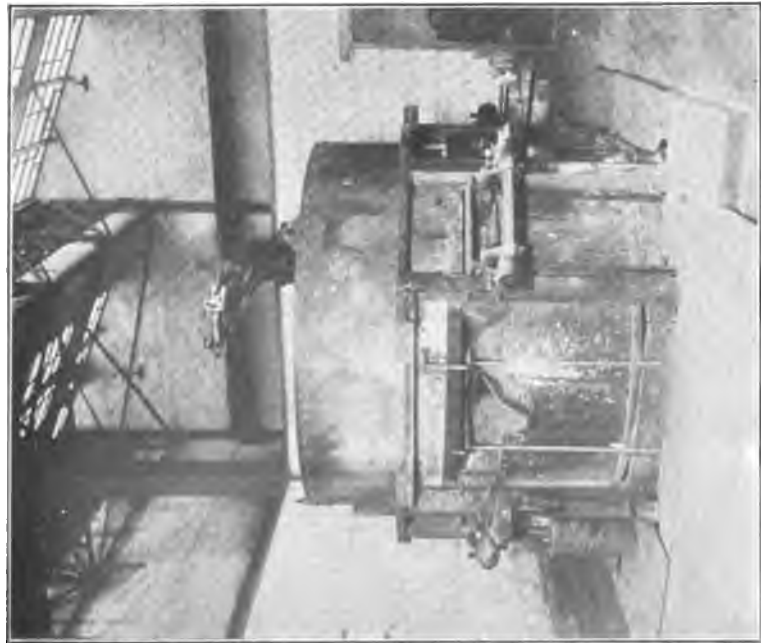
After each briquet was pressed it was set on end in a light sheet-iron cylinder slightly larger than the briquet on a flat car holding 20 briquets, and the space between the briquet and cylinder filled with crushed coke to prevent the oxidation of the surface of the briquet during baking. The car was then pushed into the baking oven. The baking was continued for 6 to 8 hours with a final temperature of 450° to 500° C. for the last few hours. This baking served to coke the pitch binder and after baking the briquet could be thrown violently to the floor without breaking. During the baking the briquet passes through a soft stage up to about 300° C. and above this temperature gradually increases in strength as the distillation of the pitch continues.

The electrical resistivity of the raw briquet is 26 to 30 ohms per cubic inch. This drops slowly as the pitch is coked until at 450° to 500° C. it drops suddenly to 0.6 to 0.7 ohm per cubic inch. At 900° to $1,100^{\circ}$ C. it is 0.015 to 0.04 ohm per cubic inch.

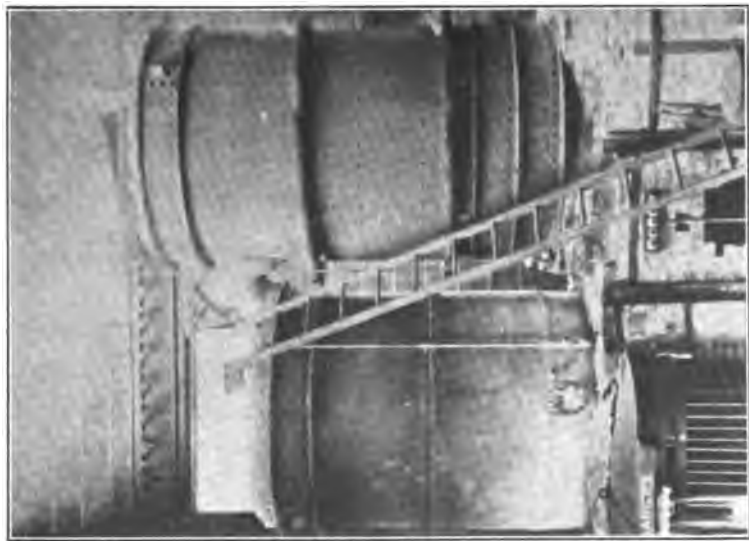
FURNACES.

The furnaces used at East St. Louis held a charge of 36 briquets, arranged in 12 columns of three each, set within a circle and operated on a three-phase circuit, four columns to a phase, connected in the usual Y connection. The amount of ore in one charge in this furnace was approximately 1,700 pounds, requiring 6 hours for distillation. Two hours were required for setting up the charge and discharging, so that three charges could be worked off per 24 hours, giving a daily capacity of 5,100 pounds of zinc concentrate.

One of the early furnaces, designated as furnace D, is shown in Figure 30. Plate I, B, is a photograph of the same furnace. The condenser structure rested upon the center part of a specially constructed, three-part base and the end parts contained the electrodes and supported the briquet charge and covering retort. In operation, the briquet charge was set up on one retort base, the retort lowered into place, and the condenser placed in position. The joint between retort and condenser was made tight with a luting of clay. The current was then turned on and the charge distilled. Meanwhile a charge was set up on the other retort base. As soon as the first charge was finished the condenser was lifted out of the way with the crane. The retort was then lifted about 3 inches and moved to one side, sweeping the briquets over an apron plate to the floor, leaving the base ready for setting up a new charge. The hot retort was at once transferred to the other base, the condenser turned around and



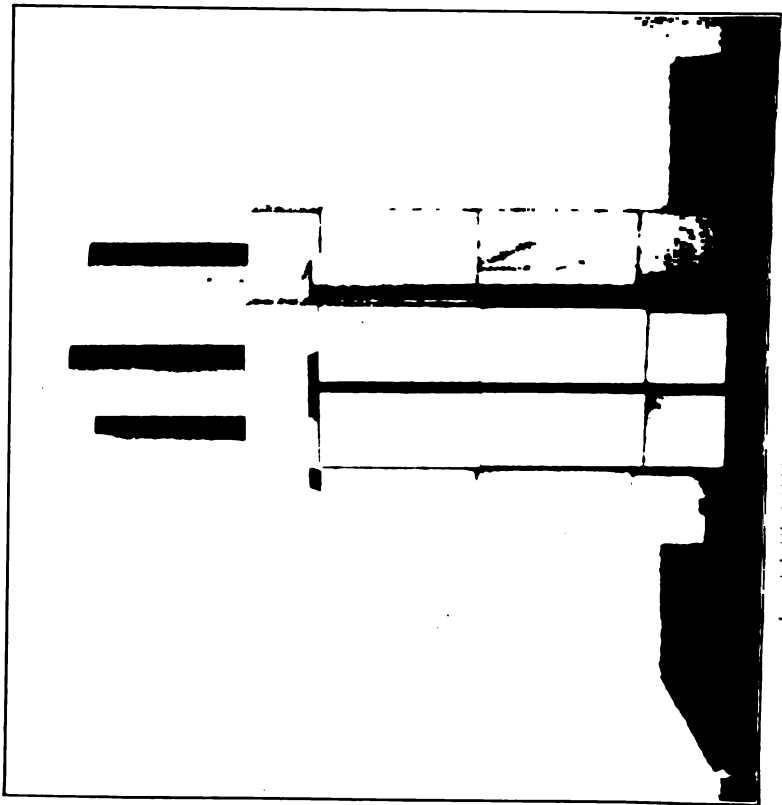
A. SNYDER'S FURNACE AT CHICAGO.



B. FULTON'S EAST ST. LOUIS FURNACE.



4 BRIQUET CHARGES FOR FRICTION TEST



1 FRICTION CHARGE FOR FRICTION TEST

were put into place, and the second charge distilled. The two end retorts were thus used alternately, so that the retort and condenser were never allowed to cool, and the same condenser served for both retort bases. When a cold condenser was started up, it could be heated by an oil burner introduced into the top of the large compartment, allowing the products of combustion to pass under the partition up the smaller compartment and out of the connector flue. With continuous operation, radiation about balanced the heat input due to the condensation of the zinc vapor and the condenser maintained

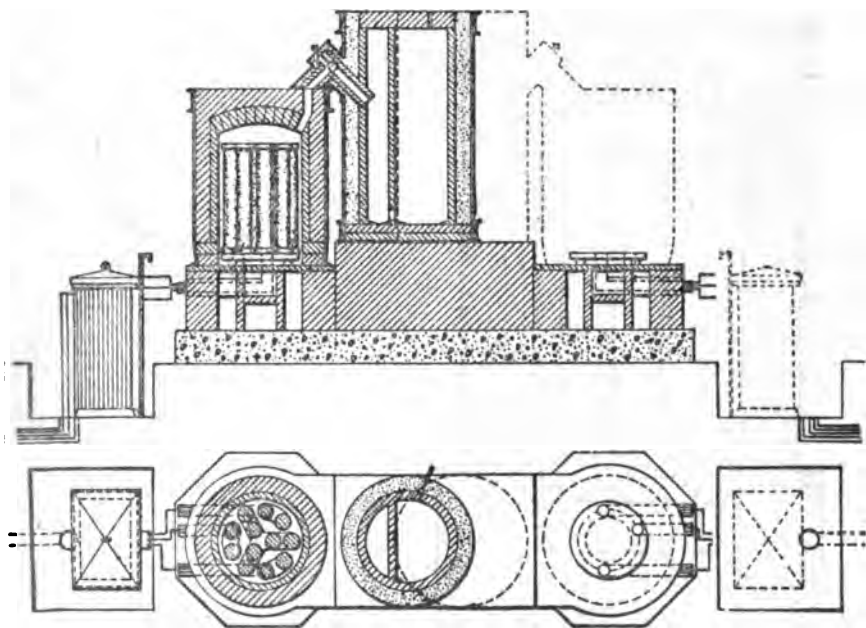


FIGURE 30.—Fulton furnace D.

itself at a temperature of about 700°C . at the inlet and 500°C . at the exit.

The retort consisted of a steel shell with a refractory lining. The inner lining was of high-grade fire brick and the space between this lining and the shell was filled with sil-o-cel for heat insulation.

The electrodes, which were of graphite, 6 inches square, entered the base of the furnace. On these were set 6 or 9 inch risers which reached up through holes in the base. The space between the risers and the walls of the opening was packed with finely crushed carbon. Graphite blocks rested upon these vertical plugs, and upon them were set the columns of briquets. The top connections were also made with 5-inch graphite blocks. Figure 31 shows the electrical arrangement and Plate II, A, is a photograph of the complete set-up.

A charge could be set up in from two to three hours by two men. After the charge was set up, a fire-clay luting was spread on the base where it made contact with the retort, and the latter was lowered into position by the crane.

With hot retorts, the voltage supplied to this furnace at the beginning of a charge was about 150 volts with an energy input of 120 to 150 kw., dropping to 75 to 90 volts with an energy input of 122 to 184 kw. as the distillation proceeded. This was the limit of the electrical equipment, and later with electrical equipment of

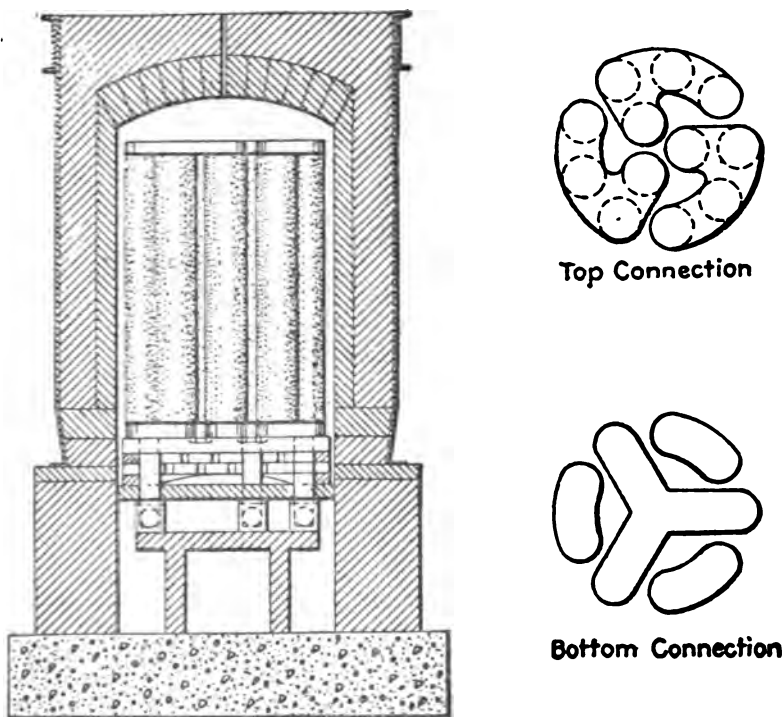


FIGURE 31.—Electrical connections of Fulton's East St. Louis furnace.

larger capacity it was shown that the furnace could easily take 200 to 250 kw. With the low-power input into furnaces D and E, the time of distillation was over eight hours and the power consumption greater, due to greater radiation, than that obtained later with a higher power input. The power factor was about 90.

Tables 2 and 3 show metal balances for several runs with this furnace. The furnace was practically new at the beginning of each series of runs, so that absorption of zinc was high.

Furnace E was a variation of furnace D and results were about the same. About 60 per cent of the zinc present in the charges was recovered as cast spelter, over 1,100 pounds being drawn from the condenser in one tap.

Furnace F was designed for use with new electrical equipment of larger capacity. There were three adjoining retort bases, a movable condenser, and one of the old retort bases arranged to preheat retorts by means of oil burners. The bases proper, which rested on the furnace foundation, were so built that they were transferable; extra ones were supplied, hence the base could be set on a car, the raw briquets placed on it, and the whole set-up moved into the baking oven and baked, and then transferred while still hot to the distilling furnace.

TABLE 2.—*Metal balance of Fulton furnace.*

[Runs 7, 8, 9, 10, and 11, May, 1917.]

| | |
|--|---------|
| Zinc in baked briquets: | Pounds. |
| Each charge | 573.0 |
| Zinc in five charges, total charged..... | 2,855.0 |
| <hr/> | |
| Zinc remaining in distilled briquets: | |
| Run 1, 2.36 per cent..... | 52.4 |
| Run 2, 0.67 per cent..... | 14.0 |
| Run 3, 1.80 per cent..... | 39.0 |
| Run 4, 0.93 per cent..... | 19.4 |
| Run 5, 5.97 per cent..... | 151.0 |
| <hr/> | |
| Total zinc in distilled briquets..... | 275.8 |
| Zinc cast as spelter..... | 1,242.5 |
| Zinc in scrapings from bridge wall, rechargeable at 41.75 per cent zinc..... | 899.0 |
| Zinc in brick from condenser, at 1.74 per cent zinc..... | 115.3 |
| Zinc unaccounted for..... | 322.4 |

SUMMARY.

| | Per cent. | Pounds. |
|-----------------------------------|-------------------|---------|
| Spelter tapped..... | 42.7 | 1,242.5 |
| Scrapings, rechargeable..... | 30.8 | 899.0 |
| Zinc in brick from condenser..... | 3.9 | 115.3 |
| Zinc in distilled briquets..... | ^a 9.1 | 275.8 |
| Unaccounted for | ^b 13.5 | 322.4 |
| <hr/> | | |
| | 100.0 | 2,855.0 |

^a The amount of zinc left in briquets is entirely at the will of the operator; this is shown on the individual runs. The runs in this series were, in part, undertaken to obtain figures on power consumption, and certain runs were discontinued at given power inputs to determine proper point of stopping.

^b This covers zinc absorption in the retort hood, base, and passageways and includes zinc escaping uncondensed.

TABLE 3.—*Metal balance of Fulton furnace.*

[Runs 12, 13, and 14, May 9-11, 1917.]

| | |
|----------------------------------|---------|
| Zinc in baked briquets: | Pounds. |
| Each charge..... | 573.0 |
| Total zinc in three charges..... | 1,719.0 |

Zinc remaining in distilled briquets:

| | |
|---|-------|
| Run 12, 6.57 per cent..... | 169.0 |
| Run 13..... | Nil. |
| Run 14, 0.131 per cent..... | 2.6 |
| Total zinc in distilled briquets..... | 171.6 |
| Zinc in scrapings from condenser, rechargeable, at 60.85 per cent zinc... | 496.0 |
| Zinc cast as spelter..... | 903.5 |
| Zinc unaccounted for..... | 147.9 |

SUMMARY.

| | Per cent. | Pounds. |
|--------------------------------------|-------------------|---------|
| Spelter tapped..... | 52.60 | 903.5 |
| Scrapings, rechargeable..... | 28.80 | 496.0 |
| Zinc left in distilled briquets..... | ^a 9.83 | 171.6 |
| Unaccounted for..... | ^b 8.77 | 147.6 |
| | 100.00 | 1,719.0 |

^a The amount of zinc left in briquets is entirely at the will of the operator; this is shown in the individual runs. The runs in this series were in part undertaken to obtain figures on power consumption, and certain runs were discontinued at given power inputs to determine proper point of stopping.

^b This covers zinc absorption in retort hood, base, and passageway, and condenser brick-work, and includes zinc escaping uncondensed.

The charge of briquets was then further heated by placing the hot retort from the previous charge over the new charge and allowing it to communicate part of its heat to the briquets. After about 1 hour, it was replaced with another retort, which had been preheated to about 1,500° C. with oil. After this had been on about 1 hour and the charge had reached distilling temperature, the current was turned on and the distillation finished by electric heat. To the cost of power must be added the cost of the oil for preheating the retorts, but the electrical power consumption was considerably reduced.

POWER CONSUMPTION.

Table 4 gives power-consumption figures for a series of runs. The initial charges with cold retorts and some underdistilled and over-distilled charges are omitted, as the power consumption in these cases varies from the normal. Runs 25 to 30 were with preheated retorts as described under furnace F.

TABLE 4.—*Power consumption at East St. Louis.*

| Run number. | Kw. h. per ton of ore. | Per cent zinc distilled from charge. ^a |
|-----------------|------------------------|---|
| 2 | 2, 270 | 100. 0 |
| 5 | 1, 645 | 98. 0 |
| 8 | 2, 240 | 98. 0 |
| 9 | 2, 180 | 93. 0 |
| 16 | 2, 200 | 96. 0 |
| 18 | 2, 120 | 98. 0 |
| 20 | 1, 760 | 91. 0 |
| 21 | 1, 720 | 95. 0 |
| 22 | 1, 920 | 97. 0 |
| 23 | 1, 760 | 95. 0 |
| 25 | 1, 665 | 99. 0 |
| 27 | 1, 548 | 100. 0 |
| 28 ^b | 1, 365 | 99. 0 |
| 29 | 1, 811 | 99. 0 |
| 30 | 1, 237 | 99. 0 |

^a Where less than all the zinc is distilled, the operation was purposely stopped to determine relation of extraction and power consumption.

^b Franklinite charge.

The theoretical power consumption for a 60 per cent zinc ore made into briquets of the usual composition was calculated to be 1,372 kw. h. exclusive of radiation losses. Of this, 421 kw. h. required to preheat the charges to 920° C., could be supplied by some such method as preheated retorts; and the carbon monoxide produced in the distillation is equivalent to 154 kw. h.

The energy in the briquet residue in cooling from 1,200° to 25° C. is 378 kw. h. Radiation can be made very small, as the heat is developed in the briquets themselves.

Runs 28, 29, and 30 were made to produce zinc dust with furnace F. A good grade of zinc dust, free from oxide, was made, 97 per cent passing through a 200-mesh screen; recoveries were very high.

The particular advantage claimed for the Fulton process is that it combines the physical and metallurgical characteristics of present-day retort practice with the other advantages possible in the electric furnace, such as generation of the heat in the charge itself, large units, low labor cost, and applicability to the treatment of low-grade ores. The charge is distilled in a closed retort within which a uniform temperature is maintained. This produces a gas nearly free from carbon dioxide, from which zinc can be condensed as spelter without the formation of blue powder. The process is applicable to high-grade ores of any composition and to complex ores, such as those of the Butte district. Impurities, such as iron and lead, which are so harmful in the retort process have no deleterious effect in the electric briquet furnace.

CONDENSATION.

The only serious condensation difficulty in the experiments at East St. Louis was the destruction of the condenser walls in the region where the temperature was about 500° C. This was found to be due to the reduction of specks of iron oxide in the refractories used and the deposition of carbon at these spots by the reaction $2\text{CO} = \text{CO}_2 + \text{C}$, which takes place at that temperature around the iron as catalyzer. This could be prevented by proper selection of refractory material for the condenser lining.

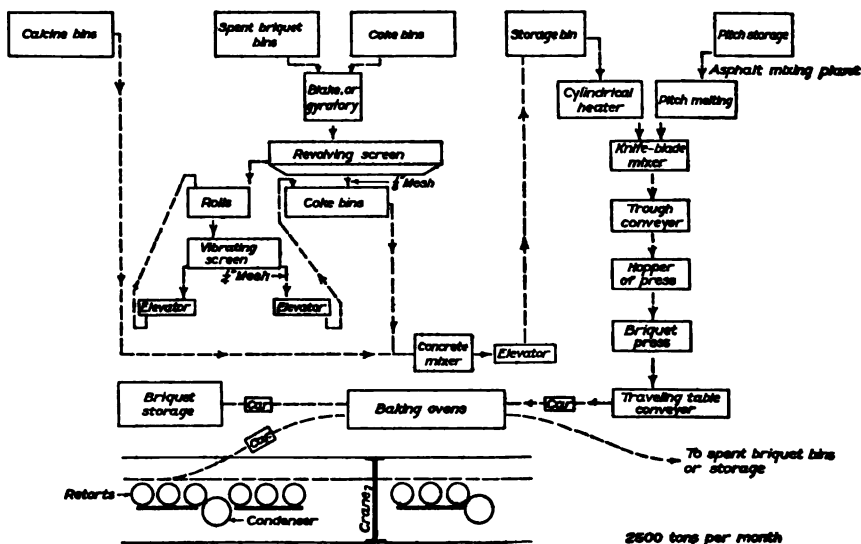


FIGURE 32.—Flow sheet of Fulton electrothermic zinc plant.

PROPOSED COMMERCIAL PLANT.

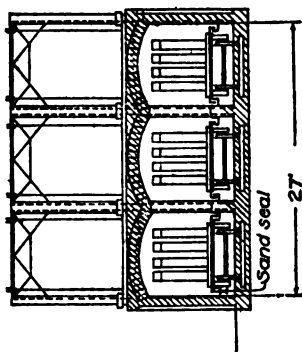
Fulton has recently described a proposed commercial plant for treating zinc ores by electrothermic dry distillation, and has estimated the cost of building and operating such a plant.^{35a} This is based upon the work at East St. Louis, but introduces some improvements later developed. His description of the plant and his cost estimates are quoted below:

GENERAL DESCRIPTION.

The basis for the proposed commercial plant is the treatment of 2,000 tons 60 per cent concentrate per month. For the treatment of high-grade zinc-bearing

^{35a} Fulton, C. H., A proposed plant for the electrothermic dry distillation of zinc ores: Eng. and Min. Jour., vol. 114, July 1, 1922, pp. 8-14.

material, it is assumed that one of three processes may be used: 1. The standard retort process. 2. The electrolytic process. 3. An electrothermic process. In all three methods the first step is roasting if sulphide concentrate is used.



Therefore in the proposed plant, the roasting furnaces are not considered and the start is made with calcines. Figure 32 is a flow sheet of the proposed plant. It consists of three sections: (1) The coke and briquet bins and crushing section, (2) the mixing and briquetting section, (3) the baking ovens and electric-distilling furnaces. The crushing plant is standard and needs no description. The coke and spent or distilled briquets are crushed to such fineness that the maximum size of particle is approximately one-eighth inch. The calcines and

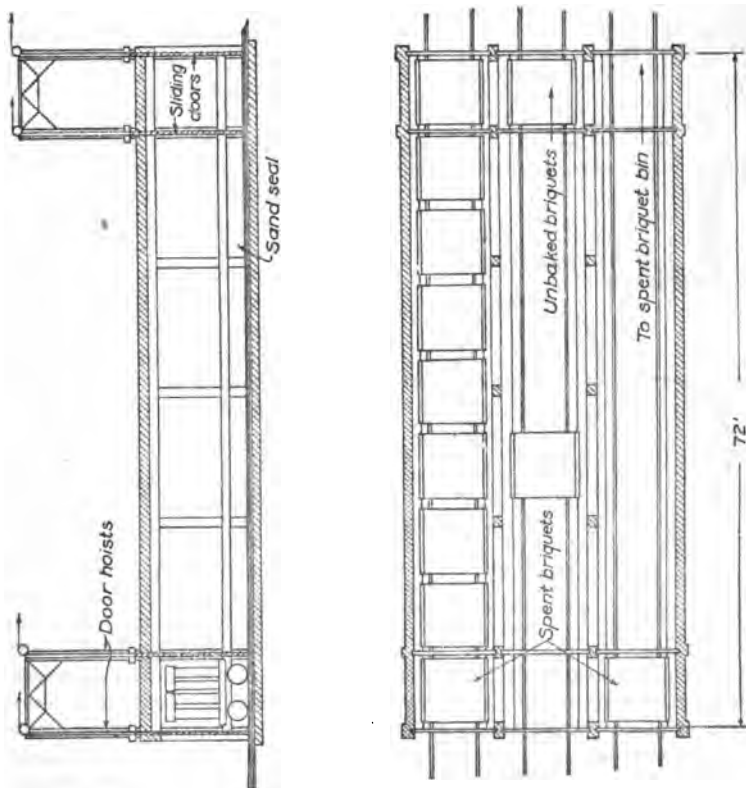


FIGURE 33.—Briquet baking oven.

the coke are fed in the proportion of 100 to 60 by conveyers to a concrete mixer and the mixture elevated to storage bins, from which it passes to the mixing plant which prepares it for briquetting. The mixing of the ore and coke with

the pitch is done in a standard asphalt road plant. In this the ore-coke mixture is heated to approximately 200° C. in a sand drier and raised to the sand bin and thence into a weigh box which in turn discharges into a knife-blade mixer. At this point the 18 to 20 parts of molten pitch are added by means shown in the illustration. The system of mixing results in a complete coating of the ore and coke particles with pitch. The mixer discharges to the hopper of the briquetting process by means of a conveyer. The press is standard and is known as a caking press used in other industries. The large briquets, 12 by 12 inches cross section and 27 inches high, are discharged on to a traveling table conveyer and then set up by hand in charge form on a furnace base placed on a baking oven car. These cars pass into the baking oven which is constructed on the

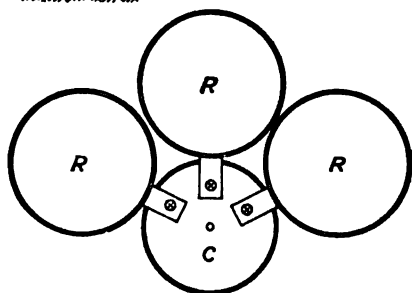
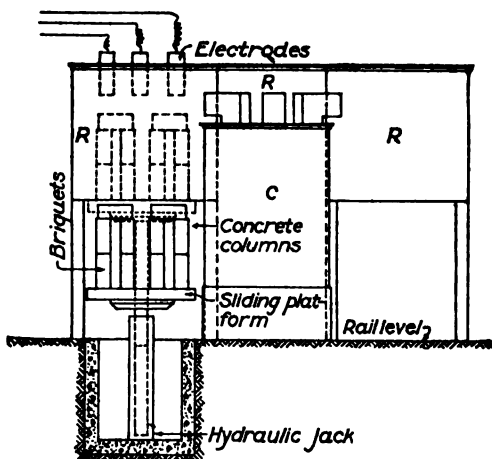


FIGURE 34.—Retort and condenser of plant proposed by Fulton.

recuperative principle and is shown in diagram form in Figure 33. The heat for baking is derived from the hot spent briquet charge which as it comes from the retort is immediately pushed into the baking oven. The distilling retorts, nine in number, are fixed in position and the furnace base is movable. This is a return to an original design used in the earlier experimental work and differs from the East St. Louis design of a fixed base and movable retort as shown in Figure 30 and Plate I, B. The fixed-retort design presents advantages for a commercial plant in connection with the baking of the briquets and the handling of the charge. The graphite electrodes (9 inches diameter) are located in the top of the retort, passing through coke-sealed glands and making contact with the briquet charge by their weight. The electrical equipment is located on a platform approximately level with the top of the retorts and the bus bars carrying the current are short. The furnace base with its charge, as it comes from the baking oven on a car, is run over the hydraulic lift below the retort and raised into place. The base is made tight to the retort by a ring of fire-clay luting. In operating, the furnace base with its distilled or spent charge will be lowered by the lift to a car and at once transferred to the baking oven, while a furnace base with a baked charge is taken from the oven and immediately lifted to the hot retort.

The three retorts in a group discharge into one condenser. The arrangement is shown in Figures 34 and 35. The connections for conducting the

zinc vapor from the retort into the condenser are permanently attached to the condenser and are as short and simple as possible. The condenser can be removed from its position by crane and replaced by another one when necessary.

Figure 36 illustrates the retort and condenser in diagram form and gives dimensions. The lining of the retort is high-grade fire brick and the insulating material is sil-o-cel. The lining of the condenser must be of a very pure fire clay or a synthetic refractory practically free from iron. This freedom from iron is one of the essential requirements of the condenser lining, since

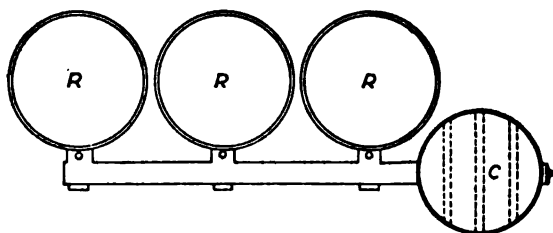
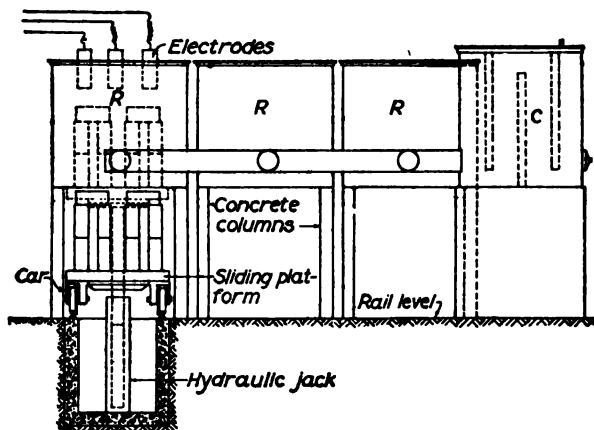


FIGURE 35.—Retort and condenser, alternate plan.

the presence of iron oxide particles in the lining leads to its destruction by carbon deposition from carbon monoxide gas—a reaction very active at about 500°C ., a prevailing temperature in some regions of the condenser. The arrangement of one condenser for three retorts permits of a continuous flow of zinc vapor to the condenser. The condenser may be arranged with either two or four passes or compartments, this arrangement being attained by curtain walls. With the combination of three retorts and one condenser, the volume of the condenser should be not less than twice the volume of the retort. The factors involved in condensation are temperature, volume, and surface. The temperature in the condenser will range from about $1,050^{\circ}\text{C}$. at the intake to about 470°C . at the gas exit. For every pound of zinc condensed from a mixture of equal volumes of zinc vapor and carbon monoxide

between 1,050° and 650° C., there is liberated 1,050 B. t. u., or 1,250,000 B. t. u. per hour for the condenser with three retorts as described. If the radiation from the condenser is greater than the quantity of heat supplied by the condensation of metal, the condenser will have to be heated. On the other hand, if it is less, the condenser will have to be cooled. In the design under discussion, the condenser will have to be cooled, which may be accomplished by water sprays. In starting a new condenser, it will be preheated to the proper temperature (700° C.) by means of an oil burner introduced into a handhole at the bottom of one of the passes, the condenser connection acting

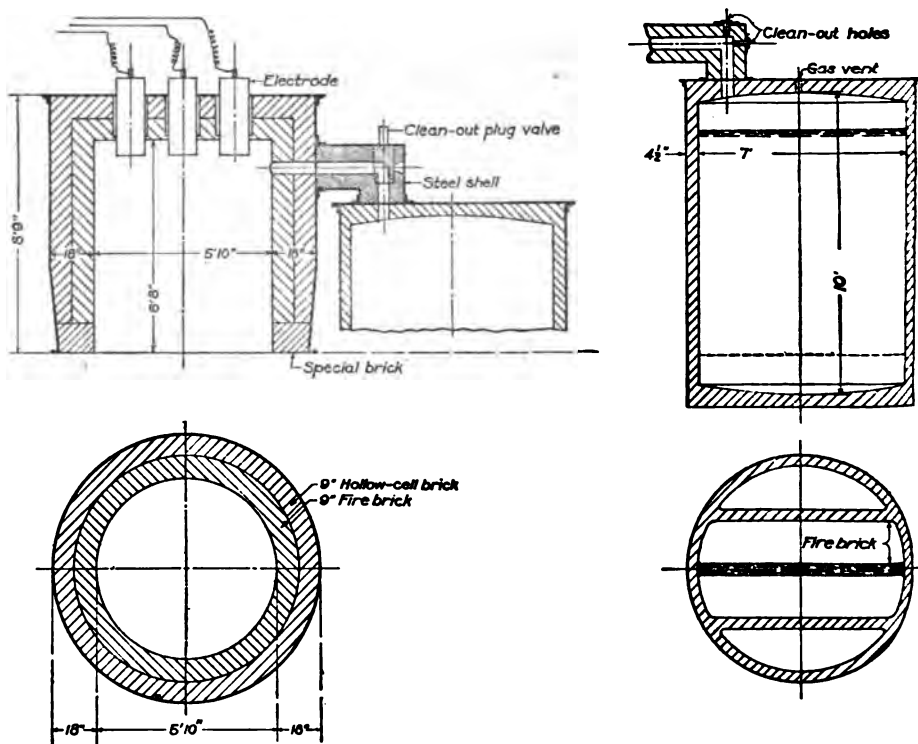


FIGURE 36.—Diagram of retort and condenser.

as a chimney. In condensing from a mixture of equal volumes of CO and Zn vapor theoretically 85 per cent of the zinc may be condensed to liquid metal with an exit temperature of 750°. In practice, it is very probable that two condensers will be placed in tandem, one operating between temperatures of 900° and 750° C. and the other between temperatures of 750° C. and 450° C. This system will afford good control. In the East St. Louis pilot furnace no difficulty was experienced in condensation due to the formation of blue powder. This is a common difficulty with electrothermic zinc smelting, but in this process the metallurgical conditions of the retort process are retained, namely, a uniformly heated space above 1,000° C. and a preheated charge so that no gases are present in the condenser except Zn vapor and carbon monoxide. It is the presence of such gases as CO₂, water vapor, etc., which leads to the formation of blue powder.

The spent briquets contain the residue of the ore. With the usual high-grade zinc concentrate this residue is from 12 to 20 per cent of the weight of the concentrate and from 15 to 25 per cent of the weight of the spent briquet, the rest of this being carbon—that is, the spent briquet is practically a high-ash coke. The briquets will be recrushed and used over again at least once, sometimes twice. If the ore contains lead and silver, approximately 80 to 85 per cent of the lead and over 90 per cent of the silver may be retained in the briquet by the close temperature control of distillation which the process permits. By using spent briquets containing lead and silver over a number of times, a relatively high concentration of these two metals may be obtained. This applies also to copper in complex zinc ores. By the reuse of the briquets the reduction fuel for the process is reduced to 30 per cent and less of the weight of the ore.

THE CHARGE.

The following figures give details for the briquet charge. The briquet mixture is assumed, for example, to be 100 parts calcine (Missouri concentrate, 60 per cent Zn), 60 parts coke, 20 parts pitch. In briquetted form this weighs 115 pounds per cubic foot. During baking approximately one-half of the weight of the pitch is distilled off and baked briquets contain 56 per cent ore, equivalent to 34 per cent zinc. The briquet is 12 by 12 inches in cross section and 27 inches long.

Plate II, *B*, illustrates a charge of briquets set up for introduction into the retort. In this charge, briquets are used as top and bottom connectors, so that the whole charge is made up of briquets. In East St. Louis, the bottom and top connectors were made of graphite, but briquet connectors present manifest advantages. The charge consists of 12 columns²⁰⁰ of 2 each, 6 top connectors, and 6 bottom connectors, a total of 36 briquets. A briquet contains 2½ cubic feet and weighs 260 pounds. The charge contains 81 cubic feet, weighs 9,315 pounds, and contains 5,216 pounds of ore, equivalent to 3,167 pounds of zinc. A retort distills three charges per 24 hours, so that the capacity per retort is 15,648 pounds, or 7.824 tons of calcine per day.

ELECTRICAL EQUIPMENT.

The furnace is operated with three-phase alternating current. The graphite electrodes make contact with the charge and are not consumed. The charge contains approximately 5,400 pounds of concentrate, and on the basis of 1,500 kw. h. per ton and a 6-hour distillation period (these two figures purposely assumed large, to provide proper margin), 4,050 kw. h. will be used in 6 hours, or an intensity of 657 kw. per retort. For the nine retorts, giving a capacity of (2.7 by 3 by 9)=72.9 tons calcine per day, or 2.187 tons per month of 30 days, or 2,573 tons green concentrate (a margin of 573 tons to allow for delays) there will be required nine 3-phase 60-cycle 700 kv. a. transformers, nine 3-phase 60-cycle 160 kv. a. regulators, and nine complete switchboard equipments. These are designed to give full kv. a. capacities at secondary voltages from 40 to 250 volts, with a primary voltage of 2,300 volts. Other primary voltages can be used equally well. It is assumed that the current can be purchased and is not generated at the plant. Other standard frequencies than 60 may be employed.

The specific resistivity of the briquet ranges from 0.01 to 0.04 ohm per cubic inch, dependent on the nature of the ore; 0.02 to 0.03 ohm is the usual figure.

²⁰⁰ For the best electrical connections it is almost essential to use 12 columns. Other numbers, while possible, present decided disadvantages in the set up.

A charge set up as in Plate II, B, shows a standard delta connection in the briquet columns, with four columns in series, and requires a terminal voltage of 217 volts at the electrodes for an intensity of energy input of 675 kilowatts; connections with the charge can, however, be made so as to lower this voltage if it should be desired. The transformers are set close to the retorts on a platform to keep the reactance low. By using electrodes in the top of the retorts the secondary leads of the transformers are about the same elevation as the electrode terminals of the furnaces, and short bus connections are obtained. All high-voltage control and all switchboard equipment will be outside the furnace room.

RECAPITULATION AND ADVANTAGES OF THE PROCESS.

The process differs essentially from other electrothermic processes for zinc. It is not a smelting process but a distillation process like the retort process. Most of the proposed electrothermic processes mix the ore with a limited amount of reduction fuel and smelt by electric heat generated by a buried arc. The furnace space is not uniformly heated and carbon dioxide gas and water vapor are present with the zinc vapor and carbon monoxide and condensation to liquid metal is very difficult and the amount of blue powder formed is large. The residue of the ore is smelted to slag which usually contains considerable zinc. It seems questionable whether in such a furnace it is possible to recover other metals in the form of bullion or matte on account of the high temperature prevailing. In Norway and Sweden electrometallurgists have gone frankly to vaporization of lead with the zinc. The electrothermic distillation process presents the following advantages:

1. Large scale units mechanically operated as in the metallurgy of the other common metals.
2. Low labor, mainly unskilled.
3. Practically complete extraction of the zinc from the ore.
4. High recovery of zinc and most of it as liquid metal.
5. Low power consumption, less than electrothermic smelting and less than in the electrolytic process.
6. Applicable to all types of zinc ores with no limitations on iron, lead, lime, or other substances now barred by the retort process.
7. Applicable to complex ores, for example, such as the Burma ores.
8. Simplicity of plant and operation. Simpler than the retort process and the electrolytic process.
9. No regular consumption of fire clay; can be built in small units in out-of-the-way places where electric power is available.
10. Lower cost of operation than the retort process or electrolytic process as set forth further on.
11. May be introduced as an adjunct to the retort process for the treatment of the blue powder produced by that process. In plants using nonregenerative gas-fired Hegeler retort furnaces with waste-heat boilers, excess steam for electric power is available.

Cost of proposed electrothermic plant.

[Capacity, 2,000 to 2,500 tons of green concentrate per month. The figures are based on the flow sheet (fig. 32).]

| | |
|--|----------|
| Calcine bins, 200 tons, and elevator----- | \$15,000 |
| Coke-crushing bins complete with rolls, screens, and 200-ton storage bin_ | 20,000 |
| Mixing system, drying, pitch melting up to the briquet process (this includes preliminary concrete mixer for mixing calcine and coke)--- | 22,000 |

| | |
|--|---------|
| Three briquet presses and table conveyer..... | 25,000 |
| Baking oven and equipment and 27 charge cars..... | 22,000 |
| Nine retorts..... | 20,000 |
| Nine rams..... | 30,000 |
| Three condensers..... | 10,000 |
| Twenty-seven 200-kv. a. single-phase transformers and switchboard equipment ^a | 70,000 |
| Buildings and cranes..... | 75,000 |
| Miscellaneous..... | 20,000 |
| | <hr/> |
| | 329,000 |

^a This is based on the assumption that high-tension power can be purchased. If power is generated at the plant, an investment of \$500,000 is necessary for a 5,000-kw. plant based on the price of \$100 a kilowatt.

The cost of a roasting and sulphuric acid plant will be about \$700,000.

CALCULATION OF COST OF ELECTROTHERMIC DRY DISTILLATION PROCESS.

Basis, 1 ton of roasted concentrate, produced from 60 per cent green concentrate. Sixty tons roasted concentrate per day, or 2,000 tons green concentrate per month. Sixty tons calcine; 36 tons coke; 12 tons pitch or asphalt per day. Cost based on flow sheet. Average cost of labor at \$3 per man per shift.

I. Cost of calcine delivered to furnace in form of baked briquets
1 foot square, 27 inches long.

II. Cost of furnacing and producing metal ready for shipment.

III. Cost of material, coke, and pitch.

IV. General expense.

V. Power.

Item I. Cost of delivering briquets to furnace:

| | | Cost per ton
calcine. | |
|---|--------|--------------------------|--------|
| 1. Unloading 60 tons calcine, 36 tons coke, 12 tons pitch, 2 men, at \$3..... | \$6.00 | \$6.00 | \$0.10 |
| 2. Crushing 36 tons coke and briquets in 8 hours— | | | |
| 1 man, at \$3..... | 3.00 | | |
| Power..... | .50 | | |
| | <hr/> | 3.50 | .06 |
| 3. Conveying and mixing ore and coke, 8 hours— | | | |
| 1 man, at \$3..... | 3.00 | | |
| Power, etc..... | 1.00 | | |
| | <hr/> | 4.00 | .07 |
| 4. Mixing for briquets, 24 hours— | | | |
| 6 men, at \$3..... | 18.00 | | |
| Power and fuel..... | 24.00 | | |
| | <hr/> | 42.00 | .70 |
| 5. Briquetting, 24 hours— | | | |
| 6 men, at \$3..... | 18.00 | | |
| Power, etc..... | 2.00 | | |
| | <hr/> | 20.00 | .33 |
| 6. Miscellaneous— | | | |
| 2 men, at \$3..... | 6.00 | | |
| Supplies..... | 7.00 | | |
| | <hr/> | 13.00 | .22 |
| | | | <hr/> |
| | | | 1.48 |

Item II. Cost of furnacing and producing metal for shipment:

972 briquets—27 charges, 9 retorts, 3 condensers.

| | | | |
|--|-------|-------|------|
| 1. Furnace men, 3 men, at \$6----- | 18.00 | | |
| 2. Helpers, 3 men, at \$3----- | 9.00 | | |
| 3. Metal drawers, 3 men, at \$3----- | 9.00 | | |
| 4. Metal to storehouse, 3 men, at \$3----- | 9.00 | | |
| 5. Repair gang, 3 men, at \$5----- | 15.00 | | |
| | | 60.00 | 1.00 |

Item III. Cost of material:

| | | | |
|---|--------|--------|------|
| 1. Coke, ^a 18 tons, at \$5.50----- | 99.00 | | |
| 2. Pitch, 12 tons, at \$9----- | 108.00 | | |
| | | 207.00 | 3.44 |

Item IV. General expense:

| | | | |
|---|-------|--------|------|
| 1. Superintendence----- | 10.00 | | |
| 2. Office expense----- | 10.00 | | |
| 3. Laboratory expense----- | 15.00 | | |
| 4. Yardmen, 3, at \$3----- | 9.00 | | |
| 5. Taxes and insurance and miscellaneous----- | 60.00 | | |
| 6. Repair supplies----- | 50.00 | | |
| | | 154.00 | 2.57 |

Item V. Power:

| | | | |
|----------------------------------|------|--|------|
| 1. 1,400 kw. h., at 5 cents----- | 7.00 | | 7.00 |
|----------------------------------|------|--|------|

15.49

Calculated to the basis of raw concentrate, assuming 15 per cent loss in roasting, equals-----

13.13

Credit on 9 per cent extra recovery, 108 pounds zinc, at 5 cents-----

5.40

7.73

^a Spent briquets used once again.

This does not include capital investment and amortization.

THE NATHUSIUS FURNACE.

Hans Nathusius in 1918 described several forms of a furnace in which the electrodes were embedded in the furnace lining, which was made of some material, such as dolomite, that would become an electrical conductor when hot. The heat was produced partly by the current flowing in this furnace lining, and partly by current flowing through the charge from one part of the lining to another. A uniform temperature was thus produced throughout the reaction zone.

In the furnace shown in Figure 37 the charge is fed into the preheating chamber *g* and thence down into the reaction chamber *r*, which is provided with electrodes embedded in the furnace lining. The zinc vapor and CO gas are drawn off through the passages *e* into the condenser *c*, where the zinc is condensed. The CO gas from the condenser is blown back through *f* into the preheater through the passages *d*, where it serves to preheat the incoming

charge. The exhaust gas is finally removed through the flue *b*. The furnace can be run on material which will remain granular during distillation, in which case the ashes are drawn off from an ash pit below the reaction zone, or if desired a fusible charge can be melted and the slag and bullion tapped from a crucible below the smelting zone.

Nathusius designed this furnace to waste as little heat as possible and in a test run reduced zinc oxide with a power consumption of 920 kw. h. per ton. He does not claim, however, that this could be duplicated in commercial work.

THE ZINC PLANT OF THE ELEKTROME- TALLURGISCHE WERKE, HORREM, GERMANY.

In 1919, A. J. Allmand and E. R. Williams described the electrothermic zinc plant of the Elektrometallurgische Werke, near Cologne, Germany.^{35c} They state that this plant began operations in September, 1913. They do not give the name of the inventor

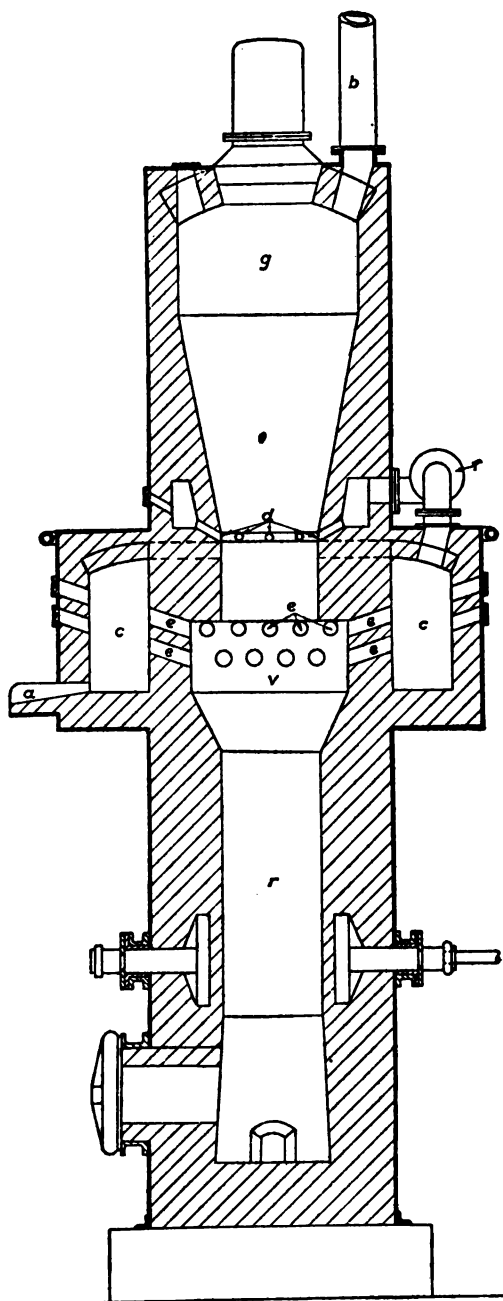


FIG. 37.—The Nathusius furnace (German Patent 208344, Aug. 9, 1916): *a*, Spout; *b*, flue; *c*, condenser; *d, e, f*, passages; *g*, preheating chamber; *r*, reaction chamber.

^{35c} Allmand, A. J., and Williams, E. R., Some chemical plants in the Cologne area, part II: Jour. Soc. Chem. Ind., vol. 38, Aug. 30, 1919, pp. 303-304 R.

of the furnace, but it seems to be similar to the furnaces designed by Queneau or Specketer. The following description of the process is quoted from their paper:

In this plant zinc is obtained electrothermally from a crude zinc oxide by reduction with coke. The ore used comes from the Harz, Ludwigshafen, etc., and contains about 70 per cent zinc with small quantities of lead, copper, silver, antimony, and cadmium. It is first mixed, slightly damp, with finely divided coke, and preheated to a dull red heat in a revolving roasting furnace (Bruchner type) by means of an oil and tar jet burner. This operation was said to take 8 to 10 hours. The charge, weighing about 5 to 5½ tons, and consisting of 70 per cent ore and 30 per cent coke, is then transferred to the electric furnace, hot from a previous operation, and smelted, this process taking anywhere from 24 to 36 hours. The zinc distills off, and is for the most part condensed and tapped as liquid, the remainder being burnt to zinc white.

The electric furnaces consisted of steel cylinders, 15 feet in length and 15 feet in diameter, mounted on trunnions and rotated on a horizontal axis by means of a gear and pinion. The shell was lined throughout by 18 inches of refractory chamotte material. Three charging doors were disposed symmetrically around the middle of the casing. Through each end of the latter passed four sets of electrodes, and copper rings and brushes made it possible to supply current in all positions of the furnace. The electrodes were of amorphous carbon rod about 5 inches in diameter and 3 feet long. From statements made it would appear that they simply carry current to the charge, which is sufficiently conducting to allow the heating currents to pass through it (high proportion of coke, preheating).

Three-phase current is employed, and with a current of 2,000 to 3,000 amperes, a voltage of 160 to 170 volts, and a power factor of 0.8; each furnace consumes about 500 to 600 kw. The exact figures, as also the time taken for the complete process, depend very much on the nature of the ore, skill of the workmen, etc.

The condenser, which abutted directly on the furnace, consisted of a brick-lined cylinder, also capable of rotation on a horizontal axis. The end of the furnace was pierced by three holes about 8 inches in diameter, placed symmetrically about 18 inches from the axis of the cylinder. To these corresponded three holes in the condenser, and the zinc vapors passed directly through and for the most part condensed to liquid. Uncondensed vapors were burned at the open end of the condenser and passed into a stack along a series of cooling flues and so to bag filters connected to a suction fan. The metallic zinc was tapped off every five to six hours and run into iron molds.

The total zinc recovery, as stated to us, was low—only 75 to 77 per cent. Of this, a certain amount is collected as zinc white (80 per cent zinc), each charge giving about 150 kg. of this product. The metal itself is 98 per cent pure, the chief impurity being lead. The slags from a single operation weigh at least 1,600 to 1,700 kg. and consist very largely of coke. At no time are they molten.

The plant when seen contained two preheaters and five electric furnaces. Four of the latter are constantly in action when working under full pressure, under which conditions a total of 300 tons of coke (for all purposes) and 10 tons of electrodes were stated to be required per month. Three hundred tons of zinc was made during the last half of 1918. Practically no work was done in November and December.

MISCELLANEOUS PROCESS AND FURNACES.

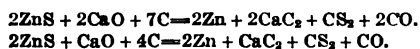
In addition to the more important furnaces described in the preceding pages, the following deserve mention either because of their historical interest or because of their possible bearing upon the future progress of electrothermic zinc metallurgy.

In 1889 an English patent³⁶ was granted to Thomas Parker on an electric smelting furnace which had the general form of a blast furnace, with electrodes introduced on either side of the smelting zone at the bottom of the shaft. The top of the furnace was closed and the charge was fed in by a screw conveyer. The following year a patent³⁷ was granted to Thomas Parker and Alfred Robinson on the application of this furnace to the reduction of roasted zinc ores with carbon.

Alfred Dorsemagen was among the first of several to attempt to smelt zinc ores in an electric furnace and to produce at the same time some valuable by-product. Dorsemagen's plan³⁸ was to reduce zinc oxide to zinc vapor and leave a residue of carborundum in the furnace, according to the equation:



Oliver W. Brown and William F. Oesterle at about the same time suggested³⁹ the possibility of producing zinc, calcium carbide, and carbon disulphide in one operation according to the reactions:



Paul Danckwardt proposed⁴⁰ to smelt mixed sulphide ores with lime, coke, and sodium sulphate in such proportions as to convert the sulphur of the ore into alkaline and alkaline earth sulphides while the zinc was liberated as vapor.

H. W. Hixon did some experimental work on a small scale in Mexico, using a buried-arc type of furnace, but had no success, due to his inability to obtain either metallic zinc or blue powder pure enough to be sold as such.⁴¹

³⁶ Parker, Thomas, Improvements in electrical furnaces and their manipulation, British Patent 17060, 1889.

³⁷ Parker, Thomas, and Robinson, Alfred, Improvements in the manufacture or production of zinc from oxidized ores of zinc, British Patent 12432, 1890.

³⁸ Dorsemagen, Alfred, Working zinc and substances containing silicic acid in electric furnace, U. S. Patent 716008, Dec. 16, 1902.

³⁹ Brown, O. W., and Oesterle, W. F., Metallurgical process, U. S. Patent 742830, Nov. 3, 1903. The electric smelting of zinc: Trans. Am. Electrochem. Soc., vol. 8, 1904, pp. 171-186.

⁴⁰ Danckwardt, Paul, Process of recovering zinc from sulphide ores, U. S. Patent 746798, Dec. 15, 1903.

⁴¹ Hixon, Hiram W., Electrothermic zinc smelting: Eng. and Min. Jour., vol. 101, June 17, 1916, pp. 1080-1081.

In 1908, Lindblad and Stalhane proposed to smelt zinc ores in a furnace similar to the Swedish electric iron-ore smelting furnace.⁴² The smelting would take place in the crucible of the furnace and any zinc vapor escaping up the shaft would be condensed in the descending charge and returned to the smelting zone, where it would progressively enrich the zinc vapor there produced, part of this being continuously withdrawn through an outlet at one side of the smelting crucible and condensed to spelter. The principle was somewhat similar to that of F. T. Synder's shaft furnace.

H. Takubo did considerable experimental work with a furnace of the buried-arc or slag resistance type.⁴³ He smelted about $1\frac{1}{2}$ tons of charge per 24 hours, with a power consumption of 1,300 kw. h. per ton of charge, and a recovery of 70 per cent of the lead and zinc.

A rotatable arc furnace designed by C. A. Weeks is shown in Figure 38. Weeks has stated that he successfully produced considerable zinc at a commercial cost from zinc scrap and dross, but was not so successful in treating ores.⁴⁴

Heinrich Specketer did a great deal of work in the smelting of zinc ores, using a variety of forms of rotating furnaces,⁴⁵ and was reported to have solved the condensation difficulty.⁴⁶

F. W. Highfield designed a furnace⁴⁷ in which the raw sulphide ore was first blast roasted to expel sulphur by means of air introduced into the bottom of the furnace; after this the blast was turned off and the charge smelted in the same furnace by electrical heat.

L. G. Rowand proposed⁴⁸ to allow the charge of ore and reducer to pass down a vertical shaft whose walls were lined with current-carrying resistors to furnish the necessary heat, or down a shaft which contains staggered rows of horizontal bars to serve as resistors.

A. G. Betts has suggested treating zinc ores or zinc slags with ferrosilicon or silicon for the reduction of the zinc.⁴⁹

Henry Swift Kimball in 1914 first suggested subjecting zinc vapor in the condenser to the action of an electric field as in the Cottrell process, claiming that this gave an increased recovery of liquid spelter.⁵⁰

⁴² Lindblad, A. R., and Stalhane, O., Method and furnace for the reduction of zinc, British Patent 25979, 1909.

⁴³ Ingalls, W. R., Unpublished report to Canadian Department of Mines on the electrothermic smelting of zinc ores.

⁴⁴ Weeks, Chas. A., Milling nonferrous metals in an electric furnace: *Met. and Chem. Eng.*, vol. 9, 1912, p. 363.

⁴⁵ Specketer, Heinrich, Process of and apparatus for producing zinc and other similar metals, U. S. Patents 1099211, June 9, 1914; 1231084, January, 1915.

⁴⁶ Ingalls, W. R., Zinc: *Mineral Industry*, vol. 21, 1912, p. 895.

⁴⁷ Highfield, F. W., Smelting of ores and apparatus therefor, U. S. Patent 1167176, Jan. 4, 1916.

⁴⁸ Rowand, Lewis G., Electric furnace, U. S. Patents 1289055, 1289056, Dec. 24, 1918.

⁴⁹ Betts, Anson G., Process of smelting zinc ores, U. S. Patent 918648, Apr. 20, 1909.

⁵⁰ Kimball, Henry Swift, Metallurgy, U. S. Patent 1189830, July 4, 1916.

In recent years, Sven Hultdt of Sweden has been granted several patents⁵¹ on methods of melting blue powder and refining zinc. The principle of his devices for melting blue powder is to heat the powder in a closed furnace to a temperature above the melting point of zinc, at the same time subjecting it to a rubbing action by giving the furnace a rotating or oscillating motion.

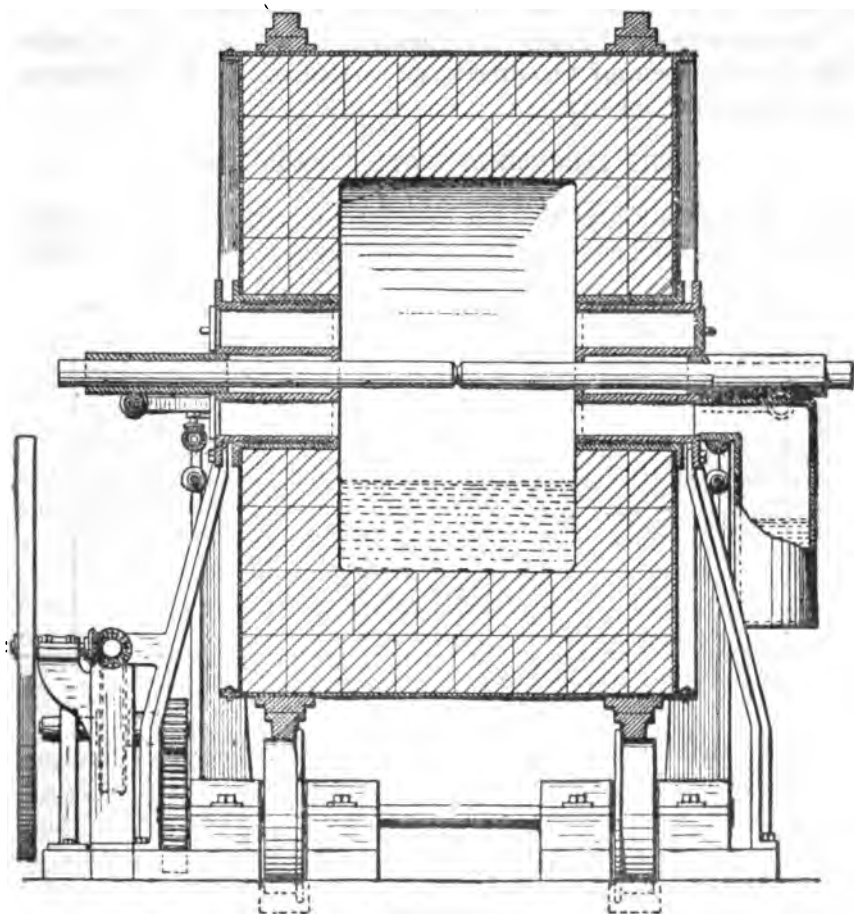


FIGURE 38.—Weeks rotatable arc furnace (U. S. Patent 949511, 1910).

Figure 39 is a diagram of E. S. Berglund's condenser for accomplishing the same result. The mixture of fluid zinc and powder collects in the bottom of the condenser and is pushed out through the opening δ by means of the screw conveyer β . This screw subjects the zinc powder both to a rubbing action and to a certain pressure,

⁵¹ Hultdt, Sven, Method of converting zinc powder into liquid zinc, U. S. Patent 1266808, May 21, 1918. Method of refining volatile metals, U. S. Patent 1298722, Jan. 4, 1919. Condenser for zinc vapors, U. S. Patent 1311604, July 29, 1919. Method of refining zinc, U. S. Patent 1312480, Aug. 5, 1919.

and materially increases the yield of metallic zinc. Berglund has also designed a condenser which will deliver the blue powder back to the smelting furnace as fast as it is condensed,⁵² and other improvements in the details of the operation of electric zinc furnaces.⁵³

E. A. Johansson has designed a condenser⁵⁴ in which the lower part of the condenser rotates, subjecting the blue powder to the action of stones or weights dragging on a horizontal surface.

The suggestions of Huldt, Berglund, and Johansson are all probably developments of the process as carried out at Trollhattan or other Scandinavian works.

THE ZINC CONDENSATION PROBLEM.

The greatest difficulty that has been encountered in all the attempts to smelt zinc ores electrically, and one that has caused the failure

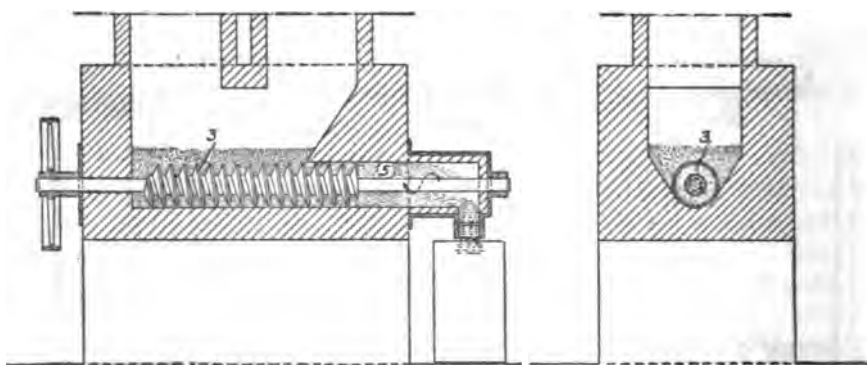


FIGURE 39.—Berglund zinc condenser (U. S. Patent 1331740, Feb. 24, 1910) : 3, Screw conveyor ; 5, opening.

of many otherwise promising attempts, is the difficulty of condensing the zinc vapor obtained from the smelting furnace to liquid zinc. In the earlier work, the major part of the zinc was invariably obtained as blue powder, which clogged the condensers and had to be resmelted to obtain a marketable product.

CONDENSATION OF PURE ZINC VAPOR.

Pure zinc vapor at atmospheric pressure begins to condense at 920° C. and will continue to condense at that temperature as long

⁵² Berglund, E. S., Method and apparatus for extracting zinc, U. S. Patent 1236395, Aug. 14, 1917.

⁵³ Berglund, E. S., Treating materials in electric furnaces, U. S. Patent 1160244, Nov. 16, 1915. Apparatus for preventing sintering in electric furnace, U. S. Patent 1196202, Aug. 29, 1916. Method of and apparatus for obtaining zinc, U. S. Patent 1207127, Dec. 5, 1916. Electrothermic extraction of zinc, U. S. Patent 1271267, July 2, 1918.

⁵⁴ Johansson, E. A., Condenser for zinc and lead vapors, U. S. Patent 1145685, July 6, 1915.

as more vapor is supplied to keep up the pressure. As rapidly as the zinc is deposited on a condensation surface, fresh vapor will move up to take its place, and as long as the temperature is properly regulated very little or no blue powder will be produced. The temperature may vary between rather wide limits, as zinc should be condensed to liquid at any temperature between the melting point of zinc and its boiling point. In practice there has been no great difficulty in distilling crude spelter or zinc scrap and obtaining a good yield of refined spelter. The Trollhattan and Sarpsborg plants were able to treat scrap and drosses at a profit long before they had any success with zinc ores, and more recently Thomson and Fitzgerald and others have redistilled considerable amounts of crude spelter in continuous runs, with almost complete condensation to liquid zinc.

CONDENSATION OF ZINC VAPOR DILUTED WITH CO GAS.

When reducing zinc oxide, however, according to the reaction $\text{ZnO} + \text{C} = \text{Zn} + \text{CO}$, a mixture of carbon monoxide and zinc vapor is produced in equal parts by volume. In a mixture of zinc vapor with other gases, zinc starts to condense when the partial pressure of the zinc vapor in the mixture is equal to the saturated vapor pressure of molten zinc. Zinc, therefore, will not start to condense from a mixture of gases at atmospheric pressure until a temperature is reached considerably below that at which pure zinc vapor will condense; then as zinc condenses the partial pressure of its vapor in the diluting gases is decreased and the temperature must be still further lowered to condense another increment of zinc. This must be continued until the amount of zinc vapor in the uncondensed gases is reduced to the desired amount. This is analogous to the condensation of water vapor from air. If the air is very nearly saturated, a slight lowering of the temperature will cause some moisture to condense, but comparatively dry air must be cooled to a much lower temperature before moisture begins to form.

Table 5⁵⁵ gives the temperatures at which condensation starts from various mixtures of zinc vapor and carbon monoxide, also temperatures at which the condensation is 50 per cent complete and 90 per cent complete, respectively.

⁵⁵ Stansfield, Alfred, *The electric furnace*, 2d ed., 1914, p. 331.

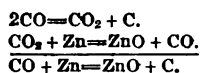
TABLE 5.—*Condensation of zinc from mixtures of zinc vapor and carbon monoxide.*

| Zinc in mixture. | Condensation starts. | 50 per cent condensed. | 90 per cent condensed. |
|------------------|----------------------|------------------------|------------------------|
| <i>Per cent.</i> | <i>Temp., ° C.</i> | <i>Temp., ° C.</i> | <i>Temp., ° C.</i> |
| 100 | 930 | 930 | 930 |
| 90 | 918 | 910 | 860 |
| 80 | 908 | 890 | 821 |
| 70 | 895 | 872 | 788 |
| 60 | 880 | 853 | 758 |
| 50 | 865 | 832 | 737 |
| 40 | 846 | 810 | 718 |
| 30 | 823 | 783 | 690 |
| 20 | 792 | 750 | 660 |
| 10 | 743 | 705 | 620 |

Zinc does not begin to condense from the mixture of equal volumes of CO and zinc vapor resulting from the reduction of zinc oxide by carbon until the temperature falls to that at which the vapor pressure of zinc is equal to half an atmosphere. This is at about 865° C. At 600° C. 1½ per cent of the zinc is still uncondensed. Furthermore, it requires an appreciable time for the zinc vapor to diffuse through the diluting gas to the surface of the condenser, so that if the temperature drops too suddenly the zinc will condense in the gas stream as minute drops which will be swept on and solidified as powder before they have an opportunity to coalesce. Any zinc vapor which remains uncondensed at the melting point of zinc (419° C.) will condense in the solid, powdery form as hoar frost. This latter, however, in the presence of only an equal amount of diluting carbon monoxide is less than 1 per cent of the total zinc.

In order to obtain a good condensation under the above conditions, it is necessary that the temperature of the condenser should be carefully regulated, with a gradual fall from 865° C. to only slightly above the melting point of zinc; that sufficient condensation surface be provided and so arranged that the zinc vapor need not diffuse through too great a distance in order to come in contact with it; and that the gas stream will pass through the condenser at a low enough velocity to allow ample time for the required diffusion and for small drops to coalesce and join the main body of liquid zinc before they reach the temperature at which they will solidify. As against these conditions we have the tendency of CO to dissociate at the temperature of the zinc condenser into carbon and carbon dioxide, according to the equation $2\text{CO}=\text{CO}_2+\text{C}$. This reaction is reversible. The mixture of CO₂ and CO in equilibrium with solid carbon at 1,000° C. contains only traces of CO₂, insufficient to oxidize zinc, but as the temperature falls the CO₂ increases until at the lower temperatures of zinc condensation (500° to 750° C.) CO₂ at equilibrium is over

50 per cent. Any CO_2 in the gas mixture above a small per cent has a strong oxidizing effect on zinc, and the final result would be



or a reversal of the carbon reduction reaction, were it not for the fact that the decomposition of CO to CO_2 and C takes place very slowly in the absence of catalyzers, and under favorable conditions the zinc is condensed before the above reactions have had time to take place to a marked extent. The tendency, however, is to coat the newly formed globules of zinc with zinc oxide and prevent their coalescence. To prevent oxidation of zinc because of this formation of CO_2 and that due to infiltration of air into the condenser, it is essential that the passage from the furnace to the condenser be as short as possible and that the zinc be condensed as rapidly as it can be without too sudden cooling.

EFFECT OF IMPURITIES IN THE ZINC VAPOR OBTAINED FROM SMELTING ORES.

Complex as is the problem of condensing zinc vapor from the reduction of pure zinc oxide under the most favorable conditions, when the question of condensing the vapor from the gases produced in smelting ores is considered, a multitude of new complexities is introduced. Zinc is oxidized by the CO_2 formed from the breaking up of carbonates and from the reduction of iron or other metal oxides at low temperatures, according to reactions such as $\text{FeO}+\text{CO}=\text{Fe}+\text{CO}_2$, and by the water vapor from moisture in the ore or from hydrous minerals. The zinc vapor is diluted more extensively because CO is formed by the reduction of other metals than zinc. The zinc vapor is, moreover, sulphidized by sulphur and SO_2 , and considerable amounts of very fine dust from the charge are likely to be carried over into the condenser by the rush of evolved gases, which will hinder efficient collection of the condensed drops of zinc.

An immense amount of time and work has been put in by all investigators of the subject in trying to solve these questions in their relation to the electrothermic metallurgy of zinc; and the variety of processes, furnaces, and condensers which have been developed is largely the result of the efforts in this direction.

TWO CLASSES OF BLUE POWDER.

The problem of blue powder may be considered in its relation to the electric smelting of ores under two heads. Blue powder is classed either as chemical blue powder, which is caused by an oxide, sulphide, or similar film on the condensed zinc droplets which prevents their collecting together in one mass (under this head can

perhaps also be classed the blue powder formed when a large amount of fine dust is carried over with the zinc vapor, though the action is really not chemical); or physical blue powder, which is caused by improper physical conditions in the condenser, such as poor temperature regulation, insufficient surface for condensation, or such a large ratio of volume to surface that diffusion of zinc to the surface can not take place rapidly enough, or such high velocity of the gas stream through the zone of favorable temperature that there is insufficient time for diffusion or for proper collection of the condensed globules of zinc.

CAUSES OF CHEMICAL BLUE POWDER.

Chemical blue powder results largely from conditions in the smelting furnace, though if its formation is to be prevented leakage of air into the condenser or the presence of iron or similar catalyzer of the reaction $2\text{CO}=\text{CO}_2+\text{C}$ in the condenser must not be allowed. In the smelting of zinc ores in fuel-fired retorts the cold charge is heated up uniformly and gradually to distilling temperature, so that moisture and CO_2 are expelled and easily reducible metals are reduced before zinc starts to distill. In addition, there is always present during distillation a large excess of hot carbon to convert any CO_2 formed to CO . The charge is so small and the evolution of vapors so gradual that little dust is carried over mechanically. Some of the electric furnaces of the intermittent type—as, for example, the Fulton furnace—duplicate these conditions very closely, and it is undoubtedly for this reason that they have given very good condensation results. Conditions in electric furnaces of the continuous-smelting type are quite different from those in a fuel-fired retort. Here the temperature in the furnace is not uniform, and if the raw charge is introduced cold, water vapor and CO_2 will be produced in the cooler parts of the furnace at the same time zinc vapor is being produced in the smelting zone. In those types in which the residue is tapped as liquid slag, matte, and bullion, no great excess of carbon can be present to reduce CO_2 , and in some types the heat is so localized and the evolution of gas from the smelting zone so rapid as to cause excessive dusting of the charge. The intense heat in the localized smelting zone may, moreover, cause such impurities as iron and silicon to be reduced and volatilized, and these will coat zinc globules in the condenser and hinder their coalescence.

The production of CO_2 and water vapor in the smelting furnaces can be largely prevented by preheating the charge before it enters the electric furnace; this is one of the chief advantages of preheating. Preheating also prereduces part of the metals other than zinc and thus saves some dilution of the zinc vapor in the smelting furnace.

Several experimenters have tried to correct the lack of excess carbon in the charge by passing the vapors through a column of hot carbon after they leave the furnace, but this seems to have been of doubtful value and with proper preheating is now considered unnecessary. The localization of heat and evolution of dust can be partly overcome by heating by the resistance of the charge between electrodes rather than by the arc, and the later furnaces of the buried-arc type all do this as much as possible. Some investigators have attempted to avoid most of these difficulties and at the same time to do away with the cost of roasting by treating green sulphide ores with metallic iron. This would give zinc vapor unmixed with CO or CO₂, but in practice it is found that the vapors of the metal employed and the sulphur vapors from the unroasted sulphides have as detrimental an effect as the gases produced by the carbon reduction.

CAUSES OF PHYSICAL BLUE POWDER.

The production of physical blue powder depends almost entirely upon the design of the condenser. The condenser used in the ordinary retort process is the result of development during many years of practical experience; and though it accomplishes the desired results fairly well, existing knowledge of the factors involved is not sufficient for successfully adapting its important features to the condensation of the immensely greater amounts of zinc produced from an electric furnace.

The temperature must be properly controlled. In the retort process, this is accomplished by the heat radiated from the front of the distillation furnace. In electric smelting, it has been a common custom to provide complicated means for heating or cooling the condenser artificially. It seems certain, however, that for a constant flow of gas it should be possible so to design the condenser that the radiation would just balance the heat produced from the condensation of the zinc, and extra means for controlling the temperature should not be necessary except when starting up a cold condenser.

The size of the condenser must be greatly increased to obtain the desired capacity, but how this is to be done is not known. If all linear dimensions of the retort condenser are proportionately increased, the volume is increased but the surface is not increased in proportion. If the surface is increased in proportion to the amount of vapor to be condensed, the volume has either been increased entirely too much or a complicated and unwieldy construction has been produced. It will probably be discovered later that a simple condenser of large capacity can be constructed which will keep the

essential features of the zinc retort condenser but will alter unessential ones to suit increased size. At present there is not enough known about the physics involved to determine certainly which are the essential features and which the unessential. During recent years, condensers of large capacity have been operated with satisfactory results, but they have been the result of many practical trials and more experimentation would probably be required to adopt them to smelting furnaces of different types or of much different size.

A CRITICAL STUDY OF THE RETORT CONDENSER.

F. T. Snyder has made an interesting critical study of the retort condenser.⁵⁶ He assumes the average temperature of the gas entering the condenser to be $1,050^{\circ}\text{C}$. at the center line of the retort and of gases leaving it to be 600°C . at the center line and 450°C . at the surface of the condenser. He finds that the average rate of condensation is 0.4 gm. of zinc per square centimeter of condenser surface per hour and the average cooling effect is 0.21 cal. per square centimeter per hour. The average velocity of the gases entering the condenser is 4.67 cm. per second, and of those leaving the condenser is 7.67 cm. per second; or a mean value of 5.17 cm. per second. The average time of the gas in the condenser is 10.6 seconds. The product of the velocity of the gases in centimeters per second times the lateral equivalent diffusion distance to the condenser surface (corrected to 864°C .) is 9.9 at the retort end and 9.1 at the nose end, an average of 9.5.

Snyder believes these are the important characteristics to be kept in mind when designing a condenser for an electric furnace.

PRESENT STATUS OF THE CONDENSATION PROBLEM.

In conclusion, it may be said that several investigators have succeeded in obtaining good yields of liquid zinc in large-scale tests covering many days or weeks, so that there is now no doubt that it can be done. On the other hand, Ingalls reports that the Scandinavian metallurgists no longer attempt to produce spelter in the first smelting, but find it more advantageous to run their furnaces at a high temperature to volatilize all zinc and lead, and to collect these as blue powder, which is melted and refined in a second furnace. It was formerly thought that blue powder could not be melted economically, but in Sweden and Norway it has been found feasible to do this by subjecting the powder to a rubbing action at melting temperatures.

⁵⁶ Snyder, F. T., The condensation of zinc vapor from electric furnaces: *Trans. Am. Electrochem. Soc.*, vol. 19, 1911, pp. 317-331.

When the physical and chemical factors involved are better understood, it will undoubtedly be found that proper condensation of zinc vapor to spelter may be fairly easily accomplished with a simple, properly designed condenser and a continuous flow of vapor.

COMMERCIAL POSSIBILITIES OF ELECTRIC ZINC SMELTING.

PRESENT COMMERCIAL DEVELOPMENT.

The electric smelting of zinc ores for the production of spelter can now be termed metallurgically feasible. The question of cost then becomes of paramount importance. Up to the present time, the plants in Norway and Sweden are the only ones that have been able to achieve commercial success. Little data on their costs are available, but the plants have been able continuously to maintain and expand their operations during more than 15 years. They are favored by having very cheap power available, and it is probable that their process would not be commercially practicable in this country. Judging from published reports, their metallurgical results are not as good as have been obtained in much of the work in this country.

BASIS FOR ESTIMATING COSTS.

Consideration of the prospects for the electrothermic process in America, then, must depend upon costs estimated from results of experimental runs with furnaces of semicommercial size. Many cost estimates have been published in the technical press, some based upon purely theoretical considerations, some on laboratory experiments, but only a few impartial ones have been based on really reliable data. The record of the development of electric smelting is full of discussions of the advantages of the electric furnace over other processes. Some of these are unprejudiced reviews of the advantages and disadvantages of each process, some are the views of overoptimistic inventors, while some claims for the electric furnace have been made which are so extravagant as to discredit the whole subject. The electric furnace undoubtedly has so many advantages that it may stand on its true merits, and its progress will be best served if those who expect great things from it will realize that there are some points which must be worked out under actual operating conditions before the maximum possibilities will be realized. Moreover, the same judgment and attention to local conditions must be employed as in adapting any other process to a particular problem.

Local conditions as to cost of power, labor, material, and character of ore supply vary so much from place to place and from year to year that detailed cost estimates are not especially valuable in a

general consideration of the subject. It will be more useful to discuss separately the different items that enter into the total cost and to compare these items with the corresponding ones in competitive processes; then from these data estimates can be made by those interested in any particular local problem by taking local conditions into consideration.

POWER.

AMOUNT OF POWER REQUIRED.

The largest single item of cost in the electrothermic process is electric power. One of the popular talking points in favor of the electric furnace as compared with the retort furnace has been that the heat was generated in the charge itself instead of having to be conducted through the retort walls, and hence was used very much more efficiently. Electric power as a source of heat is, however, much more expensive than coal, so that as close an estimate as possible must be formed of the amount which will be required in a commercial-sized furnace. The heat absorbed in the reduction of zinc oxide by carbon, according to the reaction $\text{ZnO} + \text{C} = \text{Zn} + \text{CO}$, starting with cold ZnO and C and ending with cold Zn and CO, is 55,640 cal. per mole. With cold zinc oxide and carbon at the start and with zinc vapor and CO at 1,200° C. at the finish, the heat absorbed is 102,280 cal. per mole, or 1,565 kg. cal. per kilogram of zinc = 1,419,768 kg. cal. per ton (2,000 pounds) of zinc. This is equivalent to 1,651 kw. h. per ton of zinc.

In the reduction of ores additional heat must be supplied to heat the gangue and excess carbon to smelting temperature and for other reactions which take place. Gin calculated the energy theoretically required to smelt a 50 per cent calcined calamine ore as 945,500 cal. per metric ton.⁵⁷ Richards has corrected Gin's figures and obtains 1,140,180 cal. per metric ton = 1,326 kw. h. per metric ton, or 1,203 kw. h. per ton of 2,000 pounds.⁵⁸ For a recovery of 900 pounds of zinc per ton of ore this would be 2,673 kw. h. per ton of zinc produced. Fulton gives the theoretical power consumption for 60 per cent zinc ore smelted in his briquet furnace as 1,372 kw. h. per ton of ore.⁵⁹ For 90 per cent recovery, this would be 2,541 kw. h. per ton of spelter produced.

The power required per ton of ore decreases as the percentage of zinc decreases but not in direct proportion, since more power is lost as sensible heat in the increased amount of gangue present. The power per ton of zinc produced therefore increases as the percentage of zinc in the ore decreases.

⁵⁷ Gin, Gustave, *The electrometallurgy of zinc*: Trans. Am. Electrochem. Soc., vol. 12, 1907, pp. 117-139.

⁵⁸ Richards, J. W., *Metallurgical calculations*, 2d ed., vol. 3, 1908, pp. 611-614.

⁵⁹ Fulton, C. H., *Electric furnace of large capacity for zinc ores*: Trans. Am. Inst. Min. Eng., vol. 64, 1920, pp. 188-226.

In practice, radiation losses and transmission losses must be added to the theoretical power required, but, on the other hand, part of the necessary heat may be supplied by nonelectrical means and part of the power may be recovered by burning the CO produced, and in some furnaces from the sensible heat of the products.

Snyder stated⁶⁰ that the power consumption per ton of raw ore in the experimental furnaces of the Canada Zinc Co. at Vancouver over a period of eight months approximated closely the empirical formula: Kilowatt hours per ton of raw ore = $650 + 5$ times the per cent of zinc present. Applying this to different grades of ore, he arrives at the following table:

| | Kw. h. required— | |
|---|------------------|---------------|
| | Per ton ore. | Per ton zinc. |
| Pure zinc oxide—80 per cent zinc----- | 1,050 | 1,315 |
| Calcined calamine—50 per cent zinc----- | 900 | 1,800 |
| Blende—52 per cent zinc----- | 910 | 1,750 |
| Broken Hill—26 per cent zinc----- | 780 | 3,000 |

These are lower than the theoretical power consumptions given above. In Snyder's furnace, however, the zinc was condensed in the furnace and left the furnace as liquid zinc. The CO produced was used for preheating the charge. Richards has shown⁶¹ that if the zinc and CO leave the furnace at 500° C. the figures given are theoretically possible. The power consumption given for pure zinc oxide would correspond to a furnace efficiency of 88 per cent, or if the heat absorbed in chemical reactions is subtracted and the efficiency calculated on the remainder it is 46 per cent. The power requirements in the larger plant of the Canada Zinc Co. at Nelson were not published.

Salgues reported⁶² that in a 100-kw. furnace smelting 40 to 45 per cent ore 5 kg. of zinc were produced per kilowatt day, or 1 ton of zinc per 4,355 kw. h.

Stansfield smelted an ore averaging 25 per cent zinc and 25 per cent lead in a 15-kw. furnace for 800 to 850 kw. h. per ton of ore.⁶³ Stansfield made efficient use of the heat in the products of reaction, but he could not produce liquid spelter because of condensation difficulties.

The power consumption at Trollhattan during the tests made by F. W. Harbord on Broken Hill slime containing 36 per cent zinc and 24 per cent lead was 2,078 kw. h. per ton of ore.⁶⁴ This included 500 to 600 kw. for resmelting blue powder.

The power used by the Cote and Pierron furnace is reported in various tests as from 1,700 to 2,200 kw. h. per metric ton of ore.

⁶⁰ Snyder, F. T., *Trans. Am. Electrochem. Soc.*, vol. 12, 1907, pp. 138-139. Discussion of paper by Gin.

⁶¹ Richards, J. W., *Metallurgical calculations*, 2d ed., vol. 3, 1908, p. 616.

⁶² Stansfield, Alfred, *The electric furnace*, 2d ed., 1914, p. 323.

⁶³ *Engineering and Mining Journal*, Zinc smelting at Trollhattan; summary of a report by F. W. Harbord: Vol. 93, 1912, pp. 314-315.

The 500-horsepower furnace being built at Epierre in 1917 was expected to require from 2,000 to 2,200 kw. per metric ton, or about 1,800 to 2,000 kw. per short ton of ore.⁶⁴ This was divided in the proportion of about 75 per cent in the smelting furnace and 25 per cent in the auxiliary condenser furnace.

In the largest run with the experimental furnace of the Canadian Department of Mines at Montreal, smelting about 700 pounds of ore in 24 hours, the power consumption was at the rate of 1,610 kw. h. per ton of ore. A determination of the power consumption on one day when the furnace at Nelson, British Columbia, was running well, in April, 1914, during which time 1.17 tons of ore were smelted, showed a consumption of 1,436 kw. h. per ton of ore. The furnace had a power factor of about 75 per cent.⁶⁵

Peterson smelted 50 per cent zinc concentrate in a furnace having a capacity of 1 ton per day with 1,100 to 1,300 kw. h. per ton of concentrates.⁶⁶

Johnson reported on short runs made with a 1-ton furnace in 1912, power consumption of 1,000 to 1,700 kw. h. per ton of preheated charge and from 3,600 kw. h. per ton of spelter on Joplin ore to 9,000 kw. h. per ton of spelter on low-grade ore.⁶⁷ For a 40-day run made in 1914 he reported 1,490 kw. h. per ton of charge.⁶⁸ The ore used in this run contained about 35 per cent zinc.

In Fulton's experiments at East St. Louis, the power consumption without preheating ranged from 1,700 to 2,300 kw. h. per ton of ore, excluding the initial runs with new furnaces and some under-distilled and overdistilled charges (see p. 63). The power consumption with preheated retorts, excluding the initial run, was from 1,237 to 1,811 kw. h. per ton of ore, the zinc distilled from the charge in every case being 99 per cent or better. With the exception of the data given for Harboard's tests and for the Cote and Pierron furnace, the power for redistilling the blue powder produced must be added to the above figures. If good condensation to liquid spelter can be obtained the treatment of blue powder will be a small expense, but if much blue powder is produced this expense may be considerable.

A conservative estimate for the total power requirements of a properly constructed furnace of 5 or 10 tons daily capacity in con-

⁶⁴ Flusin, G., Recent developments of the Cote and Pierron electric zinc-smelting process: *Met. and Chem. Eng.*, vol. 18, Jan. 1, 1918, p. 17.

⁶⁵ Ingalls, W. E., Unpublished report to Canadian Department of Mines on the electrothermic smelting of zinc ores.

⁶⁶ Peterson, P. E., The electric furnace for zinc smelting: *Trans. Am. Electrochem. Soc.*, vol. 24, 1913, p. 222.

⁶⁷ Johnson, W. McA., The art of electric zinc smelting: *Trans. Am. Electrochem. Soc.*, vol. 24, 1913, pp. 197-199.

⁶⁸ Johnson, W. McA., The commercial aspect of electric zinc-lead smelting: *Trans. Canadian Min. Inst.*, vol. 17, 1914, p. 122.

tinuous operation would seem to be from 3,000 to 3,500 kw. h. per ton of spelter produced, smelting a cold charge, or from 2,500 to 3,000 kw. h., with a preheated charge. It is not unreasonable to believe that this could be lowered still further when large-scale operations have been standardized. Low-grade ores may use somewhat more power per ton of spelter produced than that given, but these will usually contain valuable by-products the recovery of which with simultaneous extraction of the zinc will give an increased margin for profit.

COST OF POWER.

The cost of power varies widely in different localities so that only very general statements can be made on this subject. An electrothermic plant, like an electrolytic plant, will usually be located near a source of cheap power.

Table 6^o gives costs of electric power for certain cities in 1916.

Table 7^o gives the average annual production cost of the electric utilities in the superpower zone for 1919.

TABLE 6.—*Electric power costs in 1916.*

COST PER HORSEPOWER A YEAR, 24-HOUR POWER, 85 PER CENT LOAD FACTOR, 6 DAYS PER WEEK.

| Company or municipality. | 500 horse-power. | 1,000 horse-power. | 2,000 horse-power. |
|---|------------------|------------------------|------------------------|
| Western United Gas & Electric Co., Illinois..... | \$72.75 | \$72.75 | \$72.75 |
| Munroe Electric Co., Wisconsin..... | 118.80 | 118.80 | 118.80 |
| Springfield Gas & Electric Co., Springfield, Mo..... | 97.00 | 97.00 | 97.00 |
| Cleveland Illuminating Co., Cleveland, Ohio..... | 49.15 | <i>H. T.</i>
36.93 | <i>H. T.</i>
36.93 |
| Edison Electric Illuminating Co., Boston, Mass..... | 67.70 | <i>Power.</i>
67.30 | <i>Power.</i>
67.00 |
| Pittsburgh, Pa..... | 41.10 | 34.65 | 31.50 |
| Incuman (Argentina)..... | 162.00 | 161.30 | 161.20 |
| People's Incandescent Light Co., Meadville, Pa..... | 71.00 | 71.00 | 71.00 |
| Utah Power & Light Co..... | 54.37 | 50.05 | 47.89 |
| The United Illuminating Co., Bridgeport, Conn..... | 95.20 | 95.20 | 95.20 |
| The Evansville Public Service Co., Indiana..... | 83.36 | 82.00 | 81.00 |
| Erie County Electric Co..... | 114.00 | | |
| United States Reclamation Service, Minidoka project, Idaho..... | 46.47 | | |
| The Merchants Heat & Light Co., Indianapolis, Ind..... | 58.65 | | |
| City of Leeds, England..... | 41.83 | | |
| Duluth Edison Electric Co..... | 113.90 | | |
| Detroit Edison Co., Detroit, Mich..... | 37.45 | | |
| Detroit Electric Co., Detroit, Mich., 4,700 volts..... | 32.58 | | |
| Buffalo General Electric Co. (effective Sept. 1, 1915)..... | 59.45 | 59.41 | 59.30 |
| New Orleans Railway & Light Co..... | 53.05 | 49.26 | 48.37 |
| The Edison Illuminating Co., Boston, Mass..... | 43.55 | 31.89 | 29.80 |
| Indianapolis Light & Heat Co..... | 58.67 | | |
| New York Edison Co..... | 101.42 | | |
| Omaha Electric Light & Power Co., Omaha, Nebr..... | 38.74 | 30.24 | 29.89 |
| Texas Power & Light Co., Dallas, Tex..... | 36.00 | 31.82 | 27.45 |
| Pacific Power & Light Co., Portland, Oreg..... | 75.40 | | |
| Berlin Public Service Co., Wisconsin..... | 240.00 | | |
| The Springfield Gas & Electric Co..... | 67.50 | | |
| Pasadena, Calif..... | 133.20 | | |
| Southern California Edison Co., Los Angeles..... | 46.51 | 46.51 | 46.51 |
| Portland Railway, Light & Power Co., Portland, Oreg..... | 37.50 | 34.46 | 32.94 |
| Winnipeg..... | 35.66 | 32.66 | 31.16 |
| Calgary..... | 53.20 | 52.71 | 52.47 |

^o Report of the Royal Ontario Nickel Commission, 1917, p. 81.

^o Murray, W. S., and others, A superpower system for the region between Boston and Washington: U. S. Geological Survey Prof. Paper 123, 1921, p. 49.

COST PER HORSEPOWER A YEAR, 24-HOUR POWER, 85 PER CENT LOAD FACTOR.

| Company or municipality. | 500, 1,000,
and 2,000
horsepower. |
|----------------------------|---|
| Philadelphia D..... | \$118. 70 |
| Philadelphia E..... | 90. 20 |
| Boston D..... | 76. 00 |
| New York..... | 116. 00 |
| Washington E..... | 76. 00 |
| Minneapolis..... | 60. 80 |
| Baltimore S..... | 69. 80 |
| Milwaukee Std..... | 63. 30 |
| Cleveland (wholesale)..... | 52. 20 |
| Providence G..... | 52. 20 |
| Chicago C..... | 63. 30 |
| Chicago C..... | 57. 00 |
| Buffalo..... | 72. 65 |
| Cobalt, Ontario..... | 54. 74 |

TABLE 7.—Annual production cost of electric utilities in the superpower zone, 1919.

| Geographical division. | Cost per kilowatt hour. | | Cost kilowatt year effective capacity. | |
|--------------------------|-------------------------|------------------------|--|------------------------|
| | Steam electric plants. | Hydro-electric plants. | Steam electric plants. | Hydro-electric plants. |
| Eastern New England..... | \$0. 0264 | \$0. 0102 | \$60. 00 | \$31. 30 |
| Western New England..... | . 0297 | . 0109 | 57. 50 | 29. 50 |
| Mohawk..... | . 0670 | . 0109 | 57. 25 | 30. 70 |
| Metropolitan..... | . 0182 | . 0194 | 52. 00 | 54. 90 |
| Hudson..... | . 0375 | . 0147 | 61. 20 | 38. 60 |
| Anthracte..... | . 0186 | . 0161 | 68. 80 | 54. 70 |
| Southern..... | . 0202 | . 0066 | 48. 20 | 38. 10 |
| Superpower zone..... | . 02124 | . 0094 | 54. 30 | 34. 70 |

Twenty-four hour continuous power can be obtained at power centers in Montana, Idaho, Washington, California, Utah, and Colorado at rates varying from 0.4 to 0.7 cent per kilowatt hour. Off-peak power, which could be used advantageously in an electrothermic zinc smelter, can be obtained at somewhat lower rates, in some cases as low as 0.25 cent per kilowatt hour.

COMPARISON WITH ELECTROLYTIC PROCESS.

A fair average for power required for electrolytic zinc is 3,500 kw. h. per ton of spelter produced. This is about the maximum that would be used in the electrothermic process, and it seems that the latter must have the advantage over the electrolytic process in the item of power cost. Except where fuel is costly and power cheap, this cost will be higher than the fuel item in retort smelting, but is balanced by other advantages.

LABOR.

It is difficult to estimate with any degree of accuracy the labor required for the electrothermic process, since with the exception of

the work in Scandinavia none of the trials has ever reached the stage where economy of labor was possible. It seems certain, however, that with the large units that can be used, the labor will be much less than for the retort process and probably less than for the electrolytic process.

REDUCTION FUEL.

The cost for reduction fuel is less in electrothermic smelting than in retort smelting. Continuous smelting in an electric furnace permits the use of only slightly more than the theoretically required carbon. Fulton's process requires a large amount of coke in the briquet, but the distilled briquets can be reused as coke and finally sent to a blast furnace for the recovery of the by-product metals, where the coke contained is of value as fuel in the blast furnace.

SUPPLIES AND REPAIRS.

The large cost for retorts and condensers in retort smelting is entirely absent when the electric furnace is used. Repairs to the furnace and accessories should be no more than to the furnace and accessories of a retort plant. In the continuous slagging type of electric furnace there is an expense for electrodes, but it should be no more than in many other well-known electric furnace processes. Cote and Pierron report an electrode consumption in a small zinc furnace of 25 pounds per ton of ore smelted and in a later furnace of 11 pounds per ton of ore. Johnson reports 6 pounds per ton of ore, and Peterson estimated 9 pounds per ton. In the intermittent type of furnace there is very little expense for electrodes.

METAL RECOVERIES.

The distillation of practically all the zinc from an ore is metallurgically possible in the electric furnace. In practice, the extraction is limited by the cost of power for distilling the last trace of zinc, but the practical limit is very high. After the zinc is distilled, the losses are low since the many causes of loss, such as broken retorts, retort absorption, and diffusion of zinc vapor through the walls of the retorts, are not present when an electric furnace is used. None of the experimental electric furnace runs have been long enough for obtaining accurate data on actual final recoveries, but recoveries of 90 to 95 per cent can probably be obtained in regular operation. In addition, most of the proposed electrothermic processes either recover lead, copper, and the precious metals as bullion and matte or as a product in very desirable shape for blast-furnace treatment, whereas the product of a zinc retort is very undesirable raw material for a blast furnace, as is also the usual sludge residue from an

electrolytic plant. The treatment of low-grade and complex ores in the electric furnace offers no difficulties, and the presence of the impurities which cause so much trouble in retort smelting is not detrimental. Electric smelting has the additional advantage that since zinc sulphide is decomposed by iron reduced from iron oxides in the charge a dead roast is not necessary. This decomposition of ZnS by Fe can not be utilized in the retort process because FeS formed would quickly destroy the retorts.

MANAGEMENT AND GENERAL EXPENSE.

The charge for management and general expense for a new electrothermic smelting plant, at the present stage of development of the process, would undoubtedly be greater than for a retort or electrolytic plant, for a good many operating details would have to be worked out before operations could be reduced to a routine basis; but once the process is standardized on a commercial basis the cost of management should be no greater than for a retort or electrolytic plant.

FIRST COST OF PLANT.

The cost of an electrothermic zinc plant will be less than for either the retort or electrolytic process, if power is purchased. If a power plant must be built, the total cost will be more than for the retort smelter but less than for an electrolytic plant with its power plant.

The electrothermic process has the additional advantage over the electrolytic process that a small electric smelting plant of one or more 10-ton units will be practicable, whereas the electrolytic process has given evidence that it is only economical where a large tonnage is to be treated.

SUMMARY.

Conditions being equal, the electrothermic process may be said to have the advantage over the electrolytic process in the items of power, labor, metal recovery, cost of roasting, first cost of plant, and in its adaptability to smaller scale operations; and over the retort process in the items of reduction fuel, labor, metal recovery, ability to treat low-grade and impure ores, cost of roasting, first cost of plant, and cost of retorts and condensers.

The electrothermic smelter must be near cheap power, as must also the electrolytic, but as many retort smelters are now some distance from their supply of ore this is no great difficulty. Each of the three processes—retort, electrolytic, and electrothermic—has its particular field and there are undoubtedly places in this country where the electrothermic process could be profitably applied.

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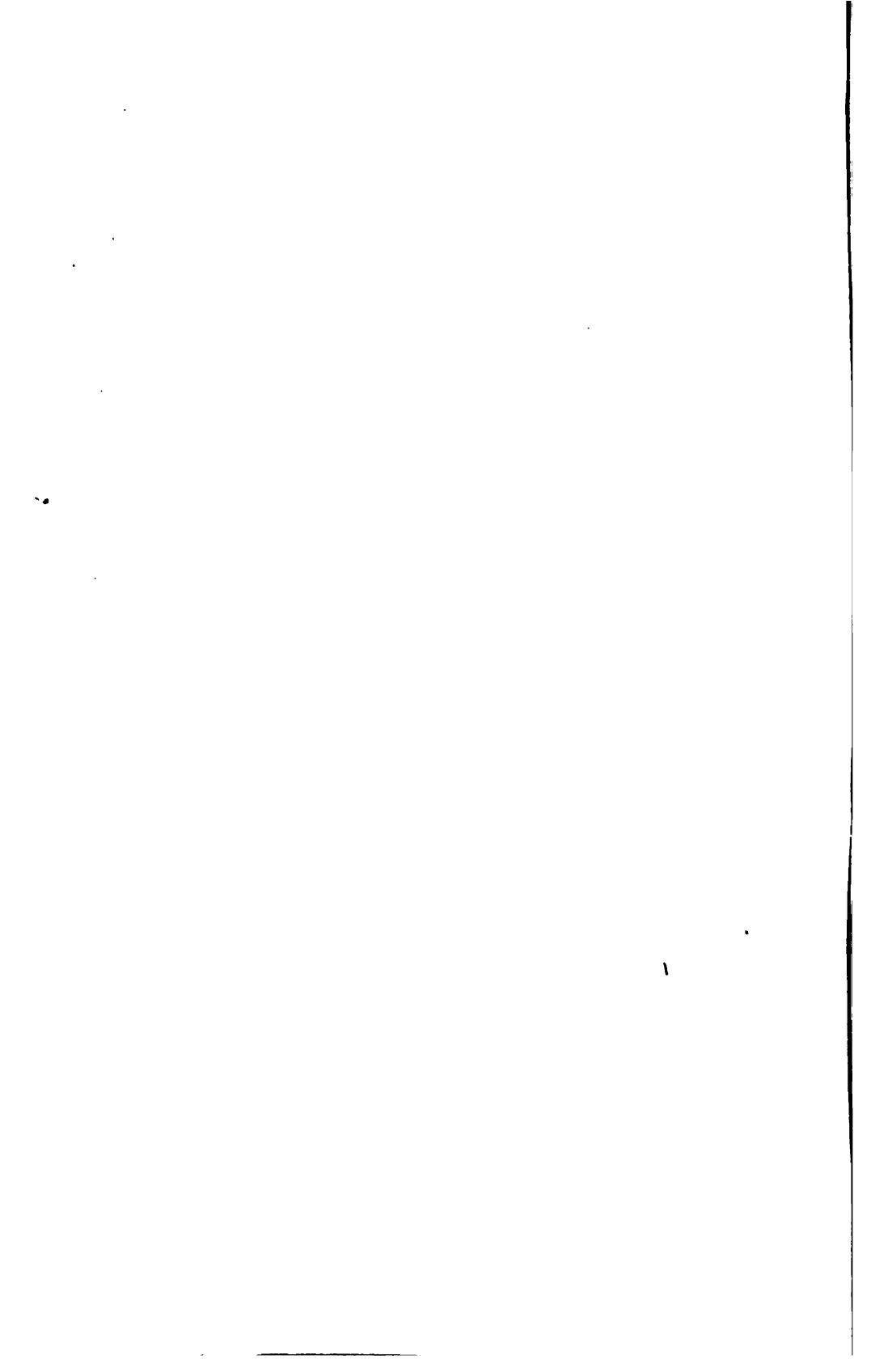
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FUSIBILITY OF ASH FROM COALS OF THE UNITED STATES

BY

W. A. SELVIG and A. C. FIELDNER



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First Edition, June 1922.

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FUSIBILITY OF ASH FROM COALS OF THE UNITED STATES.

By W. A. SELVIG and A. C. FIELDNER.

INTRODUCTION.

Information concerning the fusibility of coal ash has become of considerable value to the consumer of coal, mainly in connection with the troublesome formation of clinker resulting from the melting of the ash constituents of the burning coal. The growing interest in such data has led the Bureau of Mines to make a general survey of the "fusing" or "softening" temperatures of the ash from coals of the United States. It is hoped that this information when used together with the large number of coal analyses published¹ by the bureau will assist the consumer of coal in comparing different coals, and in selecting the coal best adapted for his purpose.

ACKNOWLEDGMENTS.

The laboratory fusibility tests were made by A. E. Hall, V. C. Allison, W. A. Selvig, C. R. Locke, C. S. Purcell, W. C. Ratliff, O. C. Brown, W. F. Holbrook, and A. M. Ondrejco, all of the chemical laboratory of the Bureau of Mines.

SOURCE AND COMPOSITION OF COAL ASH.

Coal ash is the incombustible residue remaining after the complete combustion of coal; it is derived from the inorganic mineral constituents of the coal. The ash-forming constituents are (1) inherent or intrinsic impurities that are present in an intimate mixture with the coal substance, and are derived either from the original material or from external sources such as sedimentation and precipitation while the coal-forming plant remains accumulated; (2) impurities, formed either during the laying down of the coal bed or subsequently, that occur in the form of partings, veins, and nodules of clay, shale, "slate," pyrite, and calcite; and (3) impurities that become intimately mixed with the coal in the process of mining, such as fragments of roof and floor.

¹ Lord, N. W., and others, *Analyses of coals in the United States, with descriptions of mine and field samples collected between July 1, 1904, and June 30, 1910*: Bull. 22, Bureau of Mines, 1913, 1,200 pp. (in two parts).

Fieldner, A. C., and others, *Analyses of mine and car samples collected in the fiscal years 1911 to 1913*: Bull. 85, Bureau of Mines, 1914, 444 pp.

Fieldner, A. C., and others, *Analyses of mine and car samples collected in the fiscal years 1913 to 1916*: Bull. 123, Bureau of Mines, 1917, 456 pp.

Fieldner, A. C., and others, *Analyses of mine and car samples collected in the fiscal years 1916 to 1919*: Bull. 193, Bureau of Mines (in press).

Coal ash is composed largely of compounds of silica, alumina, lime, and iron, with smaller quantities of magnesia, titanium, and alkali compounds. The chemical composition varies so widely that no typical composition can be given. In general the analyses of most coal ash will probably come between the following limits:

Typical limits of ash analyses.

| Constituent. | Per cent. | Constituent. | Per cent. |
|--------------------------------------|-----------|---|-----------|
| SiO ₂ | 40-60 | CaO..... | 1-15 |
| Al ₂ O ₃ | 20-35 | MgO..... | .5-4 |
| Fe ₂ O ₃ | 5-25 | Na ₂ O + K ₂ O..... | 1-4 |

This list of constituents shows that coal ash contains relatively large proportions of SiO₂ and Al₂O₃, and smaller proportions of the other oxides.

RELATION BETWEEN CHEMICAL COMPOSITION AND FUSIBILITY OF COAL ASH.

The fusibility of coal ash depends on several factors, such as the ratio of the silica to the bases present, the particular bases, and the percentage of alumina present. Mixtures extremely high in silica or extremely high in bases are not readily fusible. Ash that is low in iron is usually so highly siliceous that it is not readily fusible. Ash from coals high in pyrite is necessarily high in iron, and the ratio between the bases and silica is often such that easily fusible compounds may be formed. As a rule, coals containing considerable sulphur in the form of pyrite are apt to give trouble from clinker formation. Under conditions of the fuel bed the iron of the pyrite is apt to be converted to ferrous oxide, which with the silica present forms ferrous silicates that fuse at comparatively low temperatures. The chemical analyses of a series of five coal ashes ranging from very fusible (softening at 2,060° F.) to highly refractory (softening above 2,900° F.) is as follows:

Chemical analyses of five coal ashes covering a wide range of fusibility.

| Sample No. ^a | Softening temperature, ° F. | Analyses of ash, percentage of— | | | | | | | | |
|-------------------------|-----------------------------|---------------------------------|--------------------------------|--------------------------------|------------------|------|-----|-------------------|------------------|-----------------|
| | | SiO ₂ | Al ₂ O ₃ | Fe ₂ O ₃ | TiO ₂ | CaO | MgO | Na ₂ O | K ₂ O | SO ₂ |
| 1..... | 2,060 | 30.7 | 19.6 | 18.9 | 1.1 | 11.3 | 3.7 | 1.9 | 0.5 | 12.2 |
| 2..... | 2,320 | 46.2 | 22.9 | 7.7 | 1.0 | 10.1 | 1.6 | .7 | .8 | 8.9 |
| 3..... | 2,500 | 49.7 | 26.8 | 11.4 | 1.2 | 4.2 | .8 | 1.6 | 1.3 | 2.5 |
| 4..... | 2,730 | 51.0 | 30.9 | 10.7 | 1.9 | 2.1 | .9 | 1.0 | .4 | .6 |
| 5..... | +2,900 | 58.5 | 30.6 | 4.2 | 1.8 | 2.0 | .4 | .7 | .9 | .9 |

Sample No. 1, subbituminous coal, No. 3 bed, Montana; No. 2, bituminous coal, No. 6 bed, Illinois; No. 3, bituminous coal, Pittsburgh bed, Pennsylvania; No. 4, semibituminous coal, Pocahontas No. 3 bed, Virginia; No. 5, bituminous coal, Dean bed, Kentucky.

RELATION OF FUSIBILITY TESTS TO CLINKER FORMATION.

It is well to bear in mind that the conditions of fusibility tests in the laboratory may not be directly comparable to the conditions in the fuel bed of a furnace. In a laboratory test the ash-forming constituents are intimately mixed, whereas there is usually no such uniform distribution of the ash-forming constituents in coal fire.

Some coal ash is so infusible that little trouble is experienced from the formation of clinker. Ash that is slightly more fusible will often form a porous, spongy clinker which does not seriously obstruct the flow of air through the fuel bed and is easily removed. Coal ash with a low fusing temperature, say 2,100° F., not only melts in the average fire box, but is heated several hundred degrees above its melting temperature, becoming quite fluid and spreading out in a thin sheet over the grate, thereby obstructing the flow of air and localizing the heat in the fuel bed.

The method adopted for making fusibility tests was the result of considerable experimental work,¹ by the Bureau of Mines on the nature of the fusion of coal ash and the influence of various oxidizing, reducing, and neutral atmospheres on the softening temperature of ash molded in the form of Seger cones. The method takes into consideration the various factors that influence the result obtained, with special reference to the atmosphere surrounding the ash during the test. The atmosphere in which the ash is heated is controlled by burning an excess of gas; a reducing atmosphere is thus obtained by which the iron in the ash is reduced mainly to the ferrous state, giving the lowest temperature at which clinkering may result.

This change in the iron is of special importance in testing coals that contain a relatively large proportion of iron in the form of pyrite. Higher softening temperatures may be expected with such coals in tests using oxidizing atmospheres which would oxidize the iron mainly to ferric oxide, or in tests using strongly reducing atmospheres which would reduce the iron largely to the metallic state. Under both conditions a more refractory slag is formed than would result if the atmosphere were such as to reduce the iron in the ash to ferrous oxide, which, with the silica present, would form readily fusible silicates. Ferric oxide forms with silica compounds that require a high temperature for fusion; on the other hand, if the iron is reduced to the metallic state, one of the most active fluxing constituents is removed from the system and high-fusing points result.

Analyses of clinkers from boiler furnaces indicated that fuel-bed conditions favored the formation of clinkers in which the iron was chiefly in the ferrous state; consequently the values obtained in the

¹ Fieldner, A. C., Hall, A. E., and Feild, A. L., The fusibility of coal ash and the determination of the softening temperature: Bull. 129, Bureau of Mines, 1918, 146 pp.

laboratory tests are, in this respect, comparable to such conditions, and the tests give the lowest temperatures at which the intimately mixed ash will soften with the formation of clinker.

GAS FURNACE METHOD FOR DETERMINATION OF FUSIBILITY OF COAL ASH.

FURNACE.

The No. 3 melter's furnace of the American Gas Furnace Co. was used in the tests. This furnace is a type of pot furnace especially suitable for fusion determination, as the three burners arranged on a tangent near the base of the furnace produce a rotary flame that completely surrounds the crucible in which the cones are placed. The whirling flame heats the crucible uniformly, and when an excess of gas is used, a reducing atmosphere is maintained within the crucible, giving the lowest point at which the ash fuses. The air is supplied at an approximate pressure of 3 pounds per square inch, and the gas used may be either natural or artificial.

The furnace proper consists of three easily replaceable parts of fire clay, namely, a lower cylinder containing the three tangential tuyeres and forming the bottom of the furnace; a removable upper cylinder 7 inches in internal diameter and 7 inches high; and a cover plate $1\frac{1}{2}$ inches thick having a vent hole in the center for the flue gas.

The stock design was modified by providing the upper cylinder with two holes in the side; one was a 2-inch observation hole with its center 4 inches from the top of the cylinder (excluding the cover plate) and the other was a 1-inch thermocouple hole 90° to the right of the observation hole. The bottoms of these two holes were in the same horizontal plane. The interior of the furnace is cylindrical and approximately 7 inches in diameter and 11 inches high. A counterbalanced sheet-iron canopy is connected with a telescopic 8-inch flue to an exhaust system for conducting the hot gases out of the room. Two melter's No. 3 furnaces and accessories are shown in Plate I.

The joint between the upper and lower cylinder is made fairly gas-tight by spreading on the top of the lower cylinder a thick paste of alundum cement to a depth of one-half inch and then firmly pressing the upper cylinder into place. The outer part of the joint is also smoothed over with alundum cement. The fire-clay cylinders and cover will withstand a temperature of about $1,650^\circ\text{C}$.

REFRACTORY CRUCIBLES.

The interior of the furnace as arranged for making a test is shown in figure 1. The ash cones *a* are supported on a plate of alundum cement serving as a cover for the crucible, *b*, which is a Denver fire-

clay hard-burned crucible, No. E, 4 inches high and 3 inches in diameter at the top. An alundum tube of the proper height may be used in place of crucible *b*. Crucible *d* is a Denver fire-clay hard-burned crucible, No. K, $7\frac{1}{4}$ inches high and $4\frac{1}{2}$ inches in diameter at the top; it has observation and thermocouple holes corresponding to those in the furnace cylinder, and it rests on a fire-clay support. The thermo element is protected by a glazed Marquadt porcelain tube, *e*, one-fourth inch in diameter. A fused silica tube, $1\frac{1}{4}$ inches in external diameter and 6 inches long, is placed in the observation hole of the furnace, the inner end being flush with the inside furnace wall and the other end projecting from the furnace. This crucible should be provided with two holes in the side, an observation hole 2 inches in diameter with its center 2 inches from the top of the crucible and a thermocouple hole 1 inch in diameter, 90° to the right of the observation hole. The bottom of this hole should be in the same horizontal plane as the bottom of the observation hole. The hard-burned fire-clay crucibles will not withstand a temperature exceeding $1,500^\circ\text{C}$. The average life of a fire-clay crucible under these conditions is about six runs.

Corundite crucibles and covers made by the Massillon Stone & Fire Brick Co. are longer lived than the fire-clay crucibles, being

good for about 50 runs to temperatures not exceeding $1,500^\circ\text{C}$. A special form used by the Bureau of Mines is 3 inches inside diameter, $4\frac{1}{2}$ inches high outside, with a wall about one-fourth inch thick. The crucibles should be provided with two holes in the side—an observation hole 2 inches in diameter with its center $1\frac{1}{4}$ inches above the bottom of the crucible, and a thermocouple hole 1 inch in diameter, 90° to the right of the observation hole. The bottom of this hole should be in the same horizontal plane as the bottom of the observation hole.

Recent tests with Zirkite crucibles of the Foote Mineral Co. show that they are longer lived than the fire-clay crucibles. A stock Zirkite crucible was used for 70 runs before failure, the temperatures being carried up to $1,500^\circ\text{C}$.

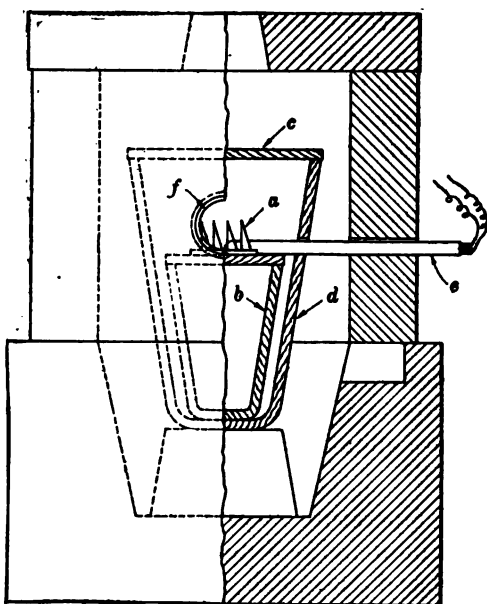


FIGURE 1.—Section of No. 3 melter's furnace arranged for fusion tests.

OBSERVATION TUBE.

A fused silica, refractory porcelain, or alundum tube, $1\frac{1}{4}$ inches in external diameter and 6 inches long, is placed in the 2-inch observation hole of the furnace, the inner end being flush with the inside furnace wall and the other end projecting out of the furnace. A brass sleeve carrying a thin clear glass window is slipped on the outer end of the observation tube to prevent the escape of burning gas, which would hinder convenient observation of the cones. When observations at high temperatures are made the cones are viewed through a piece of colored glass, such as Corning No. 6 shade, Noviweld.

BLOWING TUBE.

At furnace temperatures above $1,000^{\circ}$ C. observation of the ash cones is difficult. A Marquadt, Usalite, Impervite, or equal quality porcelain blowing tube one-fourth inch in diameter is inserted through the same hole as the thermocouple tube. This tube has small perforations along one side by which air may be forced upon the cones at the time of making the observation, thus momentarily cooling the ash cones and rendering them visible. The blowing tube is connected to the compressed-air line by means of rubber tubing, and the air is let into the tube at the time of observation by means of a pinchcock.

CONE MOLD.

Figure 2 shows a brass cone mold for making ash cones, three-fourths inch high and one-fourth inch at each side of the base, which is an equilateral triangle.

PYROMETER.

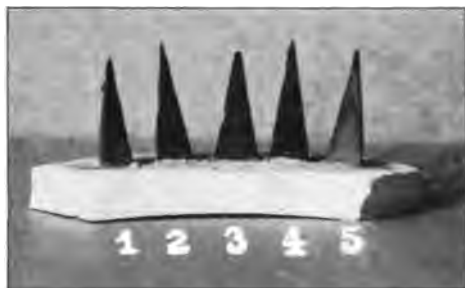
Temperature measurements may be made with a thermocouple of platinum and platinum-rhodium, protected from the furnace gases by a glazed Marquadt, Usalite, Impervite, or equal quality porcelain tube and a high resistance millivoltmeter; or with an optical pyrometer preferably of the Leeds & Northrup or Wanner type. This equipment should be checked frequently by mounting pieces of pure gold and nickel in the same manner as the cones. With a strong reducing atmosphere $1,452^{\circ}$ C. should be obtained for the melting point of nickel and $1,063^{\circ}$ C. for the gold. The pyrometer equipment should also be standardized from time to time through the temperature range for which it is used by a suitably equipped standardizing laboratory, such as that of the United States Bureau of Standards.

PREPARATION OF COAL ASH.

Spread out 50 to 100 grams of 60-mesh coal on a 5-inch fire-clay roasting dish, and completely convert it to ash in a muffle furnace at a temperature of 800° to 900° C. Transfer 5 to 10 grams of this ash



NO. 3 MELTER'S FURNACES AND ACCESSORIES.



A. CONES MOUNTED ON BASE.



B. TYPICAL FORM OF CONES FUSED IN NO. 3
MELTER'S FURNACE.

to an agate mortar and grind to a fineness of 200-mesh. A mechanical agate-mortar grinder will save time where many determinations are made. Place the finely ground ash in a silica or porcelain capsule, about five-eighths of an inch deep and $1\frac{1}{4}$ inches in diameter, and ignite for two hours in a current of oxygen, at a temperature of 800° to 850° C. The purpose of this ignition is to insure complete and uniform oxidation of the ash.

PREPARATION OF THE CONES.

Moisten the ignited ash with a 10 per cent dextrin solution, containing a small amount of salicylic acid as a preservative, and work into a plastic mass with a spatula. Mold the plastic material into small triangular pyramids three-fourths of an inch high and one-fourth inch wide at each side of the base. The pyramids are made by firmly pressing the plastic material with a steel spatula into a brass mold of the dimensions mentioned, the mold being similar to that shown in figure 2. Strike off the surface smooth and remove the cone from the mold by applying a small knife blade at the base. Mount the cones when dry in a refractory base composed of a mixture of equal parts of kaolin and calcined alumina. Moisten the base mixture to make it workable, and spread a part of it out on a sheet-iron plate. Then mount the cone in a vertical position in a small hole made in the base, and put a little of the base material into the hole around the bottom of the cone to fill the crevices and to make the cone stand firmly. Usually five cones are mounted on one base in the manner shown in Plate II, A. Dry the sheet-iron plate with the test piece on a hot plate, then ignite the cones at a dull red heat for 30 minutes in an open muffle to remove carbonaceous material.

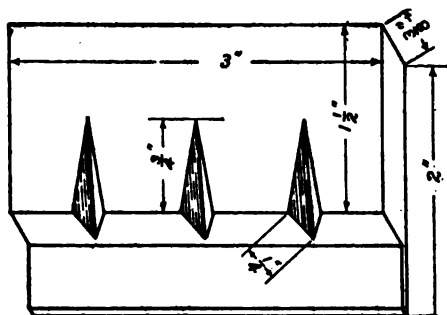


FIGURE 2.—Brass cone mold for making ash cones.

METHOD OF HEATING.

Put the test piece in the muffle furnace in the position shown in figure 1, place the loosely fitting cover *c* on the crucible, and ignite the gas. It is necessary to let the gas burn about 10 minutes to heat the furnace parts before the large cover plate of the furnace is replaced; otherwise the flame is apt to blow out. After the cover plate of the furnace has been put in position, the flow of gas and air is increased enough to cause combustion to take place just above the tuyeres and yet maintain a yellowish flame at least 6 inches above the opening in the furnace cover plate. While such a flame is main-

tained above the furnace, gradually increase the temperature by a suitable adjustment of gas and air to $800^{\circ}\text{C}.$, then reduce the rate

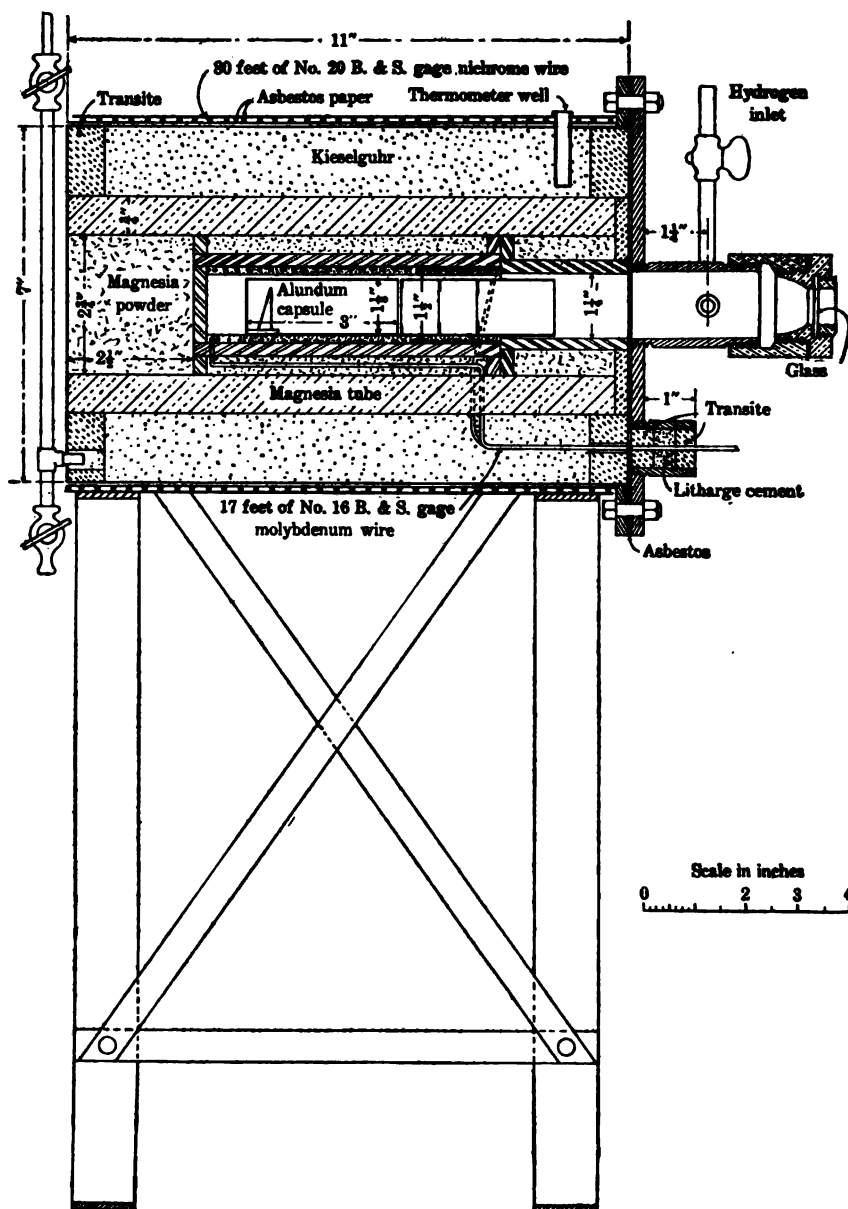


FIGURE 3.—Vertical longitudinal section through molybdenum furnace.

of heat increase to not less than $5^{\circ}\text{C}.$ and not more than $10^{\circ}\text{C}.$ per minute. Maintain this rate until the end of the test.

It is important that the 6-inch reducing flame be maintained at the furnace vent throughout the test, if possible, and at all events up to a temperature of $1,450^{\circ}\text{C}$. Temperatures above $1,450^{\circ}\text{C}$. require larger proportions of air to gas; however, a strongly reducing atmosphere is not so essential at the higher temperatures, because refractory ashes, on account of their low iron-oxide content, are only slightly affected by oxidizing or reducing atmospheres. After a test has been completed, turn the supply of gas and air off gradually to avoid cracking the muffle crucible.

The softening temperature is defined as the temperature at which the cone has fused down to a spherical lump, as shown in cones 2 and 3 of Plate II, *B*. Cone 4 has almost reached the softening temperature.

Critical points other than the softening temperature may be observed during the test and may be of value. They are as follows:

The initial deformation temperature—the temperature at which the first rounding or bending of the apex of the cone takes place, as shown in cone 1 of Plate II, *B*. Such bending must not be confused with a shrinking or warping of the cone.

The fluid temperature—the temperature at which the cone has spread over the base in a flat layer, as represented by cone 5 of Plate II, *B*.

The permissible differences of the softening temperature point in duplicate determinations are as follows: Same analyst, 30°C .; different analysts, 50°C .

MOLYBDENUM-WIRE RESISTANCE FURNACE.

Owing to the short life of the refractory crucibles used when heated to temperatures higher than $1,500^{\circ}\text{C}$., ashes that did not fuse at $1,500^{\circ}\text{C}$. were heated in a molybdenum-wire resistance furnace³ in an atmosphere of hydrogen gas to a temperature of $1,650^{\circ}\text{C}$. A vertical longitudinal section of the molybdenum-wire resistance furnace is shown in figure 3. Hydrogen gas is used in this type of furnace to prevent oxidation of the molybdenum resistance wire. Ashes softening above $1,500^{\circ}\text{C}$. have as a rule such a low iron content that the nature of the atmosphere surrounding the ash cones during the test has little influence on the fusibility.

INTERPRETATION OF THE FUSIBILITY TABLE.

With the exception of a few car samples of coal, the samples listed are all standard mine samples collected by representatives of the Bureau of Mines, the United States Geological Survey, or by various State geological surveys, according to methods used by the Bureau of

³ Fieldner, A. C., Hall, A. E., and Feld, A. L., The fusibility of coal ash and the determination of the softening temperature: Bureau of Mines, Bull. 120, 1918, pp. 74-79.

Mines.⁴ The car samples were collected by representatives of the Bureau of Mines according to standard methods.⁵ Car samples are designated as such in the table under the name of the mine.

The arrangement of the samples in Table I is alphabetical according to State, county, town, mine, and coal bed. The laboratory numbers of the samples collected from each mine are given; by means of these numbers the chemical analysis and the description of the samples can be obtained by reference to bulletins of the Bureau of Mines. (See footnote, p. 1.)

In classifying the coals according to rank the usage of the United States Geological Survey⁶ has been followed. In order to attain brevity, this rank is abbreviated. Thus, A signifies anthracite; S. A., semianthracite; S. B., semibituminous; B, bituminous; Sb. B., subbituminous; L, lignite; and N. C., natural coke.

The number of samples, the lowest, highest, and average softening temperatures in degrees Fahrenheit, and the percentage of ash and sulphur in the dry coal are tabulated for each mine. Samples remaining unfused at 3,010° F., which was the highest temperature attained in the tests, are marked by plus 3,010 (+3,010) and used as such in figuring the average values for the mine.

DISCUSSION OF FUSIBILITY VALUES.

CLASSES OF FUSIBILITY.

In general, the softening temperature of coal ash from the coals of the United States ranges from 1,900° to 3,100° F. For convenience in discussion, the order of fusibility of ash may be expressed by subdividing this range of softening temperature into three groups, as follows:

Class 1, refractory ashes, softening above 2,600° F.

Class 2, ashes of medium fusibility, softening between 2,200° and 2,600° F.

Class 3, easily fusible ashes, softening below 2,200° F.

EASTERN COALS.

The coals of Ohio which were tested gave ash of medium fusibility, mainly in class 2.

The bituminous and semibituminous coals of Pennsylvania that were tested gave refractory class 1 ash and medium fusible class 2 ash, chiefly the latter. Only a small number of coals from the Pennsylvania anthracite region were tested; these gave refractory ash in class 1.

⁴ Holmes, J. A., *The sampling of coal in the mine*; Tech. Paper 1, Bureau of Mines, 1911, 18 pp.

⁵ Pope, G. S., *Directions for sampling coal for shipment or delivery*; Tech. Paper 133, Bureau of Mines, 1917, 15 pp.

⁶ Campbell, M. R., *The coal fields of the United States*; U. S. Geological Survey Prof. Paper 100-A, 1917, pp. 3-9.

The West Virginia coals represent all three classes of fusibility. The semibituminous coals of West Virginia come principally in class 1 and class 2. Some very refractory class 1 coals are found in the Beckley bed in Raleigh County.

Many of the coals of Maryland that were tested gave a refractory ash coming in class 1 or in the upper part of class 2.

The softening temperature of the ash from coals of eastern Kentucky and of Virginia cover a wide range of fusibility from readily fusible ash, class 3, to refractory ash, class 1.

The coals tested from Tennessee and Alabama come mainly in class 2, having ash of medium fusibility, but both class 1 and class 3 are represented.

COALS OF THE INTERIOR PROVINCE.

Coals from Illinois, Indiana, western Kentucky, Kansas, Missouri, Oklahoma, and Arkansas are represented in Table I. The softening temperature of the ash from these coals is uniformly low, practically all of the coals tested coming in class 3. The uniformly low softening temperature of the ash is unquestionably due to the rather general distribution of pyrite, calcite, and frequently gypsum in the coal beds. Usually the percentage of iron and lime in the ash is high, and in general mines producing coal lower in sulphur than the average for the bed also produced coal having ash of higher fusibility than the average for the bed.

As may be expected, samples from the same mine do not show ash of uniform fusibility because the composition of the ash-forming constituents included in the samples differ. This variation of fusibility of samples of coal from the same mine in the interior province is not large, averaging but 140° F. for the mines represented in the table. In only a few instances is the difference between the lowest and highest softening temperatures of a number of samples from the same mine more than 300° F. The majority of mines gave samples coming within the average difference of 140° F. between the lowest and highest values.

WESTERN COALS.

A great variety of coal is found in the West, ranging from lignite to anthracite. In the table are listed coals from Alaska, California, Colorado, Idaho, Montana, Nevada, New Mexico, North Dakota, Utah, Washington, and Wyoming.

Most of the samples of Alaska coal listed were taken from outcrops or prospects, and range from refractory class 1 ash to ash of medium fusibility of class 2.

The coals from Washington show all three classes of fusibility. The majority of the samples, however, come in class 1 and class 2.

These coals have a high ash content, but in many the ash is refractory. Oregon and California contain little coal.

The coals tested from Colorado and New Mexico range from refractory class 1 ash to easily fusible class 3 ash, the majority of the samples tested coming in class 1 and class 2.

Many of the coals from Utah showed ash of comparatively low fusibility, in the upper part of class 3; a considerable number of samples gave ash of medium fusibility, in class 2. A few samples gave refractory class 1 ash.

Idaho and Nevada are of no commercial importance as coal producers, for only a little coal is found in these States.

The lignites tested, from North Dakota, South Dakota, and Montana, have ash of rather low fusibility, the majority of the samples coming in class 3 or the lower part of class 2.

The subbituminous coals tested from Montana and Wyoming also gave ash of rather low fusibility, most of the samples coming in class 3 and the lower part of class 2.

Most of the lignites and subbituminous coals listed in the table have a fairly low sulphur content, consequently the low melting points of the ash from these fuels can not be attributed to the presence of large amounts of iron in the form of pyrite, but rather to the presence of other mineral constituents in the proper proportion to form readily fusible mixtures.

FUSIBILITY OF COAL ASH FROM MINE AND FROM CAR SAMPLES.

In collecting mine samples of coal the Bureau of Mines endeavors to have the samples represent as nearly as possible the coal that is produced commercially; car samples as a rule, however, have a higher ash content than mine samples from the same mine because of impurities that become mixed with the coal in mining. Theoretically the extraneous impurities that are included in the coal as mined may either lower or raise the fusibility of the coal ash. If these impurities consist of substances such as pyrite, the ash of the coal mined would probably be more fusible than the ash of the mine samples; on the other hand, an addition of shale may make the ash more refractory, especially in coals having ash of rather low fusibility.

Table II lists all mines from which both mine and car samples have been collected, and gives the ash fusibility of mine and car samples from the same mine. A study of this table is of interest in connection with the variation of ash fusibility of mine and car samples. If the difference between the softening temperatures of coal ash from mine and car samples does not exceed 100° F., it may be considered that the degree of fusibility is substantially the same. Data in Table II have been summarized below to show the number of mines from

which the fusibility of ash of mine and car samples were substantially the same; that is, within 100° F. The number of mines from which the ash of the car samples was more refractory and the number of mines from which the ash from the mine samples was more refractory is as follows:

Summary of Table II comparing fusibility of ash from mine and car samples.

| State. | Number of mines from which ash of car and mine samples was substantially of the same fusibility—within 100° F. | Number of mines from which ash of car samples was more refractory. | Number of mines from which ash of mine samples was more refractory. |
|---------------------|--|--|---|
| Illinois | | 1 | 2 |
| Maryland | 1 | | |
| Pennsylvania | 2 | 1 | 3 |
| Virginia | | | 1 |
| West Virginia | 32 | 9 | 15 |

| | | | | | | | | |
|-----------------------|------------------------|---------------------------------|----------------|---|---|--------|-------|------|
| Kings River..... | Outcrop..... | No. 1..... | B. 1837..... | 1 | 1 | 2,310 | 8.73 | .58 |
| Do..... | do..... | do..... | 1839..... | 1 | 1 | 2,610 | 12.76 | .20 |
| Do..... | do..... | do..... | 1836..... | 1 | 1 | 2,620 | 10.47 | .54 |
| Do..... | do..... | do..... | 1851..... | 1 | 1 | 2,680 | 20.72 | .31 |
| Matanuska River..... | do..... | No. 2..... | 1814..... | 1 | 1 | 2,300 | 9.37 | .36 |
| Moose Creek..... | do..... | Unnamed..... | 28837..... | 1 | 1 | 2,450 | 5.66 | .26 |
| Young Creek..... | do..... | do..... | 11382..... | 1 | 1 | | | |
| NENANA FIELD. | | | | | | | | |
| California Creek..... | Outcrop..... | Unnamed..... | 28359..... | 1 | 1 | 2,350 | 25.56 | .58 |
| Do..... | do..... | do..... | 28360-61..... | 2 | 2 | 2,370 | 34.71 | .27 |
| Healy Creek..... | do..... | do..... | 17796..... | 1 | 1 | 2,450 | 6.10 | .22 |
| Do..... | do..... | do..... | 28368..... | 1 | 1 | 2,400 | 7.53 | .32 |
| Igloo Creek..... | do..... | do..... | 17794-95..... | 2 | 2 | 2,520 | 4.78 | .14 |
| Lagunita Creek..... | do..... | do..... | 28362..... | 1 | 1 | 2,340 | 13.30 | .24 |
| Do..... | do..... | do..... | 28368..... | 1 | 1 | 2,150 | 9.93 | .21 |
| Do..... | do..... | No. 5..... | 28363..... | 1 | 1 | 2,450 | 7.18 | .22 |
| Do..... | do..... | do..... | 28364..... | 1 | 1 | 2,370 | 16.42 | .26 |
| Do..... | do..... | do..... | 28365..... | 1 | 1 | 2,450 | 5.52 | .18 |
| Do..... | do..... | do..... | 28366..... | 1 | 1 | 2,210 | 11.07 | .25 |
| Do..... | do..... | do..... | 28367..... | 1 | 1 | 2,390 | 8.01 | .35 |
| Do..... | do..... | do..... | 28369..... | 1 | 1 | 2,300 | 15.43 | .54 |
| Nenana River..... | do..... | do..... | 23042..... | 1 | 1 | 2,370 | 5.07 | .21 |
| Tatlanika River..... | do..... | do..... | 17797..... | 1 | 1 | 2,670 | 22.55 | .32 |
| YAKATAGA FIELD. | | | | | | | | |
| Ducktooth Valley..... | Outcrop..... | Unnamed..... | 18246..... | 1 | 1 | +3,000 | 23.69 | .67 |
| ARKANSAS. | | | | | | | | |
| FRANKLIN COUNTY. | Denning No. 2..... | Denning (Hartshorne?)..... | 18746-48..... | 3 | 3 | 2,200 | 7.38 | 2.45 |
| LOUIS COUNTY. | Grand No. 1..... | Paris..... | 18750-52..... | 3 | 3 | 2,140 | 10.12 | 3.28 |
| FOUR COUNTY. | Bernice..... | Shinn Basin..... | 18755, 57..... | 2 | 2 | 2,280 | 10.36 | 2.23 |
| FOUR COUNTY. | Branner No. 2..... | Hartshorne..... | 18062-63..... | 2 | 2 | 2,320 | 17.30 | 1.59 |
| FOUR COUNTY. | Central No. 10..... | do..... | 28638..... | 1 | 1 | 2,080 | 11.27 | 1.15 |
| FOUR COUNTY. | do..... | do..... | 28635..... | 1 | 1 | 2,000 | 10.07 | .83 |
| FOUR COUNTY. | Central No. 6..... | Upper and Lower Hartshorne..... | 27013-16..... | 4 | 4 | 2,080 | 9.60 | 2.01 |
| FOUR COUNTY. | Jenny Lind No. 17..... | Hartshorne..... | 18267..... | 1 | 1 | 2,110 | 9.70 | 1.43 |

ALASKA.

15

| SHELBY COUNTY. | | | | | | | | | |
|-----------------------|----------------------|--------------------|--------|---------------------|---|-------|-------|-------|-------|
| Aldrich..... | Aldrich..... | Montevillo..... | B. | 26720-33..... | 4 | 2,000 | 2,470 | 2,320 | 7.24 |
| Glen Carbon..... | Glen Carbon..... | Cholson..... | B. | 10602-04, 9667..... | 4 | 2,150 | 2,410 | 2,240 | 4.08 |
| Helena..... | Helena..... | Helena..... | B. | 13288-90..... | 3 | 2,230 | 2,500 | 2,360 | 7.80 |
| Do..... | Do..... | Do..... | B. | 25543-45..... | 3 | 2,400 | 2,610 | 2,470 | 10.02 |
| Eureka..... | Eureka..... | Maylene..... | B. | 9410..... | 1 | | | 2,380 | 8.92 |
| Chumax..... | Chumax..... | Maylene..... | B. | 26301-04..... | 4 | 2,180 | 2,470 | 2,320 | 4.42 |
| Do..... | Do..... | Do..... | B. | 14391-92..... | 2 | 2,200 | 2,490 | 2,340 | 4.48 |
| Straven..... | Straven..... | Upper Straven..... | B. | | 2 | | | | 7.45 |
| TUSCALOOSA COUNTY. | | | | | | | | | |
| Abernant..... | Abernant..... | Jagger..... | B. | 14717..... | 1 | | | 2,030 | 9.33 |
| Rock Castle..... | Rock Castle..... | Do..... | B. | 18510-12..... | 3 | 2,450 | 2,640 | 2,550 | 8.90 |
| WALKER COUNTY. | | | | | | | | | |
| Carbon Hill..... | Galloway No. 11..... | Jagger..... | B. | 24954-57..... | 4 | 2,300 | 2,910 | 2,600 | 11.19 |
| Dora..... | Sipesy..... | Black Creek..... | B. | 18404-07..... | 4 | 2,330 | 2,580 | 2,420 | 3.25 |
| Parrish..... | Parrish..... | Mary Lee..... | B. | 77605-08..... | 4 | 2,530 | 2,740 | 2,670 | 14.13 |
| ALASKA. | | | | | | | | | |
| BERING RIVER. | | | | | | | | | |
| Wardall Ridge..... | Outcrop..... | Unnamed..... | S. B. | 22022..... | 1 | | | 2,520 | 7.27 |
| Do..... | Do..... | Do..... | S. B. | 22023..... | 1 | | | 2,410 | 1.27 |
| FAIRHAVEN DISTRICT. | | | | | | | | | |
| Candle..... | Kugruk..... | Unnamed..... | Sb. B. | 19928..... | 1 | | | 2,410 | 6.44 |
| IDITAROD DISTRICT. | | | | | | | | | |
| Iditarod..... | Prospect..... | Unnamed..... | A. | 19347..... | 1 | | | 2,380 | 7.35 |
| INUOKO DISTRICT. | | | | | | | | | |
| Trailway..... | Outcrop..... | Unnamed..... | A. | 26370..... | 1 | | | 2,710 | 5.27 |
| KENAI PENINSULA. | | | | | | | | | |
| Port Graham..... | Outcrop..... | Unnamed..... | Sb. B. | 17489..... | 1 | | | 2,490 | 15.80 |
| MATANUSKA FIELD. | | | | | | | | | |
| Anthracite Ridge..... | Outcrop..... | Unnamed..... | S. A. | 12745..... | 1 | | | 2,250 | 26.23 |
| Do..... | Do..... | Do..... | B. | 17792..... | 1 | | | 2,960 | 8.02 |
| Boulder Creek..... | Do..... | Do..... | B. | 19346..... | 1 | | | 2,910 | 9.14 |

^a The abbreviations for the rank of coal are as follows: A., anthracite; S. A., semianthracite; S. B., semibituminous; B., bituminous; Sb. B., subbituminous; L., lignite; and N. C., natural coke.

TABLE I.—Softening temperatures of coal ash from coals of the United States—Continued.

| Locality. | Mine. | Coal bed. | Rank of coal. | Laboratory No. | Number of samples from mine. | Softening temperature ° F. | | | Average analysis of dry coal, per-centage of— | |
|----------------------------|---------------------------------|--------------|---------------|----------------|------------------------------|----------------------------|----------|----------|---|----------|
| | | | | | | Lowest. | Highest. | Average. | Ash. | Sulphur. |
| 1 | | | | | | | | | | |
| ALASKA—Contd. | | | | | | | | | | |
| MATANUSKA FIELD—continued. | | | | | | | | | | |
| Chickaloon River. | U. S. N. A. C. I. E., tunnel B. | Lower No. 5. | S. B. | 18148. | 1 | | | 3,000 | 9.55 | 0.53 |
| Do. | do. | No. 54. | S. B. | 18141. | 1 | | | 2,040 | 8.14 | 0.62 |
| Do. | do. | No. 6. | S. B. | 18143. | 1 | | | 2,270 | 3.07 | 0.69 |
| Do. | do. | No. 9. | S. B. | 18150. | 1 | | | 2,000 | 20.54 | 0.55 |
| Do. | U. S. N. A. C. I. E., tunnel D. | D. | S. B. | 17708. | 1 | | | 2,980 | 12.23 | 0.54 |
| Do. | do. | E. | S. B. | 19046. | 1 | | | 2,260 | 11.13 | 1.51 |
| Do. | U. S. N. A. C. I. E., tunnel 1. | No. 8. | S. B. | 18065, 101. | 2 | 2,420 | 2,880 | 2,650 | 10.19 | 0.64 |
| Do. | Chickaloon. | No. 8. | S. B. | 31226-37. | 2 | 2,370 | 2,840 | 2,600 | 11.94 | 0.43 |
| Do. | U. S. N. A. C. I. E., tunnel 2. | Upper No. 5. | S. B. | 18153. | 1 | | | 2,980 | 7.89 | 0.57 |
| Do. | U. S. N. A. C. I. E., tunnel 3. | No. 3. | S. B. | 18066. | 1 | | | 2,640 | 5.78 | 0.69 |
| Do. | do. | No. 4. | S. B. | 18138. | 1 | | | 2,230 | 6.49 | 0.65 |
| Do. | U. S. N. A. C. I. E., tunnel 4. | D. | S. B. | 18100. | 1 | | | 2,380 | 11.03 | 0.51 |
| Coal Creek. | Outcrop. | C. | S. B. | 18321. | 1 | | | 2,490 | 3.47 | 0.33 |
| Do. | do. | No. 2. | S. B. | 19004. | 1 | | | 2,360 | 4.65 | 0.41 |
| Do. | do. | do. | S. B. | 19040. | 1 | | | 2,960 | 9.77 | 0.43 |
| Do. | do. | No. 3. | S. B. | 18103. | 1 | | | 2,380 | 10.31 | 0.36 |
| Do. | do. | do. | S. B. | 19041. | 1 | | | 2,510 | 5.93 | 0.40 |
| Do. | do. | No. 4. | S. B. | 18320. | 1 | | | 2,270 | 6.83 | 0.52 |
| Do. | do. | No. 5. | S. B. | 18322. | 1 | | | 3,000 | 9.88 | 0.40 |
| Do. | do. | No. 6. | S. B. | 18092. | 1 | | | 3,000 | 11.94 | 0.45 |
| Do. | do. | do. | S. B. | 18109. | 1 | | | 2,980 | 9.65 | 0.46 |
| Do. | U. S. N. A. C. I. E., tunnel A. | No. 8. | S. B. | 18104, 08, 39. | 3 | 2,290 | 2,520 | 2,390 | 8.62 | 0.47 |
| Do. | U. S. N. A. C. I. E., drift 1. | No. 1. | S. B. | 18107. | 1 | | | 2,950 | 9.21 | 0.52 |
| Do. | U. S. N. A. C. I. E., drift 2. | No. 4. | S. B. | 18096. | 1 | | | 2,820 | 13.44 | 0.63 |
| Emery. | Kelly. | Kelly. | S. B. | 26731-32. | 2 | 2,210 | 2,660 | 2,440 | 8.27 | 0.50 |
| Emery. | David. | David. | S. B. | 26732. | 1 | | | 2,280 | 5.79 | 0.35 |
| Emery. | Emery. | Emery. | S. B. | 26733. | 1 | | | 2,740 | 10.26 | 0.35 |
| Emery. | Emery. | Emery. | S. B. | 26734. | 1 | | | 2,660 | 18.39 | 0.39 |
| Do. | Emery. | Emery. | S. B. | 26735. | 1 | | | 2,180 | 7.63 | 0.46 |
| Do. | Emery. | Emery. | S. B. | 26736. | 1 | | | 3,010 | 9.96 | 0.57 |

| | | | | | | | | |
|-----------------------|---------------------------------|---------|----------------|---|-------|--------|-------|------|
| Kings River..... | No. 1..... | B. B. | 18137..... | 1 | | 2,570 | 11.18 | .51 |
| Do..... | do..... | do..... | 18319..... | 1 | | 2,310 | 8.73 | .58 |
| Do..... | do..... | N. C. | 18138..... | 1 | | 2,610 | 12.76 | .50 |
| Do..... | do..... | do..... | 18151..... | 1 | | 2,680 | 16.47 | .54 |
| Matanuska River..... | No. 2..... | B. | 18144..... | 1 | | 2,680 | 20.72 | .31 |
| Moose Creek..... | Unnamed..... | B. | 28337..... | 1 | | 2,300 | 9.37 | .36 |
| Young Creek..... | do..... | B. | 11382..... | 1 | | 2,450 | 5.06 | .26 |
| NENANA FIELD. | | | | | | | | |
| California Creek..... | Unnamed..... | L. | 26359..... | 1 | | 2,350 | 25.56 | .58 |
| Do..... | do..... | L. | 26360-61..... | 2 | 2,360 | 2,370 | 34.71 | .27 |
| Healy Creek..... | do..... | L. | 17786..... | 1 | | 2,450 | 6.10 | .22 |
| Do..... | do..... | L. | 26368..... | 1 | | 2,400 | 7.53 | .32 |
| Igloo Creek..... | do..... | L. | 17794-95..... | 2 | 2,560 | 2,520 | 4.78 | .14 |
| Lagunita Creek..... | do..... | L. | 26362..... | 1 | | 2,340 | 13.30 | .24 |
| Do..... | No. 5..... | L. | 26368..... | 1 | | 2,150 | 9.93 | .21 |
| Do..... | Unnamed..... | L. | 26363..... | 1 | | 2,450 | 7.18 | .22 |
| Do..... | do..... | L. | 26364..... | 1 | | 2,370 | 16.42 | .26 |
| Do..... | do..... | L. | 26365..... | 1 | | 2,450 | 5.52 | .18 |
| Do..... | do..... | L. | 26366..... | 1 | | 2,210 | 11.07 | .25 |
| Do..... | do..... | L. | 26367..... | 1 | | 2,390 | 8.01 | .35 |
| Do..... | do..... | L. | 26369..... | 1 | | 2,300 | 15.43 | .54 |
| Nenana River..... | do..... | L. | 23042..... | 1 | | 2,370 | 5.07 | .21 |
| Tatianka River..... | do..... | L. | 17797..... | 1 | | 2,670 | 22.55 | .32 |
| YAKATAGA FIELD. | | | | | | | | |
| Duktoth Valley..... | Unnamed..... | S. B. | 19345..... | 1 | | +3,000 | 23.69 | .67 |
| ARKANSAS. | | | | | | | | |
| FRANKLIN COUNTY. | | | | | | | | |
| Denning..... | Denning (Hartshorne?)..... | S. B. | 18746-48..... | 3 | 2,180 | 2,230 | 7.38 | 2.45 |
| LOGAN COUNTY. | | | | | | | | |
| Paris..... | Paris..... | S. B. | 18750-52..... | 3 | 2,130 | 2,160 | 10.12 | 3.28 |
| POPE COUNTY. | | | | | | | | |
| Russellville..... | Shinn Basin..... | S. A. | 18755, 57..... | 2 | 2,180 | 2,370 | 10.36 | 2.23 |
| SEBASTIAN COUNTY. | | | | | | | | |
| Hackett..... | Hartshorne..... | S. B. | 18002-63..... | 2 | 2,310 | 2,340 | 17.30 | 1.59 |
| Harford..... | do..... | S. B. | 29836..... | 1 | | 2,080 | 11.37 | 1.15 |
| Do..... | do..... | S. B. | 29835..... | 1 | | 2,000 | 10.07 | .83 |
| Huntington..... | Upper and Lower Hartshorne..... | S. B. | 27812-16..... | 4 | 2,060 | 2,080 | 9.60 | 2.01 |
| Jenny Lind..... | Hartshorne..... | S. B. | 18267..... | 1 | | 2,110 | 8.70 | 1.42 |

TABLE I.—Softening temperatures of coal ash from coals of the United States—Continued.

| Locality. | Mine. | Coal bed. | Rank of coal. | Laboratory No. | Number of samples from mine. | Softening temperature * F. | | | Average analysis of dry coal, per-centage of— | |
|-----------------------|--------------------|-----------------------|---------------|----------------------|------------------------------|----------------------------|----------|----------|---|----------|
| | | | | | | Lowest. | Highest. | Average. | Ash. | Sulphur. |
| 1 | | | | | | | | | | |
| CALIFORNIA. | | | | | 2 | 3 | 4 | 5 | 6 | 7 |
| MENDOCINO COUNTY. | | | | | | | | | | |
| Dos Rios..... | Outcrop..... | Unnamed..... | S. B. | 31139..... | 1 | | | 2,220 | 6.91 | 2.98 |
| MONTREY COUNTY. | | | | | 2 | 2,210 | 2,470 | 2,340 | 11.26 | 4.62 |
| Stone Canon..... | Stone Canon..... | Unnamed..... | B. | 31100-01..... | | | | | | |
| COLORADO. | | | | | | | | | | |
| ADAMS COUNTY. | | | | | 1 | | | 2,120 | 11.35 | .48 |
| Bennett..... | Thomas..... | Unnamed..... | Sb. B. | 13141..... | | | | | | |
| BOULDER COUNTY. | | | | | | | | | | |
| Broomfield..... | Monarch No. 2..... | Unnamed..... | Sb. B. | 31314-15..... | 2 | 1,980 | 2,080 | 2,010 | 6.54 | .33 |
| Lafayette..... | Simpson..... | Simpson..... | Sb. B. | 15165, 31391..... | 2 | 2,030 | 2,130 | 2,080 | 4.45 | .48 |
| Louisville..... | Acme..... | Lower Acme..... | Sb. B. | 31394..... | 1 | | | 2,040 | 5.46 | .40 |
| Superior..... | Industrial..... | Unnamed..... | Sb. B. | 32253-55..... | 3 | 2,060 | 2,140 | 2,100 | 6.89 | .37 |
| DELTA COUNTY. | | | | | | | | | | |
| Bowie..... | King..... | "King"..... | B. | 15212, 29920..... | 2 | 2,470 | 2,740 | 2,600 | 5.06 | .90 |
| Dominguez..... | Wells Gulch..... | Unnamed..... | B. | 11105..... | 1 | | | 2,160 | 6.20 | 1.01 |
| EL PASO COUNTY. | | | | | | | | | | |
| Calhan..... | Mcaby..... | Mcaby..... | Sb. B. | 10723..... | 1 | | | 2,520 | 20.77 | .45 |
| Colorado Springs..... | El Paso..... | Colorado Springs..... | Sb. B. | 29907-09..... | 3 | 2,190 | 2,260 | 2,230 | 7.99 | .43 |
| Pike View..... | do..... | Pike View..... | Sb. B. | 12099, 29911-15..... | 6 | 2,070 | 2,340 | 2,250 | 7.37 | .41 |
| Ramah..... | Purdon..... | Purdon..... | Sb. B. | 10741..... | 1 | | | 2,510 | 27.45 | .48 |

| FERMONT COUNTY. | | | | | | | | | |
|-------------------|--------------------------|---------|---------------------------|---|-------|-------|-------|-------|------|
| Chandler..... | Chandler..... | 8th. B. | 76376-79 | 1 | 2,080 | 2,080 | 2,080 | 7.34 | .82 |
| Do..... | do..... | 8th. B. | 12893-94 | 2 | 2,080 | 2,130 | 2,100 | 7.55 | .48 |
| Rockvale..... | Rockvale..... | 8th. B. | 11086-88 | 2 | 2,080 | 2,860 | 2,220 | 10.43 | .48 |
| Williamsburg..... | Magnet..... | B. | 10127-28 | 2 | 2,080 | 2,130 | 2,080 | 7.90 | .90 |
| GARFIELD COUNTY. | | | | | | | | | |
| New Oastle..... | Violen..... | B. | 12925-26, 420, 12865, 574 | 5 | 2,080 | 2,370 | 2,320 | 5.75 | .80 |
| South Canon..... | Wheeler..... | B. | 15165-67 | 2 | 2,380 | 2,410 | 2,380 | 8.78 | .58 |
| GUTENBERG COUNTY. | | | | | | | | | |
| Mount Carbon..... | Alpine..... | B. | 10082 | 1 | | | 2,310 | 7.90 | 1.05 |
| Do..... | Kinder..... | B. | 10081 | 1 | | | 2,330 | 8.51 | .46 |
| Somerset..... | Somerset..... | B. | 12522-23 | 2 | 2,260 | 2,260 | 2,260 | 9.74 | .46 |
| HUEFANO COUNTY. | | | | | | | | | |
| Camp Shumway..... | Gordon..... | B. | 32644-45 | 2 | 2,280 | 2,300 | 2,290 | 11.48 | .94 |
| Do..... | Vesta..... | B. | 32647 | 1 | | | 2,270 | 8.89 | .98 |
| Del Carbon..... | Breiman..... | B. | 32651-62 | 1 | 2,180 | 2,350 | 2,240 | 7.98 | .72 |
| Do..... | Turner..... | B. | 32651 | 1 | | | 2,240 | 7.98 | .71 |
| Do..... | Do..... | B. | 32650-49 | 9 | 2,330 | 2,430 | 2,250 | 11.49 | .66 |
| Farr..... | Caneron..... | B. | 34080-81 | 2 | 2,430 | 2,620 | 2,310 | 15.43 | .80 |
| Lester..... | Do..... | B. | 34108-04 | 2 | 2,400 | 2,640 | 2,330 | 11.08 | .54 |
| Maland..... | Do..... | B. | 32619 | 1 | | | 2,680 | 13.70 | .76 |
| Do..... | Do..... | B. | 32619 | 1 | | | 2,400 | 12.31 | .94 |
| McGure..... | Pinon..... | B. | 10189-90 | 2 | 2,330 | 2,340 | 2,340 | 10.80 | .78 |
| Oakdale..... | Unnamed..... | B. | 12423-28 | 3 | 2,180 | 2,660 | 2,360 | 10.60 | .57 |
| Do..... | Wannomoth..... | B. | 32461 | 1 | | | 2,330 | 8.19 | .72 |
| Picton..... | Walsen..... | B. | 32467 | 1 | | | 2,370 | 11.62 | .59 |
| Do..... | Walsen and Robinson..... | B. | 32468 | 1 | | | 2,210 | 10.48 | .59 |
| Ravenwood..... | Caneron..... | B. | 31493-97 | 2 | 2,500 | 2,520 | 2,510 | 8.38 | .77 |
| Strong..... | Sunnyside..... | B. | 32555-56 | 2 | 2,460 | 2,580 | 2,500 | 9.24 | .65 |
| Trigs..... | Robinson..... | B. | 32724-35 | 2 | 2,100 | 2,150 | 2,130 | 8.29 | .66 |
| Talbot..... | Caneron..... | B. | 32463 | 1 | | | 2,280 | 10.51 | 1.01 |
| Do..... | Do..... | B. | 32463 | 1 | | | 2,700 | 10.43 | .81 |
| Walsen..... | Robinson No. 1..... | B. | 32857-60 | 3 | 2,400 | 2,440 | 2,410 | 11.65 | .84 |
| Do..... | Robinson No. 2..... | B. | 32858-66 | 3 | 2,280 | 2,360 | 2,290 | 10.84 | .66 |
| Walsenbg..... | Mutual..... | B. | 32846 | 1 | | | 2,500 | 13.63 | .47 |
| JACKSON COUNTY. | | | | | | | | | |
| Oakmont..... | Bleach..... | 8th. B. | 12601 | 1 | | | 2,580 | 8.94 | .91 |
| Do..... | Mitchell..... | 8th. B. | 12659 | 1 | | | 2,270 | 12.56 | 1.05 |
| Walsen..... | Marr..... | 8th. B. | 12415 | 1 | | | 2,440 | 4.31 | .19 |
| Do..... | McCallum..... | 8th. B. | 12774 | 1 | | | 2,270 | 7.78 | .33 |
| Do..... | Sidduth..... | 8th. B. | 12414 | 1 | | | 2,440 | 6.26 | .74 |
| Do..... | Do..... | 8th. B. | 12775 | 1 | | | 2,190 | 14.50 | .98 |
| Do..... | Wincom..... | 8th. B. | 12775 | 1 | | | 2,190 | 14.50 | .98 |

TABLE I.—Softening temperatures of coal ash from coals of the United States—Continued.

| Locality. | Mine. | Coal bed. | Rank of coal. | Laboratory No. | Number of samples from mine. | Softening temperature ° F. | | | Average analysis of dry coal, percentage of— | |
|---------------------|--------------------|-----------------------|---------------|------------------------|------------------------------|----------------------------|----------|----------|--|----------|
| | | | | | | Lowest. | Highest. | Average. | Ash. | Sulphur. |
| 1 | | | | | | | | | | |
| COLORADO—Continued. | | | | | | | | | | |
| JEFFERSON COUNTY. | | | | | | | | | | |
| Morrison..... | White Ash..... | Jumbo..... | Sb. B. | 16615..... | 1 | | | 2, 260 | 8.12 | 1.05 |
| LA PLATA COUNTY. | | | | | | | | | | |
| Durango..... | Cinder Butte..... | B..... | Sb. B. | 17747..... | 1 | | | | | |
| Do..... | San Juan..... | B..... | B. | 14770-71..... | 2 | 2, 990 | 3, 010 | +3, 010 | 19.13 | .68 |
| Do..... | Soda Springs..... | B..... | B. | 17748..... | 1 | | | 3, 000 | 7.05 | .74 |
| Hesperus..... | Hesperus..... | B..... | B. | 14773-74..... | 2 | 2, 710 | 3, 000 | 2, 880 | 15.74 | .72 |
| Do..... | Mormon..... | Unamed..... | B. | 17745..... | 1 | | | | 6.37 | .71 |
| Do..... | Wheeler..... | "Upper No. 8"..... | B. | 17855..... | 1 | | | | 14.18 | .90 |
| Manco..... | Hawert..... | Speaker..... | B. | 20268..... | 1 | | | | 7.12 | .62 |
| Paria..... | Paria Peak..... | Peacock..... | B. | 14776-78..... | 3 | 2, 080 | 2, 540 | 2, 320 | 9.57 | .69 |
| LAS ANIMAS COUNTY. | | | | | | | | | | |
| Agular..... | Empire..... | Walsen..... | B. | 32881..... | 1 | | | 2, 360 | 13.39 | .69 |
| Do..... | Royal..... | Peerless..... | B. | 32882..... | 1 | | | 2, 780 | 11.88 | .73 |
| Do..... | do..... | Walsen..... | B. | 32883..... | 1 | | | 2, 420 | 11.16 | .69 |
| Broadhead..... | Temple No. 9..... | Broadhead No. 4..... | B. | 32883..... | 1 | | | 2, 320 | 8.20 | .45 |
| Do..... | Temple No. 10..... | Rugby..... | B. | 32883..... | 1 | | | 2, 430 | 11.13 | .46 |
| Cokedale..... | Bon Carbo..... | Primero..... | B. | 31441..... | 1 | | | 2, 300 | 14.17 | .50 |
| Do..... | Cokedale..... | Cokedale..... | B. | 11768-70..... | 3 | 2, 710 | 2, 990 | 2, 860 | 17.64 | .52 |
| Delagua..... | Ces..... | Delagua..... | B. | 31423..... | 1 | | | 2, 480 | 8.65 | .75 |
| Do..... | Do..... | do..... | B. | 11435-37 26351..... | 4 | 2, 310 | 2, 480 | 2, 480 | 11.51 | .56 |
| Do..... | Do..... | do..... | B. | 31423-25..... | 2 | 2, 550 | 2, 610 | 2, 300 | 7.51 | .60 |
| Do..... | Do..... | do..... | B. | 31427..... | 1 | | | 2, 300 | 8.63 | .53 |
| Forbes..... | Forbes No. 9..... | Walsen..... | B. | 11926-29..... | 4 | 2, 310 | 2, 590 | 2, 400 | 10.04 | .72 |
| Hastings..... | Hastings..... | A and B..... | B. | 26277-78 26465..... | 2 | 2, 650 | 2, 810 | 2, 710 | 12.53 | .68 |
| Merley..... | Merley..... | Engle-Starkville..... | B. | 13361-63 32046-47..... | 5 | 2, 340 | 2, 900 | 2, 660 | 15.01 | .73 |
| Primero..... | Primero..... | Trinidad..... | B. | 12027 34068-67..... | 3 | 2, 870 | 2, 560 | 2, 460 | 13.57 | .71 |
| Sopris..... | Piedmont..... | "Lower"..... | B. | 10204-10..... | 2 | 2, 740 | 2, 680 | 2, 840 | 16.02 | .65 |
| Do..... | Sopris No. 2..... | Cameron..... | B. | 31946..... | 1 | | | 2, 790 | 18.11 | .75 |

| | | | | | | | | |
|---------------------|-------------|-------------------------------|----|-------|-------|-------|-------|-------|
| Starkville..... | B. | 11134-35, 12108-97,
32360. | 5 | 2,800 | 2,900 | 2,800 | 15.31 | .60 |
| Tollerburg..... | B. | 13253-54..... | 2 | 2,530 | 2,760 | 2,640 | 12.08 | .68 |
| MESA COUNTY. | | | | | | | | |
| Cameo..... | B. | 28917-18..... | 2 | 2,710 | 2,900 | 2,840 | 9.38 | .63 |
| MOWAT COUNTY. | | | | | | | | |
| Axial..... | B. | 14528..... | 1 | | | 2,220 | 7.17 | 1.04 |
| Do..... | B. | 14529..... | 1 | | | 2,350 | 4.49 | .78 |
| James..... | B. | 17704..... | 1 | | | 2,380 | 4.37 | .61 |
| Do..... | B. | 14543..... | 1 | | | 2,480 | 2.51 | .36 |
| Joe Collom..... | B. | 14909..... | 1 | | | 2,300 | 4.50 | .68 |
| Do..... | B. | 17666..... | 1 | | | 2,210 | 7.76 | .79 |
| Craig..... | Sub. B. (?) | | 1 | | | 2,740 | 6.49 | .63 |
| Do..... | Sub. B. | 17782..... | 1 | | | 2,190 | 5.67 | .92 |
| Do..... | Sub. B. | 17833..... | 1 | | | 2,890 | 4.87 | .69 |
| Do..... | Sub. B. | 17840..... | 1 | | | 2,010 | 9.29 | .85 |
| Do..... | Sub. B. | 17862..... | 2 | 2,080 | 2,000 | 2,080 | 7.36 | 1.06 |
| Do..... | Sub. B. | 17866..... | 1 | | | 2,440 | 6.75 | 1.04 |
| Lay..... | Sub. B. (?) | 32958-69..... | 2 | 2,230 | 2,540 | 2,380 | 2.90 | .33 |
| Mount Streeter..... | Sub. B. (?) | | 1 | | | | | |
| MONTEZUMA COUNTY. | | | | | | | | |
| Cortes..... | B. | 12783..... | 1 | | | 2,080 | 15.43 | .63 |
| Do..... | B. | 12896..... | 1 | | | 2,710 | 13.64 | .80 |
| Do..... | B. | 16472-74..... | 33 | 2,240 | 2,560 | 2,370 | 6.36 | .54 |
| Do..... | B. | 12896-16479..... | 2 | 2,110 | 2,130 | 2,120 | 16.13 | 7.64 |
| Do..... | B. | 16476-77, 20287..... | 3 | 2,190 | 2,380 | 2,260 | 7.63 | .78 |
| Do..... | B. | 12851..... | 1 | | | 2,810 | 9.11 | .89 |
| Dolores..... | B. | 20496..... | 1 | | | 2,480 | 5.83 | 1.06 |
| Do..... | B. | 20500..... | 1 | | | 2,510 | 4.80 | .66 |
| OURAY COUNTY. | | | | | | | | |
| Ridgway..... | B. | 12651..... | 1 | | | 2,450 | 8.24 | .66 |
| PITKIN COUNTY. | | | | | | | | |
| Phactia..... | B. | 26826..... | 1 | | | 2,370 | 6.75 | .50 |
| RIO BLANCO COUNTY. | | | | | | | | |
| Meeker..... | B. | 12776..... | 1 | | | 2,990 | 8.95 | .55 |
| Do..... | B. | 12885..... | 1 | | | 2,310 | 2.57 | .33 |
| Do..... | B. | 12618..... | 1 | | | 2,370 | 6.97 | .96 |
| Do..... | B. | 19426..... | 1 | | | 2,310 | 6.88 | .50 |
| Do..... | B. | 12777..... | 1 | | | 2,210 | 7.00 | .81 |
| Do..... | B. | 12778..... | 1 | | | 2,780 | 9.26 | .66 |
| Do..... | B. | 12705..... | 1 | | | 2,720 | 7.03 | .95 |

TABLE I.—Softening temperatures of coal ash from coals of the United States—Continued.

| Locality. | Mine. | Coal bed. | Rank of coal. | Laboratory No. | Number of samples from mine. | Softening temperature ° F. | | | Average analysis of dry coal, percentage of— | | |
|---------------------|------------------------|-------------------|---------------|----------------|------------------------------|----------------------------|----------|----------|--|----------|------|
| | | | | | | Lowest. | Highest. | Average. | Ash. | Sulphur. | |
| 1 | | | | | | | | | | | |
| COLORADO—Continued. | | | | | | | | | | | |
| SOUTHERN COUNTY. | | | | | | | | | | | |
| Coalview. | Routt-Pinnacle. | Bear Run. | B. | 32987 | 1 | | | | 2,400 | 7.84 | 0.54 |
| Hayden. | Carry. | Unnamed. | Sb. B. | 22775 | 1 | | | | 2,480 | 6.52 | .47 |
| Do. | Dry Creek. | do. | Sb. B. | 23024 | 1 | | | | 2,340 | 4.54 | .45 |
| Do. | Green. | do. | Sb. B. | 9083 | 1 | | | | 2,250 | 5.26 | .50 |
| McGregor. | McNeil No. 1. | Wolf Creek. | B. | 32974 | 1 | | | | 2,780 | 12.07 | .53 |
| Do. | Do. | Wadga. | B. | 32975 | 1 | | | | 2,660 | 7.46 | .73 |
| Milner. | Curtis No. 1. | Wadga. | B. | 22750 | 1 | | | | 2,430 | 8.53 | .73 |
| Do. | Chargo. | Wadga. | B. | 31038 | 1 | | | | 2,430 | 6.41 | .49 |
| Do. | Elk Creek. | Wadga. | B. | 32971-72 | 2 | 2,540 | 2,980 | | 2,760 | 10.72 | .50 |
| Do. | Schuster. | Unnamed. | B. | 22749 | 1 | | | | 2,600 | 6.01 | .55 |
| Do. | Bear River. | do. | B. | 22774 | 1 | | | | 2,420 | 5.64 | .57 |
| Do. | Colorado & Utah No. 1. | Harris. | B. | 22737-38 | 3 | 2,580 | 2,870 | | 2,710 | 13.98 | .49 |
| Do. | International. | Wolf Creek. | B. | 22736 | 1 | | | | 2,570 | 2,940 | .46 |
| Do. | Mount Harris. | Wadga. | B. | 31199-206 | 8 | 2,570 | 2,940 | | 2,820 | 6.80 | .46 |
| Do. | Wadga. | do. | B. | 31233-34 | 2 | 2,540 | 2,710 | | 2,620 | 5.94 | .49 |
| Do. | Argo. | Argo or Pinnacle. | B. | 31130-33 | 4 | 2,330 | 2,520 | | 2,420 | 4.62 | .50 |
| WELD COUNTY. | | | | | | | | | | | |
| Frederick. | Baum. | Unnamed. | Sb. B. (?) | 31344 | 1 | | | | 2,080 | 5.29 | .46 |
| Puritan. | Puritan. | do. | Sb. B. | 31323 | 1 | | | | 2,170 | 4.75 | .45 |
| IDAHO. | | | | | | | | | | | |
| BOISE COUNTY. | | | | | | | | | | | |
| Horsehoe Bend. | Henry | Unnamed. | Sb. B. | 12391, 703 | 2 | 2,480 | 2,810 | | 2,640 | 30.90 | .46 |
| CASSIA COUNTY. | | | | | | | | | | | |
| Oakley. | Worthington. | Worthington. | L. | 12643 | 1 | | | | 2,130 | 27.54 | .56 |

| | | | | | | | |
|---------------------|--|--------------------------|-------------------------|--------|-------|-------|------|
| FREMONT COUNTY. | | | | | | | |
| Haden..... | Brown Bear..... | Unnamed..... | 15115..... | 1..... | 2,000 | 4.86 | .61 |
| Do..... | Horseshoe..... | do..... | 15116..... | 1..... | 2,010 | 2.36 | .41 |
| Monida..... | Scott-Buey..... | do..... | 29281..... | 1..... | 2,270 | 12.80 | .87 |
| TETON COUNTY. | | | | | | | |
| Driggs..... | Bellent..... | Unnamed..... | 29182..... | 1..... | 1,900 | 4.91 | .88 |
| Victor..... | Pine Creek Pass..... | do..... | 29280..... | 1..... | 2,160 | 88.40 | 1.41 |
| ILLINOIS. | | | | | | | |
| BUREAU COUNTY. | | | | | | | |
| Cherry..... | Cherry No. 2..... | No. 2..... | 12467-68..... | 2..... | 1,980 | 8.97 | 4.51 |
| Marquette..... | Marquette No. 1..... | No. 3..... | 14584-86..... | 3..... | 1,900 | 8.90 | 2.76 |
| CHRISTIAN COUNTY. | | | | | | | |
| Pana..... | Spring Slide..... | No. 6 (Herrin)..... | 25748-49, 26336-37.. | 4..... | 1,920 | 11.37 | 4.58 |
| FRANKLIN COUNTY. | | | | | | | |
| Buckner..... | United No. 2..... | No. 6 (Herrin)..... | 22472-77..... | 5..... | 2,080 | 9.51 | 1.17 |
| Do..... | United No. 14..... | do..... | 26129..... | 1..... | 2,150 | 8.54 | 1.14 |
| Bush..... | Bush No. 2..... | do..... | 30857-80..... | 4..... | 1,920 | 11.26 | 3.61 |
| Christopher..... | Old Ben No. 10..... | do..... | 26741-46..... | 6..... | 2,320 | 9.54 | .84 |
| Do..... | Old Ben No. 11..... | do..... | 30892-95..... | 4..... | 2,060 | 8.68 | .88 |
| Do..... | United No. 1..... | do..... | 22886-90..... | 5..... | 2,240 | 2,280 | 1.03 |
| Herrin..... | Franklin Emery No. 5..... | do..... | 30857-70..... | 4..... | 2,050 | 2,210 | .83 |
| Orient..... | Orient..... | do..... | 22442-43..... | 2..... | 2,380 | 2,440 | .98 |
| Do..... | Orient (egg and lump, 32 cars)..... | do..... | 30266-67..... | 2..... | 2,120 | 2,260 | 1.29 |
| Royalton..... | North..... | do..... | 20080-82, 20723-25..... | 6..... | 2,170 | 2,650 | .76 |
| Seaser..... | Seaser No. 1..... | do..... | 26492-96..... | 5..... | 2,130 | 2,200 | 1.21 |
| Do..... | Seaser No. 1 (egg and nut, 14 cars)..... | do..... | 30206-06, 30485-86..... | 4..... | 2,190 | 2,360 | 1.22 |
| West Frankfort..... | Brasil Block No. 18..... | do..... | 12508-10..... | 2..... | 2,150 | 2,240 | 2.00 |
| Do..... | Old Ben No. 8..... | do..... | 30857-89..... | 3..... | 2,010 | 2,170 | 7.93 |
| Do..... | Old Ben No. 9..... | do..... | 30889-95..... | 4..... | 2,030 | 2,150 | 1.45 |
| Do..... | West Frankfort No. 1..... | do..... | 22915-17, 19-20..... | 5..... | 2,140 | 2,260 | 1.14 |
| FULTON COUNTY. | | | | | | | |
| Cuba..... | Big Creek No. 3..... | No. 5 (Springfield)..... | 12479-80..... | 2..... | 1,980 | 13.21 | 3.62 |
| St. David..... | Big Creek No. 2..... | do..... | 14554-57, 59..... | 5..... | 1,900 | 12.87 | 3.56 |
| GALLATIN COUNTY. | | | | | | | |
| Shawneetown..... | Saline, middle mine..... | No. 5..... | 22425..... | 1..... | 2,010 | 8.55 | 2.91 |
| Do..... | Strong..... | No. 6..... | 22426..... | 1..... | 2,060 | 9.77 | 3.15 |
| GRUNDY COUNTY. | | | | | | | |
| Coal City..... | Wilmington Star No. 7..... | No. 2..... | 12447..... | 1..... | 1,880 | 6.71 | 2.96 |

TABLE I.—Softening temperatures of coal ash from coals of the United States—Continued.

| Locality. | Mine. | Coal bed. | Rank of coal. | Laboratory No. | Number of samples from mine. | Softening temperature ° F. | | | Average analysis of dry coal, percentage of— | |
|--------------------|---|---------------------|---------------|-----------------------------|------------------------------|----------------------------|----------|----------|--|----------|
| | | | | | | Lowest. | Highest. | Average. | Ash. | Sulphur. |
| 1 | | | | | | | | | | |
| ILLINOIS—Contd. | | | | | 2 | 3 | 4 | 5 | 6 | 7 |
| M'DONOUGH COUNTY. | | | | | | | | | | |
| Industry..... | Burdick..... | No. 2..... | B. | 15117-18..... | 2 | 2,140 | 2,240 | 2,190 | 7.55 | 3.26 |
| Plymouth..... | Stoncking..... | do..... | B. | 14972-73..... | 2 | 1,970 | 2,120 | 2,060 | 6.76 | 3.68 |
| M'LEAN COUNTY. | | | | | | | | | | |
| Bloomington..... | Bloomington..... | No. 2..... | B. | 14664-68..... | 5 | 1,960 | 2,030 | 2,000 | 10.82 | 3.32 |
| MACOUTIN COUNTY. | | | | | | | | | | |
| Gillespie..... | Superior No. 1..... | No. 6 (Herrin)..... | B. | 18545-46..... | 2 | 2,140 | 2,160 | 2,150 | 11.36 | 4.59 |
| Do..... | Superior No. 1 (car sample, run-of-mine)..... | do..... | B. | 18910..... | 1 | | | 2,040 | 15.11 | 5.76 |
| MADISON COUNTY. | | | | | | | | | | |
| Collinsville..... | Cantine No. 3..... | do..... | B. | 10959-58..... | 3 | 1,970 | 2,020 | 1,990 | 10.67 | 4.01 |
| MENARD COUNTY. | | | | | | | | | | |
| Athens..... | Wabash No. 2..... | No. 5..... | B. | 12506-07..... | 2 | 1,880 | 1,970 | 1,930 | 11.47 | 3.44 |
| MERCER COUNTY. | | | | | | | | | | |
| Matherville..... | Alden No. 7..... | No. 1..... | B. | 14608-11..... | 6 | 2,040 | 2,180 | 2,110 | 11.74 | 4.86 |
| MONTGOMERY COUNTY. | | | | | | | | | | |
| Panama..... | Shoal Creek No. 1..... | No. 6..... | B. | 21903-05, 01, 11332-33..... | 6 | 2,010 | 2,170 | 2,060 | 13.49 | 4.54 |
| PEORIA COUNTY. | | | | | | | | | | |
| Bartonville..... | Collier..... | No. 5..... | B. | 21032-34..... | 3 | 1,900 | 1,970 | 1,950 | 11.61 | 3.35 |
| Hanna City..... | Hanna City..... | do..... | B. | 22982-83..... | 4 | 1,910 | 2,030 | 1,960 | 13.72 | 3.42 |

| PERRY COUNTY. | | | | | | | | | |
|--------------------|------------------------------------|------------|--------------------------------|----|-------|-------|-------|-------|------|
| Duquoin..... | Paradise..... | No. 6..... | 1379-83, 1385-90,
26463-47. | 18 | 2,330 | 2,610 | 2,490 | 11.18 | 1.00 |
| Do..... | Security No. 1..... | do..... | 31033-36. | 4 | 2,060 | 2,130 | 2,030 | 10.60 | 1.70 |
| Pinekeyville..... | Ritchey No. 1..... | do..... | 14175, 77, 79-81..... | 5 | 1,920 | 2,170 | 2,090 | 11.46 | 3.65 |
| ST. CLAIR COUNTY. | | | | | | | | | |
| O'Fallon..... | Taylor..... | No. 6..... | 12905, 09..... | 2 | 1,980 | 2,030 | 2,000 | 12.70 | 5.04 |
| SALINE COUNTY. | | | | | | | | | |
| Eldorado..... | Dering No. 2..... | No. 5..... | 33101-03..... | 3 | 1,990 | 2,100 | 2,040 | 9.29 | 2.76 |
| Grayson..... | Grayson No. 6..... | do..... | 33091-83..... | 3 | 1,940 | 2,060 | 2,020 | 12.82 | 4.24 |
| Do..... | Saline No. 6..... | do..... | 28449..... | 1 | | | 2,090 | 10.28 | 4.14 |
| Harrisburg..... | O'Gara No. 3..... | do..... | 33081-83..... | 3 | 2,010 | 2,040 | 2,020 | 10.09 | 3.66 |
| Do..... | O'Gara No. 4..... | do..... | 12516-17..... | 2 | 2,020 | 2,020 | 2,020 | 7.87 | 2.92 |
| Do..... | O'Gara No. 9..... | do..... | 12795, 14110-13,
14138-39. | 7 | 1,960 | 2,210 | 2,090 | 7.94 | 2.15 |
| VERMILION COUNTY. | | | | | | | | | |
| Danville..... | Electric..... | No. 7..... | 13464, 86-89, 91-96. | 11 | 1,940 | 2,180 | 2,040 | 9.70 | 3.31 |
| Do..... | Schaefer..... | No. 6..... | 13450-53..... | 4 | 1,960 | 2,170 | 2,090 | 10.19 | 3.86 |
| Fairmont..... | Fairmont..... | do..... | 13548-50, 52-54. | 6 | 1,950 | 2,150 | 2,050 | 11.78 | 2.76 |
| Georgetown..... | Sharon..... | do..... | 13467-71..... | 5 | 2,070 | 2,160 | 2,100 | 12.76 | 2.61 |
| Steelton..... | Dering No. 4..... | do..... | 13569-69, 71-75,
75-77. | 8 | 2,040 | 2,190 | 2,110 | 10.53 | 2.83 |
| Westville..... | Bunsen (car sample, nut coal). | do..... | 25413..... | 1 | | | 2,180 | 10.45 | 1.99 |
| Do..... | Bunsen No. 3..... | do..... | 12445..... | 1 | | | 1,940 | 11.93 | 3.74 |
| Do..... | Little Vermilion..... | do..... | 13436-41, 43-48..... | 12 | 1,930 | 2,160 | 2,070 | 9.84 | 2.52 |
| WILLIAMSON COUNTY. | | | | | | | | | |
| Hartin..... | C. & H. (car sample, run-of-mine). | No. 7..... | 19714..... | 1 | | | 2,090 | 11.53 | 2.06 |
| Do..... | Pond Creek..... | No. 6..... | 20872-75..... | 4 | 2,010 | 2,430 | 2,220 | 9.43 | 1.01 |
| Do..... | Rand No. 2..... | do..... | 20810-14..... | 5 | 2,280 | 2,360 | 2,300 | 8.89 | 1.06 |
| Do..... | Weaver No. 2..... | do..... | 20863-65..... | 3 | 1,960 | 2,280 | 2,140 | 9.74 | 1.41 |
| INDIANA. | | | | | | | | | |
| GIBSON COUNTY. | Fort Branch..... | No. 5..... | 20865-67..... | 3 | 2,060 | 2,100 | 2,080 | 9.72 | 3.83 |
| GREENE COUNTY. | | | | | | | | | |
| Isacoville..... | Gilmour No. 7..... | No. 4..... | 31298-98..... | 4 | 2,570 | 2,790 | 2,660 | 8.43 | 1.33 |
| Linton..... | Vandalia No. 24..... | do..... | 31278-80..... | 3 | 2,570 | 2,710 | 2,590 | 8.30 | 1.49 |
| KNOX COUNTY. | | | | | | | | | |
| Bicknell..... | Indian Creek..... | No. 5..... | 19435-36..... | 2 | 2,040 | 2,210 | 2,130 | 10.28 | 4.83 |
| Bruceville..... | Oliphaunt Johnson No. 1..... | do..... | 27632-34, 36, 60-61. | 6 | 2,060 | 2,130 | 2,070 | 11.90 | 3.54 |

TABLE I.—Softening temperatures of coal ash from coals of the United States—Continued.

| Locality. | Mine. | Coal bed. | Rank of coal. | Laboratory No. | Number of samples from mine. | Softening temperature ° F. | | | Average analysis of dry coal, percentage of— | |
|--------------------|--|---------------|---------------|-----------------------------------|------------------------------|----------------------------|----------|----------|--|----------|
| | | | | | | Lowest. | Highest. | Average. | Ash. | Sulphur. |
| 1 | | | | | | | | | | |
| INDIANA.—Contd. | | | | | | | | | | |
| PIKE COUNTY. | | | | | | | | | | |
| Winslow..... | Ayrshire No. 7 (car sample, 35 cars, 2-inch lump). | No. 5..... | B. | 30228-31..... | 4 | 2,480 | 2,560 | 2,530 | 6.40 | 1.06 |
| SULLIVAN COUNTY. | | | | | | | | | | |
| Carlisle..... | Viola..... | No. 5..... | B. | 22217-19..... | 3 | 1,960 | 2,200 | 2,080 | 10.29 | 3.27 |
| Dugger..... | Ayrshire No. 27..... | No. 4..... | B. | 31280-83..... | 4 | 2,350 | 2,710 | 2,480 | 8.20 | 1.42 |
| Do..... | Vandalia No. 10..... | do..... | B. | 10960-61, 22225-28, 31262-66..... | 9 | 2,150 | 2,390 | 2,260 | 7.01 | 1.45 |
| Do..... | Vandalia No. 22..... | do..... | B. | 31274-76..... | 3 | 2,080 | 2,340 | 2,170 | 8.91 | 2.93 |
| Sullivan..... | Vandalia No. 17..... | No. 6..... | B. | 31268-72..... | 5 | 2,010 | 2,120 | 2,040 | 9.91 | 2.65 |
| Do..... | Vandalia No. 28..... | No. 4..... | B. | 31282-84..... | 3 | 2,300 | 2,420 | 2,350 | 7.81 | 1.43 |
| VANDERBURG COUNTY. | | | | | | | | | | |
| Evansville..... | Sunnyside..... | No. 5..... | B. | 20250-51..... | 2 | 2,080 | 2,100 | 2,080 | 10.31 | 2.97 |
| VERMILION COUNTY. | | | | | | | | | | |
| Blanford..... | West Clinton No. 1..... | No. 5..... | B. | 21874-75..... | 2 | 1,860 | 2,070 | 1,960 | 10.49 | 3.65 |
| Clinton..... | Clinton No. 4..... | No. 4..... | B. | 31257-60..... | 4 | 2,250 | 2,320 | 2,280 | 8.50 | 1.26 |
| Do..... | Crown Hill No. 3..... | No. 3..... | B. | 22312-13, 15-16..... | 4 | 2,040 | 2,220 | 2,140 | 9.92 | 3.64 |
| Do..... | Dering No. 1..... | do..... | B. | 22346-48, 46..... | 4 | 1,960 | 2,070 | 2,060 | 9.83 | 3.86 |
| VIGO COUNTY. | | | | | | | | | | |
| Coal Bluff..... | Vigo No. 74..... | Minabell..... | B. | 26124..... | 1 | | | 2,120 | 9.60 | 2.99 |
| Liggett..... | Vandalia No. 52..... | No. 3..... | B. | 31286-88..... | 3 | 2,030 | 2,100 | 2,060 | 12.08 | 5.51 |
| Do..... | do..... | No. 5..... | B. | 23113-16, 24033-54..... | 6 | 1,950 | 2,080 | 2,040 | 10.82 | 3.45 |
| WARREN COUNTY. | | | | | | | | | | |
| Elberfeld..... | Elberfeld..... | No. 5..... | B. | 16578..... | 1 | | | 2,190 | 11.83 | 5.30 |

| | | | | | | | | | |
|----------------------------|----------------------------|-------------------------------|----|------------------|---|-------|-------|-------|------|
| KANSAS. | | | | | | | | | |
| BARTON COUNTY. | | | | | | | | | |
| Mulberry..... | Carney Cherokee No. 2..... | Wear-Pittsburg..... | B. | 7557..... | 1 | | 1,960 | 13.04 | 5.33 |
| CHEROKEE COUNTY. | | | | | | | | | |
| Stone City..... | Mayer No. 9..... | Cherokee..... | B. | 27229-25..... | 4 | 1,900 | 2,000 | 8.72 | 3.30 |
| GRAYSON COUNTY. | | | | | | | | | |
| Edison..... | Wear No. 21..... | Cherokee..... | B. | 30238-04..... | 2 | 2,090 | 2,150 | 12.33 | 4.96 |
| Franklin..... | Western No. 16..... | do..... | B. | 17032..... | 1 | | | 12.67 | 5.63 |
| Fuller..... | Sheidan No. 2..... | do..... | B. | 11181-63..... | 3 | 1,950 | 2,110 | 9.56 | 5.23 |
| Pittsburg..... | Central (strip pit)..... | do..... | B. | 22338-39 44..... | 3 | 2,170 | 2,390 | 6.90 | 1.19 |
| Do..... | Patton..... | Unamed..... | B. | 22387..... | 1 | | | 13.56 | 3.45 |
| Yale..... | Western No. 13..... | Wear-Pittsburg..... | B. | 11208-11..... | 3 | 2,000 | 2,040 | 12.81 | 5.08 |
| LEAVENWORTH COUNTY. | | | | | | | | | |
| Landing..... | Penitentiary..... | Bevier..... | B. | 12841-43..... | 3 | 1,970 | 2,070 | 14.08 | 4.47 |
| Leavenworth..... | Home Riverside No. 1..... | do..... | B. | 12846-51..... | 3 | 1,810 | 2,020 | 13.68 | 4.96 |
| Do..... | Home Riverside No. 3..... | do..... | B. | 12846-47..... | 3 | 1,890 | 2,120 | 18.26 | 5.46 |
| KENTUCKY. | | | | | | | | | |
| BELL COUNTY. | | | | | | | | | |
| Arjay..... | Glendon..... | Straight Creek..... | B. | 24557-60..... | 4 | 2,100 | 2,120 | 3.12 | .90 |
| Chemco..... | Chenoco Hignite..... | Lower Hignite..... | B. | 21787-89..... | 3 | 2,370 | 2,500 | 4.67 | 1.10 |
| Do..... | Log Mountain No. 1..... | Mason..... | B. | 21781-84..... | 4 | 2,140 | 2,390 | 2.53 | .86 |
| Comar..... | Amru..... | do..... | B. | 21689-94..... | 3 | 2,240 | 2,380 | 6.09 | 1.61 |
| Fourmile..... | East Jellico..... | Fireclay (Dean or No. 4)..... | B. | 21799-97..... | 2 | 2,910 | 3,010 | 7.92 | 1.02 |
| Do..... | Magnet..... | do..... | B. | 21792-94..... | 3 | 2,440 | 2,910 | 6.71 | 1.10 |
| Fox Ridge..... | For Ridge..... | Straight Creek..... | B. | 21559-55..... | 4 | 1,970 | 2,070 | 1.88 | 1.07 |
| Harrison..... | Log Mountain No. 52..... | Poplar Lick..... | B. | 21617-20..... | 4 | 2,440 | 2,960 | 5.30 | 1.06 |
| Kettle Island..... | Pioneer..... | Straight Creek..... | B. | 21547-50..... | 4 | 1,990 | 2,380 | 4.85 | 1.38 |
| Rim..... | Rim No. 4..... | Hickory..... | B. | 21622-25..... | 4 | 2,020 | 2,490 | 5.37 | 1.07 |
| Straight Creek..... | Barker Nos. 2 and 3..... | Straight Creek..... | B. | 21597-99 71..... | 4 | 2,110 | 2,170 | 3.66 | 1.33 |
| Tejay..... | Tejay..... | Mason..... | B. | 21592-95..... | 4 | 2,320 | 2,430 | 3.18 | .97 |
| Varilla..... | Varilla..... | Upper Hance..... | B. | 21776-79..... | 4 | 2,150 | 2,550 | 4.74 | 1.61 |
| BUTLER COUNTY. | | | | | | | | | |
| Morgantown..... | Gillam (country bank)..... | Not classified..... | B | 18999-400..... | 2 | 2,020 | 2,180 | 8.33 | 3.7 |
| CHRISTIAN COUNTY. | | | | | | | | | |
| Empire..... | Empire..... | Empire..... | B. | 19294-96..... | 3 | 1,890 | 1,960 | 5.10 | 2.31 |
| CRITTENDEN COUNTY. | | | | | | | | | |
| Sullivan..... | Barnaby..... | Bell (?)..... | B. | 22394..... | 1 | | | 8.05 | 3.59 |
| Do..... | Newcome..... | Unamed..... | B. | 22395..... | 1 | | | 7.11 | 3.59 |

TABLE I.—Softening temperatures of coal ash from coals of the United States—Continued.

| Locality. | Mine. | Coal bed. | Rank of coal. | Laboratory No. | Number of samples from mine. | Softening temperature ° F. | | | Average analysis of dry coal, percentage of— | |
|-------------------|------------------|------------------------|---------------|----------------|------------------------------|----------------------------|----------|----------|--|----------|
| | | | | | | Lowest. | Highest. | Average. | Ash. | Sulphur. |
| 1 | | | | | | | | | | |
| KENTUCKY—Con. | | | | | | | | | | |
| DAVIES COUNTY. | | | | | | | | | | |
| Owensboro. | Fulkerson. | No. 9. | B. | 18970-71. | 2 | 2,030 | 2,080 | 2,060 | 12.18 | 4.31 |
| Do. | George Rudy. | do. | B. | 18968-69. | 3 | 2,060 | 2,190 | 2,100 | 10.76 | 3.63 |
| HAILEY COUNTY. | | | | | | | | | | |
| Bearham. | Bearham. | Kelloks (C). | B. | 24836. | 1 | | | 2,230 | 2.21 | .49 |
| Harlan. | Clover Fork. | Harlan. | B. | 24728-29 31. | 3 | 2,510 | 2,930 | 2,720 | 2.95 | .75 |
| Do. | do. | do. | B. | 24714-15. | 2 | 2,520 | 2,550 | 2,540 | 3.52 | .88 |
| Do. | Wood. | do. | B. | 24770-72. | 3 | 2,600 | +3,010 | +2,840 | 5.36 | .92 |
| HENDERSON COUNTY. | | | | | | | | | | |
| Beckett. | No. 1. | No. 9. | B. | 18992-64. | 3 | 2,020 | 2,100 | 2,060 | 11.53 | 3.14 |
| Corydon. | Corydon. | No. 12. | B. | 19108-08. | 4 | 1,860 | 2,080 | 1,970 | 12.72 | 3.89 |
| Henderson. | Nicholson. | No. 9. | B. | 18973-74. | 2 | 2,080 | 2,480 | 2,280 | 12.31 | 3.56 |
| Robards. | Panama. | do. | B. | 18976-77. | 2 | 1,960 | 2,070 | 2,030 | 11.41 | 4.12 |
| Smith Mills. | Smith Mills. | No. 12. | B. | 19108-11. | 4 | 1,990 | 2,140 | 2,070 | 9.41 | 2.03 |
| HOPKINS COUNTY. | | | | | | | | | | |
| Dawson Springs. | Workman. | Dawson. | B. | 19298-301. | 4 | 2,000 | 2,250 | 2,100 | 5.49 | 3.11 |
| Earlington. | Arnold No. 9. | No. 9. | B. | 19294-11. | 8 | 1,960 | 2,310 | 2,080 | 6.99 | 2.78 |
| Madisonville. | Bancroft. | No. 11. | B. | 19194-292. | 9 | 1,960 | 2,310 | 2,070 | 7.95 | 4.00 |
| Nebo. | Nebo. | No. 14. | B. | 19287-88. | 6 | 1,860 | 2,380 | 2,070 | 6.17 | 3.02 |
| Nortonville. | Norton No. 1. | No. 11. | B. | 19287-88. | 6 | 1,860 | 2,380 | 2,040 | 7.46 | 3.77 |
| St. Charles. | Cardinals No. 1. | No. 9. | B. | 19281-86. | 6 | 1,910 | 2,190 | 2,020 | 9.06 | 3.59 |
| Do. | For Run. | do. | B. | 19282-89. | 8 | 1,930 | 2,170 | 2,030 | 10.03 | 3.43 |
| JOHNSON COUNTY. | | | | | | | | | | |
| Van Lear. | Van Lear No. 1. | Millers Creek (No. 1). | B. | 10548-49. | 2 | 2,010 | 2,280 | 2,150 | 3.50 | 1.17 |
| Do. | Van Lear No. 2. | do. | B. | 10541. | 1 | | | 2,080 | 3.64 | 1.90 |
| Do. | Van Lear No. 3. | do. | B. | 10552. | 1 | | | 2,080 | 1.64 | 1.64 |
| Do. | Van Lear No. 4. | do. | B. | 10553. | 1 | | | 2,400 | 2.46 | .69 |

KNOX COUNTY.

Bredel.....
Elys.....
Fireday (Dean or No. 4).....
Jellico.....

LAUREL COUNTY.

East Barnstead.....
Pittsburg.....
Bonar.....
Acme.....

LETICHER COUNTY.

Flat Gap (Va.).....
Do.....
Fleming.....
Do.....
Jenkins.....
Do.....
Mader.....
McRoberts.....
Do.....
Lower Standford.....
Lower Bolling.....
Elkhorn.....
Do.....
Consolidation No. 2M.....
Local.....
Elkhorn or Koma.....
Elkhorn.....
Do.....
Do.....
Do.....
Consolidation No. 214.....
Consolidation No. 214.....

M'LEAN COUNTY.

Island.....
O'Neil.....
No. 9.....

MULLENBERG COUNTY.

Bevier.....
Central City.....
Cleaton.....
Graham.....
Lam.....
Central.....
Bevier.....
Skibo.....

OHIO COUNTY.

McHenry.....
Rockport.....
Crown.....
No. 9.....
do.....

PERRY COUNTY.

Domino.....
Douglass.....
Hazard.....
Do.....
Do.....
Lennut.....
Lothair.....
Do.....
Hazard (Haddix or No. 6).....
Fireday (Dean or No. 4).....
Hazard.....
Hazard-Dean.....
Do.....
Do.....
North Fork.....
Ashless.....
Kentucky Jewel.....
Flag (No. 7).....

Bennett No. 1.....
New Hughes.....
21537, 39-40.....
21542-45.....
21499-52.....
21539-42.....
15172.....
15173.....
20945-46.....
21308-11.....
14904-05.....
14970.....
21303-06.....
21294-97.....
21299-301, 317.....
19281-84.....
19331-32, 34-36.....
13287-89.....
15339-41.....
16674-75.....
15103.....
19338-43.....
21334-37.....
21354-57.....
21364-66.....
21368-69.....
21686.....
21359-62.....
21349-50, 52.....
21344-47.....

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1,920.....
1,970.....
1,880.....
2,020.....
2,620.....
2,910.....
2,370.....
2,630.....
2,320.....
2,730.....

2,900.....
2,590.....
2,460.....
2,120.....
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2,390.....
2,640.....
2,660.....
2,520.....
2,510.....
2,670.....
2,730.....
2,170.....
2,030.....
2,230.....
1,960.....
2,030.....
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2,120.....
2,040.....
3,010.....
3,010.....
2,460.....
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3,010.....
2,610.....
2,970.....

2,910.....
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2,080.....
2,290.....
2,890.....
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1,940.....
2,000.....
2,120.....
2,040.....
2,460.....
2,860.....
2,420.....
2,900.....
2,490.....
2,890.....

6.42.....
6.92.....
4.97.....
7.39.....
5.24.....
11.65.....
1.01.....
4.18.....
5.24.....
3.92.....
4.16.....
3.42.....
3.79.....
10.62.....
9.83.....
8.34.....
10.27.....
10.20.....
9.51.....
11.06.....
8.56.....
3.55.....
3.64.....
4.12.....
6.16.....
4.24.....
5.39.....
7.52.....

.78.....
1.56.....
2.40.....
3.70.....
1.81.....
1.01.....
.64.....
.83.....
.86.....
1.02.....
.56.....
.60.....
3.48.....
3.98.....
2.87.....
3.60.....
3.76.....
2.91.....
3.77.....
.79.....
.74.....
.63.....
.86.....
.68.....
.87.....
.67.....
.83.....

TABLE I.—Softening temperatures of coal ash from coals of the United States—Continued.

| Locality. | Mine. | Coal bed. | Rank of coal. | Laboratory No. | Number of samples from mine. | Softening temperature ° F. | | | Average analysis of dry coal, per-centage of— | |
|-------------------------|-------------------------------|------------|---------------|-----------------|------------------------------|----------------------------|----------|----------|---|----------|
| | | | | | | Lowest. | Highest. | Average. | Ash. | Sulphur. |
| 1 | | | | | | | | | | |
| MISSOURI—Contd. | | | | | | | | | | |
| HENRY COUNTY—continued. | | | | | | | | | | |
| Lewis. | Pigg No. 1. | Tobo. | B. | 11267-69 | 3 | 1,960 | 2,010 | 1,980 | 14.24 | 4.54 |
| Windsor. | Bowen No. 1. | Bituminous | B. | 75354 | 1 | | | 1,970 | 10.51 | 8.82 |
| Do. | Do. | Canal | B. | 75355 | 1 | | | 2,220 | 14.80 | 8.18 |
| Do. | Bowen No. 4. | Bowen. | B. | 10349-51 | 3 | 1,920 | 1,960 | 1,940 | 13.18 | 4.61 |
| JOHNSON COUNTY. | | | | | | | | | | |
| Sutherland. | Sutherland No. 1. | Berlier. | B. | 10346, 48 | 2 | 1,800 | 2,060 | 1,980 | 9.07 | 4.44 |
| LAFAYETTE COUNTY. | | | | | | | | | | |
| Corder. | Black Diamond. | Lexington. | B. | 10243-45 | 3 | 1,960 | 1,980 | 1,970 | 12.00 | 5.13 |
| Do. | Wilson. | do. | B. | 10242-44 | 3 | 1,950 | 1,980 | 1,970 | 12.25 | 3.54 |
| Higdonville. | Farmer No. 1. | do. | B. | 10239-41 | 3 | 2,000 | 2,120 | 2,060 | 12.02 | 3.76 |
| Lexington. | Graddy. | do. | B. | 10236-37 | 1 | 1,960 | 2,070 | 1,990 | 12.40 | 3.90 |
| Do. | Summit. | do. | B. | 75348 | 1 | | | 1,960 | 12.49 | 4.11 |
| Napoleon. | Independence. | do. | B. | 10222-24 | 3 | 1,990 | 1,990 | 1,990 | 12.85 | 3.71 |
| Warley. | Buchhorn. | Waverly. | B. | 10241-42, 75567 | 3 | 1,990 | 2,080 | 2,010 | 13.83 | 7.37 |
| Wellington. | Lebanon Exchange, branch 306. | Lexington. | B. | 10238-39 | 3 | 1,960 | 2,070 | 2,040 | 11.54 | 3.61 |
| LEWIS COUNTY. | | | | | | | | | | |
| Brockfield. | Crandall No. 1. | Tobo. | B. | 11421-23 | 3 | 1,900 | 2,160 | 2,040 | 9.51 | 4.97 |
| Do. | Walker No. 1. | do. | B. | 11412-15 | 3 | 1,900 | 2,200 | 2,040 | 8.82 | 5.12 |
| Marceline. | Landreth No. 1. | do. | B. | 11417-19 | 3 | 1,960 | 2,040 | 2,000 | 8.49 | 4.89 |
| MACON COUNTY. | | | | | | | | | | |
| Arden. | Berlier No. 2. | Berlier. | B. | 75568 | 1 | | | 1,960 | 12.48 | 6.24 |
| Do. | Do. | do. | B. | 10103 | 1 | | | 2,000 | 11.13 | 4.44 |
| Do. | Do. | do. | B. | 10104 | 1 | | | 2,000 | 11.13 | 4.44 |

MARYLAND.

ALLEGANY COUNTY.

| | | | | | | | | |
|------------------|--|-------------------------|----------------------|----|--------|--------|-------|------|
| Allegany | Tyson No. 3 | Upper Sewickley (Tyson) | 26306 | 1 | | | 6.16 | 1.01 |
| Barclaville | Bond | Bluebaugh | 26307 | 1 | | | 13.32 | 2.98 |
| Do | Emrick No. 1 | do | 26304 | 1 | | | 5.10 | .84 |
| Do | Parker | Clarion (Parker) | 26339 | 1 | | | 2.470 | 1.17 |
| Do | Pratt | Brush Creek | 26542 | 1 | | | 2.110 | 1.26 |
| Barton | Moscow No. 3 | Bakerstown | 26533 | 1 | | | 11.22 | 2.27 |
| Do | do | do | 26978 | 1 | | | 7.95 | 1.99 |
| Do | do | do | 26996 | 1 | | | 3.010 | 1.18 |
| Borden Shaft | Consolidation No. 12 | Pittsburgh ("Big Bed") | 26528 | 1 | | | 3.010 | 1.64 |
| Eckhart | Washington No. 2 | Upper Sewickley (Tyson) | 26507 | 1 | | | 18.13 | .91 |
| Ellerslie | Ellerslie Clay | Mercer (Mount Savage) | 26987 | 1 | | | 2.450 | 3.89 |
| Do | do | Quakertown | 26988 | 1 | | | 3.010 | 2.92 |
| Franklin Station | Müller & Greene No. 1 | Clarion (Parker) | 26544 | 1 | | | 2.180 | 3.67 |
| Frostburg | Washington No. 9 | Upper Sewickley (Tyson) | 26510 | 1 | | | 2.810 | 1.03 |
| Garnons Station | Consolidation No. 1 | Lower Kittanning | 26979 | 1 | | | 3.010 | 2.55 |
| Hoffman | Consolidation No. 3 | Pittsburgh (Big Bed) | 26526 | 1 | | | 2.600 | 1.07 |
| Little Allegheny | Union No. 1 | do | 26529 | 1 | | | 2.780 | 1.41 |
| Lonaconing | Georges Creek No. 3 | Upper Sewickley (Tyson) | 26509 | 1 | | | 2.410 | 1.07 |
| Do | Georges Creek (test opening) | Franklin | 26540 | 1 | | | 8.48 | 1.36 |
| Do | Do | Upper Sewickley (Tyson) | 26532-35 | 4 | 2,580 | +3,010 | 8.30 | 1.31 |
| Do | Kingsley | Waynesburg | 26984 | 1 | | | 13.75 | 2.58 |
| Lord Village | Consolidation No. 7 | Pittsburgh (Big Bed) | 26530 | 1 | | | 2.960 | .96 |
| Luka | Devon | do | 26511 | 1 | | | 11.05 | 1.53 |
| Mertens | Mertens | Pittsburgh (Big Bed) | 17498-97 | 5 | 2,150 | +3,010 | 11.81 | 2.72 |
| Midland | Consolidation No. 8 | do | 68192-95 | 2 | +3,010 | +3,010 | 17.35 | .72 |
| Do | Consolidation No. 8 (dipple samples run-ekman) | do | 68203-09-11-13 | 4 | 3,010 | +3,010 | 7.33 | .88 |
| Montell | Montell Tunnel (Mertens) | Bluebaugh | 26505 | 1 | | | 15.01 | 1.46 |
| Do | do | Lower Kittanning | 26977 | 1 | | | 8.58 | 1.40 |
| Morrison | Gin Sang | Bakerstown | 26532 | 1 | | | 2.390 | 2.15 |
| Mount Savage | Henry Mullaneys | Lower Freeport | 26543 | 1 | | | 20.51 | 4.11 |
| Do | Union No. 6 | Mercer (Mount Savage) | 26580 | 1 | | | 3.010 | 1.32 |
| Ocean | Consolidation No. 1 | Pittsburgh (Big Bed) | 26527 68178, 181-82 | 10 | 2,710 | +3,010 | 16.15 | .87 |
| Short Gap | Stanton-Georges Creek No. 1 | Upper Freeport | 197, 216-218, 21, 26 | 1 | | | 2.980 | |
| Westonport | Tacoma | Lower Kittanning | 26531 | 1 | | | 2.790 | 1.40 |
| | | | 26530 | 1 | | | 2,410 | 1.61 |

GARRETT COUNTY.

| | | | | | | | | |
|-------------|---------------|------------------|-------|---|--|--|-------|------|
| Barnum | Monroe No. 1 | Lower Kittanning | 26983 | 1 | | | 10.09 | 1.64 |
| Barton | Monroe No. 2 | Bakerstown | 26992 | 1 | | | 9.12 | .62 |
| Bayard | Nethall | Upper Freeport | 26524 | 1 | | | 10.30 | 1.56 |
| Blaine | Hamill No. 2 | Lower Kittanning | 26980 | 1 | | | 2.200 | 2.51 |
| Do | Perries No. 3 | Upper Freeport | 26981 | 1 | | | 7.88 | 1.83 |
| Bloomington | No. 7 | Lower Kittanning | 26512 | 1 | | | 2.490 | 2.12 |
| Chaffee | Chaffee | do | 26514 | 1 | | | 2,510 | 2.06 |
| Cuthrie | Guthrie | do | 26989 | 1 | | | 2,560 | 2.12 |

TABLE I.—Softening temperatures of coal ash from coals of the United States—Continued.

| Locality. | Mine. | Coal bed. | Rank of coal. | Laboratory No. | Number of samples from mine. | Softening temperature ° F. | | | Average analysis of dry coal, percentage of— | |
|---------------------------|-------------------------------|----------------------------|---------------|-------------------|------------------------------|----------------------------|----------|----------|--|----------|
| | | | | | | Lowest. | Highest. | Average. | Ash. | Sulphur. |
| 1 | | | | | | | | | | |
| MARYLAND—Continued. | | | | | | | | | | |
| GARRETT COUNTY continued. | | | | | | | | | | |
| Dodson..... | Dodson..... | Lower Kittanning | S. B. | 17647-52..... | 6 | 2,280 | 2,680 | 2,540 | 11.78 | 2.62 |
| Do..... | do..... | Upper Kittanning | S. B. | 17654..... | 1 | | | 3,010 | 10.37 | 1.04 |
| Friendsville..... | McCulloch..... | Lower Kittanning | B. | 26519..... | 1 | | | 2,190 | 11.84 | 4.11 |
| German..... | Strathmore..... | Upper Freeport..... | S. B. | 26523..... | 1 | | | 2,710 | 16.52 | 1.49 |
| Gannons Station..... | Washington No. 5..... | Bakerstown..... | S. B. | 26531..... | 1 | | | 2,660 | 10.06 | 1.47 |
| Grantsville..... | Beachey..... | Grantsville..... | S. B. | 26541..... | 1 | | | 2,490 | 8.23 | 1.22 |
| Harrison..... | Dodson No. 3..... | Lower Kittanning | S. B. | 26513..... | 1 | | | 2,530 | 9.53 | 1.84 |
| Do..... | Dodson No. 5..... | Upper Kittanning | S. B. | 26537..... | 1 | | | + 3,010 | 8.72 | .68 |
| Hubbard..... | Alex Hooking No. 1..... | Lower Kittanning | S. B. | 26515..... | 1 | | | 2,260 | 13.74 | 2.34 |
| Lonsconing..... | Koonz..... | Upper Freeport..... | S. B. | 26522..... | 1 | | | 2,170 | 22.72 | 3.20 |
| Do..... | Maryland (new opening)..... | Upper Freeport..... | S. B. | 26522..... | 1 | | | 2,510 | 8.01 | 4.01 |
| Oakland..... | Chisholm..... | Lower Kittanning | S. B. | 26517..... | 1 | | | 2,130 | 15.14 | 1.49 |
| Do..... | Leighton..... | "Galitzen"..... | B. | 26536..... | 1 | | | 2,410 | 20.13 | 4.62 |
| Do..... | John Sines..... | Mercer (Mount Savage)..... | B. | 26538..... | 1 | | | 2,170 | 8.63 | 3.93 |
| Do..... | Taylor Sines..... | Upper Freeport..... | S. B. | 26525..... | 1 | | | 2,130 | 10.30 | 2.51 |
| Do..... | Tower..... | "Galitzen"..... | S. B. | 26531..... | 1 | | | 2,130 | 10.93 | 2.51 |
| Swallow Falls..... | Beeghly..... | do..... | B. | 26535..... | 1 | | | 2,130 | 11.86 | 3.85 |
| Do..... | Shaeffer..... | Lower Kittanning..... | B. | 26518..... | 1 | | | 2,190 | | |
| MISSOURI. | | | | | | | | | | |
| ADAIR COUNTY. | | | | | | | | | | |
| Connellsville..... | Manufacturers No. 1..... | Bevier..... | B. | 10085-87..... | 3 | 1,870 | 2,100 | 1,980 | 14.96 | 4.10 |
| Kirksville..... | Rocky Ford No. 1..... | do..... | B. | 10101-02..... | 2 | 1,980 | 2,030 | 2,010 | 16.11 | 4.35 |
| Do..... | Star No. 1..... | do..... | B. | 10099-100..... | 2 | 1,940 | 1,960 | 1,950 | 15.44 | 6.13 |
| Do..... | Star..... | do..... | B. | 14799-800..... | 2 | 1,960 | 2,010 | 2,000 | 11.07 | 5.91 |
| Nowinger..... | Electric..... | do..... | B. | 75581..... | 1 | | | 2,000 | 13.56 | 3.98 |
| Do..... | Great Northern No. 21..... | do..... | B. | 10075-76, 84..... | 3 | 1,920 | 1,980 | 1,960 | 11.97 | 4.17 |
| Do..... | K. C. Midland No. 4..... | do..... | B. | 75595..... | 1 | | | 2,040 | 23.08 | 7.85 |
| Do..... | Rombauer No. 3..... | do..... | B. | 10078, 82-83..... | 1 | 1,870 | 1,940 | 1,910 | 13.96 | 3.86 |
| Stahl..... | Consolidated Stahl No. 1..... | Lexington..... | B. | 10077, 80..... | 2 | 1,940 | 1,960 | 1,950 | 14.62 | 4.62 |

| | | | | | | | | | | |
|-------------------|-------------------|----------------------|---------------------|----|----------|---|-------|-------|-------|------|
| AUDRAIN COUNTY. | Martinsburg | Martinsburg No. 1 | Mulky | B. | 11484 | 1 | 1,940 | 1,940 | 11.66 | 5.77 |
| | Vandalia | Standard | do. | B. | 9989-94 | 3 | 1,970 | 1,920 | 13.68 | 5.41 |
| BARTON COUNTY. | Pittsburg (Kans.) | Stephenson | Cherokee | B. | 29341-43 | 3 | 2,080 | 2,180 | 7.51 | 1.97 |
| | Mincedmines | Fullen No. 1 | Lower War-Pittsburg | B. | 11177-79 | 3 | 1,920 | 1,930 | 9.51 | 3.72 |
| BATES COUNTY. | Do. | Fullen No. 17 | do. | B. | 11186-87 | 3 | 1,900 | 1,940 | 12.14 | 5.17 |
| | Amsterdam | Amsterdam No. 1 | Mulberry | B. | 11300-08 | 3 | 1,940 | 1,980 | 16.19 | 4.30 |
| BOONE COUNTY. | Hume | Holland No. 1 | do. | B. | 11226-28 | 2 | 1,940 | 2,070 | 12.95 | 2.55 |
| | New Home | New Home No. 1 | Lower Rich Hill | B. | 11323-24 | 3 | 1,980 | 1,900 | 16.04 | 5.59 |
| CALDWELL COUNTY. | Rich Hill | Fleming Pitt No. 1 | do. | B. | 11197-99 | 3 | 1,960 | 2,000 | 13.70 | 5.46 |
| | Do. | Ritchie Pitt No. 1 | do. | B. | 11198-94 | 2 | 1,920 | 1,970 | 16.43 | 5.24 |
| COLUMBIA COUNTY. | Columbia | Blackfoot | Bevier | B. | 75566 | 1 | 1,900 | 1,980 | 9.22 | 3.73 |
| | Do. | Davis & Watson No. 1 | do. | B. | 11478-77 | 3 | 1,900 | 1,930 | 11.51 | 4.57 |
| CALDWELL COUNTY. | Do. | Prather No. 1 | do. | B. | 11479-81 | 3 | 1,980 | 1,970 | 14.79 | 5.55 |
| | Hamilton | Caldwell No. 1 | Bevier | B. | 10166 | 1 | 1,900 | 1,980 | 16.44 | 6.65 |
| CALLAWAY COUNTY. | Fulton | Grennel | Bevier | B. | 75562 | 1 | 1,980 | 2,000 | 14.96 | 6.24 |
| | Do. | No. 1 | do. | B. | 11501-08 | 3 | 1,980 | 1,990 | 11.74 | 4.42 |
| CLAY COUNTY. | Missouri City | Missouri City No. 1 | Lexington | B. | 10219-21 | 3 | 1,980 | 2,030 | 14.54 | 3.46 |
| | Do. | Trenton No. 3 | Tebo | B. | 10151-53 | 3 | 2,140 | 2,320 | 13.64 | 3.61 |
| DAVISON COUNTY. | Cainesville | Cainesville | Cainesville | B. | 12679-81 | 3 | 1,940 | 2,030 | 12.71 | 5.78 |
| | Melbourne | Trenton No. 1 | Lexington | B. | 11374-76 | 3 | 1,950 | 1,990 | 7.84 | 3.52 |
| HENRY COUNTY. | Calhoun | Parks No. 1 | Tebo | B. | 11314-16 | 3 | 1,920 | 2,010 | 12.87 | 4.01 |
| | Clinton | Lane No. 1 | do. | B. | 11248-50 | 3 | 1,960 | 2,000 | 14.50 | 4.28 |
| HARRISON COUNTY. | Do. | Pharis No. 1 | Jordan | B. | 11370-72 | 3 | 1,970 | 2,160 | 12.20 | 2.86 |
| | Do. | Sheldon & Holt No. 1 | do. | B. | 11263-64 | 3 | 1,980 | 2,010 | 12.88 | 5.99 |
| DEEPWATER COUNTY. | Deepwater | Dickey No. 1 | do. | B. | 11310-12 | 3 | 1,980 | 2,040 | 11.22 | 4.01 |
| | Do. | Hurst No. 1 | do. | B. | 11252-54 | 3 | 1,940 | 1,970 | 14.65 | 4.81 |

TABLE I.—Softening temperatures of coal ash from coals of the United States—Continued.

| Locality. | Mine. | Coalbed. | Rank
of coal. | Laboratory No. | Number
of
samples
from
mine. | Softening temperature ° F. | | | Average analysis
of dry coal, per-
centage of— | |
|-----------------------------|-----------------------------|------------|------------------|------------------|--|----------------------------|----------|----------|--|----------|
| | | | | | | Lowest. | Highest. | Average. | Ash. | Sulphur. |
| 1 | | | | | | | | | | |
| MISSOURI—Contd. | | | | | | | | | | |
| HENRY COUNTY—
continued. | | | | | | | | | | |
| Lewis. | Pigg No. 1. | Tebo. | B. | 11287-89. | 3 | 1,960 | 2,010 | 1,980 | 14.24 | 4.54 |
| Windsor. | Bowen No. 1. | Bituminous | B. | 75554. | 1 | | | 1,970 | 10.51 | 5.52 |
| Do. | Bowen. | Cannel | B. | 75555. | 1 | | | 2,220 | 14.80 | 8.18 |
| Do. | Bowen No. 4. | Bowen. | B. | 10348-51. | 3 | 1,920 | 1,960 | 1,940 | 13.18 | 4.61 |
| JOHNSON COUNTY. | | | | | | | | | | |
| Sutherland. | Sutherland No. 1. | Bevier. | B. | 10346, 48. | 2 | 1,890 | 2,060 | 1,980 | 9.07 | 4.44 |
| LAFAYETTE COUNTY. | | | | | | | | | | |
| Order. | Black Diamond. | Lexington. | B. | 10343-45. | 3 | 1,960 | 1,980 | 1,970 | 13.00 | 5.13 |
| Do. | Wilson. | do. | B. | 10242-44. | 3 | 1,950 | 1,960 | 1,970 | 13.25 | 3.94 |
| Higginsville. | Farmers No. 1. | do. | B. | 10239-41. | 3 | 2,000 | 2,120 | 2,040 | 19.02 | 3.76 |
| Lexington. | Graddy. | do. | B. | 10235-37. | 3 | 1,980 | 2,070 | 1,990 | 15.00 | 3.90 |
| Do. | Summit. | do. | B. | 75598. | 1 | | | 1,960 | 13.49 | 4.11 |
| Napoleon. | Independence. | do. | B. | 10232-34. | 3 | 1,950 | 1,990 | 1,960 | 16.59 | 3.71 |
| Waverly. | Buckhorn. | Waverly. | B. | 10341-42, 75567. | 3 | 1,960 | 2,060 | 2,010 | 16.53 | 7.37 |
| Wellington. | Labor Exchange, branch 305. | Lexington. | B. | 10238-30. | 3 | 1,990 | 2,070 | 2,040 | 11.54 | 3.51 |
| LINN COUNTY. | | | | | | | | | | |
| Brookfield. | Crandall No. 1. | Tebo. | B. | 11427-28. | 3 | 1,950 | 2,160 | 2,040 | 9.51 | 4.97 |
| Do. | Walker No. 1. | do. | B. | 11413-15. | 3 | 1,980 | 2,200 | 2,040 | 8.82 | 5.12 |
| Marceline. | Landreth No. 1. | do. | B. | 11417-19. | 3 | 1,960 | 2,040 | 2,000 | 8.49 | 4.86 |
| MACON COUNTY. | | | | | | | | | | |
| Ardmore. | Bevier No. 2. | Bevier. | B. | 75565. | 1 | | | 1,940 | 12.43 | 5.24 |
| Bevier. | Central No. 61. | do. | B. | 10191-93. | 3 | 1,940 | 2,060 | 2,030 | 11.27 | 3.68 |
| Do. | Northwestern No. 9. | do. | B. | 9988-90. | 3 | 1,920 | 1,990 | 1,960 | 11.12 | 4.05 |
| Macon. | Home No. 1. | Mulky. | B. | 9983-86, 75559. | 3 | 1,950 | 2,100 | 2,030 | 10.85 | 4.74 |

| | | | | | | | | | |
|---------------------------|-----------------------------|----------------|--------|-------------------|---|------------|------------|------------|------------|
| POTNAM COUNTY. | | | | | | | | | |
| Mendota..... | Mendota No. 2..... | Lexington..... | B. | 11392-400..... | 3 | 1,900..... | 1,940..... | 1,980..... | 12.05..... |
| Unionville..... | Anderson..... | do..... | B. | 14880-81..... | 2 | 2,080..... | 2,100..... | 2,070..... | 13.84..... |
| RANDOLPH COUNTY. | | | | | | | | | |
| Huntsville..... | Carson No. 1..... | Bever..... | B. | 11452-54..... | 3 | 1,990..... | 1,980..... | 1,970..... | 6.11..... |
| Do..... | Northern Central No. 2..... | do..... | B. | 11448-50..... | 3 | 1,980..... | 1,960..... | 1,970..... | 9.95..... |
| Jacksonville..... | Jacksonville..... | do..... | B. | 75580..... | 1 | 1,800..... | 1,800..... | 1,840..... | 4.46..... |
| Moheby..... | Smith Marriott..... | do..... | B. | 75584..... | 1 | 1,870..... | 1,870..... | 1,840..... | 5.08..... |
| Do..... | Ordis No. 1..... | Malley..... | B. | 11497-99..... | 3 | 1,870..... | 1,830..... | 1,910..... | 15.87..... |
| Ryder..... | Jones No. 1..... | Bever..... | B. | 10180-82..... | 3 | 1,830..... | 1,910..... | 1,860..... | 3.15..... |
| RAY COUNTY. | | | | | | | | | |
| Richmond..... | Ray County..... | Lexington..... | B. | 75583..... | 1 | 1,830..... | 2,140..... | 2,000..... | 3.20..... |
| Do..... | Ray County No. 2..... | do..... | B. | 10197-99..... | 3 | 1,800..... | 2,030..... | 2,040..... | 14.84..... |
| Do..... | Ray County No. 50..... | do..... | B. | 10194-96..... | 3 | 1,940..... | 2,220..... | 2,010..... | 15.34..... |
| Vidward..... | Vidward No. 1..... | do..... | B. | 11365-67..... | 3 | 1,850..... | 1,960..... | 1,910..... | 4.49..... |
| SULLIVAN COUNTY. | | | | | | | | | |
| Milan..... | Milan No. 1..... | Bever..... | B. | 10125-26..... | 2 | 1,850..... | 1,960..... | 1,910..... | 4.49..... |
| VERNON COUNTY. | | | | | | | | | |
| Panama..... | Jones No. 1..... | Rich Hill..... | B. | 11205-07..... | 3 | 1,930..... | 1,980..... | 1,970..... | 3.14..... |
| MONTANA. | | | | | | | | | |
| BIG HORN COUNTY. | | | | | | | | | |
| Lodgegrass..... | Local prospect..... | Carney..... | Sb. B. | 26149..... | 1 | | | 2,180..... | 6.52..... |
| Sanders..... | Prospect..... | do..... | Sb. B. | 17711..... | 1 | | | 2,260..... | 7.75..... |
| BLAINE COUNTY. | | | | | | | | | |
| Cleveland..... | Cook..... | Unnamed..... | Sb. B. | 14782..... | 1 | | | 2,240..... | .90..... |
| BROADWATER COUNTY. | | | | | | | | | |
| Lombard..... | Western..... | Unnamed..... | B. | 28555-57..... | 2 | 2,350..... | 2,470..... | 2,410..... | 8.23..... |
| CARBON COUNTY. | | | | | | | | | |
| Bear Creek..... | International..... | No. 2..... | Sb. B. | 18991-93..... | 3 | 2,010..... | 2,090..... | 2,040..... | 9.44..... |
| Do..... | No. 2..... | No. 2..... | Sb. B. | 15146-49, 51..... | 5 | 2,080..... | 2,100..... | 2,080..... | 10.46..... |
| Do..... | North Side..... | No. 3..... | Sb. B. | 15143..... | 1 | 2,040..... | 2,110..... | 1,980..... | 9.87..... |
| Do..... | South Side..... | do..... | Sb. B. | 15127-29..... | 3 | 2,040..... | 2,110..... | 2,080..... | 1.78..... |
| Red Lodge..... | Red Lodge No. 4..... | No. 4..... | Sb. B. | 29465-69..... | 4 | 2,120..... | 2,170..... | 2,080..... | 12.50..... |
| Washoe..... | Washoe..... | No. 3..... | Sb. B. | 28522-25..... | 4 | 2,000..... | 2,190..... | 2,150..... | 1.13..... |
| | | | Sb. B. | | | | | | 12.57..... |
| | | | Sb. B. | | | | | | 1.26..... |

TABLE I.—Softening temperatures of coal ash from coals of the United States—Continued.

| Locality. | Mine. | Coal bed. | Rank of coal. | Laboratory No. | Number of samples from mine. | Softening temperature ° F. | | | Average analysis of dry coal, per-centage of— | |
|------------------|----------------------|--------------|---------------|----------------|------------------------------|----------------------------|----------|----------|---|----------|
| | | | | | | Lowest. | Highest. | Average. | Ash. | Sulphur. |
| 1 | | | | | | | | | | |
| MONTANA—Con. | | | | | | | | | | |
| CHOUTEAU COUNTY. | | | | | | | | | | |
| Big Sandy..... | Mackton..... | Mackton..... | Sb. B. | 14613, 15..... | 2 | 2,250 | 2,280 | 2,250 | 14.62 | 0.70 |
| Virelle..... | Deia..... | Unamed..... | Sb. B. | 19790..... | 1 | | | 2,260 | 13.84 | .98 |
| Do..... | Price Sexton..... | do..... | Sb. B. | 19793..... | 1 | | | 2,200 | 13.54 | .74 |
| CUSTER COUNTY. | | | | | | | | | | |
| Terry..... | Cameron..... | Unamed..... | L. | 10877..... | 1 | | | 2,460 | 22.15 | .35 |
| Westmore..... | Prospect..... | do..... | L. | 10910..... | 1 | | | 2,170 | 13.03 | .46 |
| DAWSON COUNTY. | | | | | | | | | | |
| Glendive..... | Prospect..... | "B"..... | L. | 10914..... | 1 | | | 2,420 | 9.24 | .95 |
| Do..... | Snyder..... | Unamed..... | L. | 11045..... | 1 | | | 2,030 | 11.92 | 2.00 |
| Jordan..... | Foster prospect..... | do..... | Sb. B. | 19923..... | 1 | | | 2,240 | 16.17 | .68 |
| FALLON COUNTY. | | | | | | | | | | |
| Camp Creek..... | Hornet..... | Kerr..... | L. | 20370..... | 1 | | | 2,180 | 12.25 | 1.05 |
| Do..... | Kerr..... | do..... | L. | 20372..... | 1 | | | 2,000 | 23.75 | 2.03 |
| FERGUS COUNTY. | | | | | | | | | | |
| Hilger..... | Stone..... | Unamed..... | Sb. B. | 14495..... | 1 | | | 2,220 | 16.66 | 1.40 |
| Winifred..... | Caldenwood..... | do..... | Sb. B. | 20090..... | 1 | | | 2,270 | 15.25 | 1.05 |
| Do..... | Hilltop..... | do..... | Sb. B. | 20042..... | 1 | | | 2,150 | 21.15 | 1.10 |
| Prospect..... | Do..... | do..... | Sb. B. | 22545..... | 1 | | | 2,580 | 12.45 | .70 |
| Do..... | do..... | do..... | Sb. B. | 22546..... | 1 | | | 2,500 | 19.45 | .45 |
| Do..... | do..... | do..... | Sb. B. | 22550..... | 1 | | | 2,240 | 19.33 | 2.47 |
| Do..... | do..... | do..... | Sb. B. | 22519..... | 1 | | | 2,360 | 15.53 | .69 |
| Do..... | do..... | do..... | Sb. B. | 22930..... | 1 | | | 2,160 | 12.83 | .67 |
| Do..... | do..... | do..... | Sb. B. | 22946..... | 1 | | | 2,170 | 14.91 | 1.61 |
| Do..... | do..... | do..... | Sb. B. | 22948..... | 1 | | | 2,370 | 18.09 | .80 |

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| COLFAX COUNTY. | | | | | | | | | |
|--------------------|----------------------|-------------------|--------|------------------------------------|---|-------|--------|--------|-------|
| Brilliant. | Brilliant. | Tin Pan. | B. | 17703. | 1 | 2,580 | 2,910 | 2,870 | 16.51 |
| Dawson. | Dawson No. 2. | Raton. | B. | 18166, 69, 71, 74-76,
12233-34. | 7 | 2,580 | 2,910 | 2,860 | 13.67 |
| Do. | Dawson No. 6. | Dawson. | B. | 33022 | 1 | | | 2,880 | 14.96 |
| Gardiner. | Gardiner No. 1. | Raton. | B. | 14796 | 1 | | | +3,000 | 15.60 |
| Koehler. | Koehler. | do. | B. | 33017 | 1 | | | 2,940 | 11.16 |
| Do. | Koehler No. 1. | do. | B. | 12335-37, 14796 | 1 | | | +2,880 | 12.45 |
| Do. | Koehler No. 2. | do. | B. | 33018 | 4 | 2,710 | +3,000 | 3,040 | 11.77 |
| Sugarite. | Sugarite No. 1. | Sugarite. | B. | 14791, 34589-91 | 4 | 2,670 | 2,880 | 2,760 | 9.10 |
| Do. | Sugarite No. 2. | do. | B. | 14792, 34586-98 | 5 | 2,660 | 2,860 | 2,780 | 10.13 |
| Van Houten. | Van Houten No. 1. | Raton. | B. | 32592 | 1 | | | 3,000 | 11.27 |
| Do. | Van Houten No. 4. | do. | B. | 32598 | 1 | | | +3,000 | 12.18 |
| Yankee. | Yankee. | Yankee. | B. | 17746 | 1 | | | 2,460 | 13.02 |
| Do. | Yankee No. 3. | do. | B. | 13965-66 | 2 | 2,800 | 2,850 | 2,820 | 13.89 |
| LINCOLN COUNTY. | | | | | | | | | |
| White Oaks. | Old Abe. | Old Abe. | B. | 15053 | 1 | | | 2,380 | 15.07 |
| Do. | Wild Cat. | Unnamed. | B. | 15054 | 1 | | | 2,730 | 17.30 |
| MCINTOSH COUNTY. | | | | | | | | | |
| Alhambra. | Coal Basin. | Coal Basin No. 1. | SB. B. | 32272-73 | 2 | 2,000 | 2,180 | 2,060 | 7.19 |
| Do. | Diamond. | Astec (?) | SB. B. | 19217-19 | 3 | 2,150 | 2,440 | 2,270 | 9.68 |
| Do. | do. | Diamond No. 1. | SB. B. | 32278 | 1 | | | 2,210 | 7.75 |
| Do. | do. | Diamond No. 2. | SB. B. | 32279 | 1 | | | 2,380 | 9.97 |
| Black Rock. | Unnamed. | Unnamed. | SB. B. | 15032 | 1 | | | 2,860 | 10.34 |
| Gallup. | Zuni Indian School. | Black Diamond (?) | SB. B. | 19136 | 1 | | | 2,950 | 11.79 |
| Do. | Bartlett. | Astec (?) | SB. B. | 19136 | 1 | | | 2,120 | 5.86 |
| Do. | Beddow. | do. | SB. B. | 19136 | 1 | | | 2,540 | 4.92 |
| Do. | Carreto. | Otero (?) | SB. B. | 19163 | 1 | | | 2,460 | 7.93 |
| Do. | Defiance. | Defiance. | SB. B. | 19223 | 1 | | | 2,490 | 7.35 |
| Do. | Gallup Southwestern. | Black Diamond. | SB. B. | 19162 | 1 | | | 2,370 | 6.68 |
| Do. | Heaton. | No. 2. | SB. B. | 19286 | 1 | | | 2,070 | 3.93 |
| Do. | do. | No. 3. | SB. B. | 19287 | 1 | | | 2,400 | 6.04 |
| Do. | Jones. | Defiance. | SB. B. | 19221 | 1 | | | 2,490 | 8.56 |
| Do. | Myers. | Myers. | SB. B. | 19213 | 1 | | | 2,670 | 10.49 |
| Do. | Navajo. | No. 1. | SB. B. | 19131 | 1 | | | 2,150 | 4.65 |
| Do. | do. | No. 2. | SB. B. | 19132 | 2 | 2,350 | 2,610 | 2,480 | 8.30 |
| Do. | do. | No. 5. | SB. B. | 19133-34 | 1 | | | 2,350 | 7.58 |
| Gibson. | Navajo No. 1. | do. | SB. B. | 32350 | 4 | 2,600 | 2,700 | 2,650 | 9.02 |
| Do. | Navajo No. 2. | do. | SB. B. | 32359-41 | 1 | | | 2,060 | 5.64 |
| Do. | Weaver. | No. 2. | SB. B. | 19137 | 1 | | | 2,060 | 5.64 |
| Do. | do. | No. 3. | SB. B. | 19138, 32332 | 2 | 1,960 | 2,060 | 2,000 | 3.92 |
| Do. | do. | No. 3. | SB. B. | 19139 | 1 | | | 2,170 | 7.21 |
| Do. | do. | No. 4. | SB. B. | 19140, 32335 | 2 | 2,340 | 2,720 | 2,530 | 8.43 |
| Do. | do. | No. 5. | SB. B. | 32378 | 1 | | | 2,150 | 5.31 |
| Heaton. | Heaton. | No. 2. | SB. B. | | | | | | |
| RIO ARriba COUNTY. | | | | | | | | | |
| Lumberton. | Prospect. | Unnamed. | B. | 31078 | 1 | | | 2,420 | 2.28 |
| Monero. | Old Blinn. | Upper. | B. | 26279 | 1 | | | 2,340 | 5.68 |

TABLE I.—Softening temperatures of coal ash from coals of the United States—Continued.

| Locality. | Mine. | Coal bed. | Rank of coal. | Laboratory No. | Number of samples from mine. | Softening temperature ° F. | | | Average analysis of dry coal, percentage of— | |
|-----------------------|--------------------------|---------------|---------------|----------------|------------------------------|----------------------------|----------|----------|--|----------|
| | | | | | | Lowest. | Highest. | Average. | Ash. | Sulphur. |
| 1 | | | | | | | | | | |
| NEW MEXICO—Continued. | | | | | | | | | | |
| SAN JUAN COUNTY. | | | | | | | | | | |
| Durango (Colo.) | New Mexico. | "A" | B. | 17749. | 1 | | | 2,550 | 14.06 | 0.71 |
| Farmington | Bill Thomas | Carbonero | B. | 20249. | 1 | | | 2,940 | 11.68 | 0.67 |
| Do. | Blake | Unnamed | Sb. B. | 23003. | 1 | | | 2,220 | 6.68 | 1.14 |
| Do. | Government | Hogback | B. | 17760. | 1 | | | 2,280 | 3.46 | 1.72 |
| Do. | Jones | Carbonero | B. | 26250. | 1 | | | 2,670 | 12.80 | 1.42 |
| Do. | Local | Unnamed | Sb. B. | 22685. | 1 | | | 2,660 | 8.31 | 0.83 |
| Do. | Marcellus | Carbonero | Sb. B. | 26025. | 1 | | | 2,650 | 9.13 | 0.68 |
| Do. | Prospect | do. | Sb. B. | 26026. | 1 | | | 2,160 | 14.02 | 2.61 |
| Do. | do. | Carbonero (?) | Sb. B. | 22807. | 1 | | | 2,260 | 7.68 | 0.83 |
| Do. | Ship Rock Indian School. | Unnamed | Sb. B. | 26006. | 1 | | | 2,370 | 4.64 | 0.95 |
| Fruitland | Black Diamond | Carbonero | Sb. B. | 22508. | 1 | | | 2,340 | 11.19 | 0.68 |
| Do. | Hendrickson | do. | Sb. B. | 22509. | 1 | | | 2,720 | 10.33 | 0.67 |
| Pueblo Bonita | do. | do. | Sb. B. | 23004. | 1 | | | 2,660 | 8.78 | 1.80 |
| SANTA FE COUNTY. | | | | | | | | | | |
| Madrid | Anthracite No. 4 | White Ash | A. | 14886. | 1 | | | 2,210 | 10.08 | .82 |
| Do. | Blacksmith | Peacock (?) | B. | 14884. | 1 | | | 2,500 | 7.97 | 1.33 |
| Do. | Peacock prospect | Peacock | B. | 14887. | 1 | | | 2,180 | 11.37 | 1.64 |
| SOCORRO COUNTY. | | | | | | | | | | |
| Carthage | Government | Carthage | B. | 30214-15. | 2 | 2,760 | 2,960 | 2,860 | 14.35 | .85 |
| Magdalena | Prospect | Unnamed | Sb. B. | 20072. | 1 | | | 2,260 | 12.24 | .83 |
| Do. | do. | do. | Sb. B. | 17728. | 1 | | | 2,480 | 12.26 | .83 |
| Do. | do. | do. | B. | 17002. | 1 | | | 2,220 | 7.58 | .58 |
| NORTH DAKOTA. | | | | | | | | | | |
| ADAMS COUNTY. | | | | | | | | | | |
| Haynes | Nipper & Moore | Haynes | L. | 14642. | 1 | | | 2,270 | 12.81 | 2.27 |
| Do. | William Pinkham | do. | L. | 14644. | 1 | | | 2,140 | 12.68 | 2.21 |

| | | | | | | | | | |
|-------------------------|------------------------|------------------------------|--------------------------------|---|------------|------------|------------|-----------|--|
| BOWMAN COUNTY. | | | | | | | | | |
| Amor..... | Durkin prospect..... | T-Cross..... | 14857..... | 1 | | 2,060..... | 10.96..... | 1.15..... | |
| Bowman..... | Open pit..... | Unnamed..... | 14553..... | 1 | | 2,130..... | 18.39..... | | |
| Scranton..... | Scranton..... | Harmon (?)..... | 14485..... | 1 | | 2,180..... | 12.47..... | 1.01..... | |
| MERCER COUNTY. | | | | | | | | | |
| Beulah..... | Beulah..... | Beulah..... | 33069..... | 1 | | 2,270..... | 9.51..... | 1.07..... | |
| Volmer..... | Manhaven..... | Manhaven..... | 33068..... | 1 | | 2,140..... | 9.75..... | .66..... | |
| Stanton..... | Teuber..... | Unnamed..... | 33067..... | 1 | | 2,390..... | 9.02..... | 1.10..... | |
| MORTON COUNTY. | | | | | | | | | |
| Almont..... | Ramsland..... | Unnamed..... | 19798..... | 1 | | 2,290..... | 9.17..... | .64..... | |
| Hebron..... | Hebron..... | do..... | 32442-43..... | 2 | 2,030..... | 2,060..... | 13.14..... | 1.56..... | |
| Leth..... | Kalbank..... | Haynes (?)..... | 17537..... | 1 | | 2,090..... | 16.13..... | 2.30..... | |
| Do..... | Jones..... | do..... | 14729..... | 1 | | 2,150..... | 13.63..... | 1.07..... | |
| New Salem..... | Dakota Products..... | Unnamed..... | 20053..... | 1 | | 2,100..... | 11.84..... | 2.13..... | |
| Do..... | New Salem No. 1..... | do..... | 34524-25..... | 2 | 2,290..... | 2,370..... | 10.22..... | 1.40..... | |
| Do..... | Unnamed..... | do..... | 19801..... | 1 | | 2,180..... | 12.83..... | 1.07..... | |
| WARD COUNTY. | | | | | | | | | |
| Burlington..... | Conon..... | Lignite..... | 31702-04..... | 3 | 2,130..... | 2,200..... | 16.63..... | .40..... | |
| WILLIAMS COUNTY. | | | | | | | | | |
| Avoca..... | Bruegger..... | "C"..... | 12687..... | 1 | | 2,240..... | 7.39..... | .51..... | |
| Ray..... | Pittsley..... | Lignite..... | 29251, 53..... | 2 | 2,240..... | 2,310..... | 8.02..... | .62..... | |
| Trenton..... | Geltz..... | Unnamed..... | 12411..... | 1 | | 2,270..... | 11.90..... | 2.41..... | |
| Wheelock..... | Jim Monuen..... | do..... | 29252..... | 1 | | 2,430..... | 14.63..... | 2.28..... | |
| Williston..... | Husely-Ellithorpe..... | Top..... | 19370-71, 917, 18..... | 4 | 2,000..... | 2,140..... | 11.75..... | 1.82..... | |
| Do..... | Powell..... | "C"..... | 12688..... | 1 | | 2,190..... | 9.28..... | 1.24..... | |
| Do..... | U. S. Reclamation..... | Middle or "B"..... | 12533, 19365-66, 19621-22..... | 5 | 2,190..... | 2,310..... | 10.19..... | .98..... | |
| OHIO. | | | | | | | | | |
| BELMONT COUNTY. | | | | | | | | | |
| Alliedonia..... | Shipman..... | Meigs Creek (Sewickley)..... | 20237..... | 1 | | 2,060..... | 18.46..... | 4.20..... | |
| Do..... | Moore..... | Washington..... | 20238..... | 1 | | 2,520..... | 21.90..... | 2.98..... | |
| Do..... | Stoffel..... | Waynesburg..... | 20246..... | 1 | | 2,390..... | 15.14..... | 2.71..... | |
| Bailey Mills..... | Cochran No. 2..... | Pittsburgh (No. 4)..... | 20187-88..... | 2 | 1,960..... | 1,960..... | 9.35..... | 4.64..... | |
| Barneville..... | Davy..... | Meigs Creek (Sewickley)..... | 20176..... | 1 | | 2,490..... | 11.72..... | 3.82..... | |
| Boston..... | Thomas..... | Waynesburg..... | 20241..... | 1 | | 2,370..... | 15.44..... | 3.16..... | |
| Hunter..... | Milhoan..... | do..... | 20174..... | 1 | | 2,370..... | 16.95..... | 3.69..... | |
| Do..... | Kemp..... | Uniontown..... | 20175..... | 1 | | 2,160..... | 16.10..... | 2.99..... | |
| Marlin Ferry..... | Laughlin..... | Pittsburgh (No. 8)..... | 34072-04..... | 3 | 2,040..... | 2,040..... | 8.82..... | 3.58..... | |
| Somerton..... | Brown..... | Waynesburg..... | 20234..... | 1 | | 2,460..... | 16.15..... | 3.03..... | |
| Temperanceville..... | Jeffries..... | Pittsburgh (No. 8)..... | 20230..... | 1 | | 2,290..... | 9.86..... | 4.75..... | |

| | | | | | | | | |
|-----------------|-----------------------|-------------------------|-----------------|---|-------|-------|-------|-------|
| Cream City | Cream City | Middle Kittanning | 25800 | 1 | | 2,410 | 11.45 | 2.12 |
| Empire | Empire | Pittsburgh (No. 8) | 25784 | 2 | 2,160 | 2,150 | 9.67 | 4.15 |
| Hopdale | Parlet |do. | 13444-45 | 1 | | 2,210 | 8.11 | 2.91 |
| Irontide | Nicholson | Upper Freeport (No. 7) | 25031 | 1 | 2,250 | 2,210 | 8.52 | 2.98 |
| Do. | Do. | Lower Kittanning | 25038 | 1 | | 1,960 | 10.90 | 6.07 |
| Piney Fork | East Ohio No. 2 | Pittsburgh (No. 8) | 15491 | 1 | | 2,710 | 4.98 | 2.99 |
| Do. | Cabbage Run |do. | 15491 | 1 | | 2,120 | 7.63 | 2.73 |
| Do. | Piney Fork No. 1 |do. | 15498 | 1 | | 2,630 | 6.80 | 2.81 |
| Do. | Piney Fork No. 2 |do. | 15498 | 1 | | 2,160 | 7.26 | 2.68 |
| Smithfield | Plum Run No. 5 |do. | 15498 | 1 | | 2,200 | 7.43 | 1.99 |
| Staebenville | La Balle (W. Va.) | Lower Freeport | 15678 | 2 | 2,100 | 2,170 | 10.34 | 4.08 |
| Yellow Creek | New Somerset | Pittsburgh (No. 8) | 25786 | 1 | 2,140 | 2,240 | 8.22 | 3.54 |
| Do. | Yellow Creek | Upper Freeport (No. 7) | 25800-91 | 2 | | | | |
| MONROE COUNTY. | | | | | | | | |
| Coats Station | Mobley | Uniontown | 20249 | 1 | | 2,280 | 16.10 | 4.16 |
| NOBLE COUNTY. | | | | | | | | |
| Belle Valley | Noble | (No. 7) | 17213-14 | 2 | 2,130 | 2,240 | 8.71 | 3.13 |
| Mount Ephraim | Wiley Carter | Melgs Creek (Sewickley) | 20235 | 1 | | 2,320 | 10.81 | 4.32 |
| Quaker City | Griffin |do. | 20185 | 1 | | 2,460 | 13.17 | 3.77 |
| Steamtown | Moore |do. | 20240 | 1 | | 2,310 | 10.92 | 5.05 |
| OKLAHOMA. | | | | | | | | |
| COAL COUNTY. | | | | | | | | |
| Lehigh | Folsom Morris No. 5 | McAlester | 30707-11 | 5 | 1,960 | 2,120 | 12.89 | 4.56 |
| Do. | Folsom Morris No. 8 |do. | 30713-17 | 5 | 1,970 | 2,150 | 12.20 | 4.19 |
| Do. |do. |do. | 10064 | 1 | | 2,160 | 11.06 | 3.92 |
| Phillips | Folsom Morris No. 6 |do. | 30719-23 | 1 | 2,110 | 2,190 | 9.30 | 3.76 |
| CRAIG COUNTY. | | | | | | | | |
| Bluesacket | Coates | Unnamed | 20716 | 1 | | 2,160 | 10.61 | 6.34 |
| Estalia | Boot strip pit. |do. | 20718 | 1 | | 2,050 | 10.68 | 6.39 |
| Vinita | Heldebrand strip pit. |do. | 20717 | 1 | | 2,110 | 10.61 | 6.83 |
| Welch | Mills strip pit. |do. | 20715 | 1 | | 2,040 | 12.33 | 6.06 |
| HASKELL COUNTY. | | | | | | | | |
| McCurtain | Blue Ridge No. 4 | Panama | 28940-41 | 2 | 2,130 | 2,190 | 6.13 | 1.01 |
| Do. | Blue Ridge No. 5 |do. | 28939, 42 | 2 | 2,100 | 2,120 | 7.49 | .91 |
| Do. | San Bois No. 2 | McCurtain (Hartshorne) | 10057, 13868-92 | 6 | 1,990 | 2,110 | 6.92 | .84 |
| Stigler | Turner strip pit. | Stigler | 17946 | 1 | | 1,940 | 4.00 | .64 |
| Do. |do. |do. | 26323 | 1 | | 2,090 | 2.63 | .65 |
| Do. | Strip pit. |do. | 30344 | 1 | | 2,050 | 4.69 | .71 |
| Tamaha | Old Slope |do. | 30708 | 1 | | 1,920 | 8.09 | 3.60 |
| Do. | Nunnally strip pit. |do. | 26324 | 1 | | 2,250 | 6.25 | 3.95 |
| Whitfield | Lagon strip pit. | Stigler (?) | 26325 | 1 | | 2,000 | 2.62 | 1.44 |

TABLE I.—Softening temperatures of coal ash from coals of the United States—Continued.

| Locality. | Mine. | Coal bed. | Rank of coal. | Laboratory No. | Number of samples from mine. | Softening temperature ° F. | | | Average analysis of dry coal, percentage of— | |
|---------------------------|--------------------------|-------------------|---------------|--------------------------------------|------------------------------|----------------------------|----------|----------|--|----------|
| | | | | | | Lowest. | Highest. | Average. | Ash. | Sulphur. |
| 1 | | | | | | | | | | |
| OKLAHOMA—Continued. | | | | | | | | | | |
| LAFAYETTE COUNTY. | | | | | | | | | | |
| Adams. | Adams No. 6. | Lower Hartshorne. | B. | 19829-53. | 2 | 1,920 | 1,990 | 1,960 | 3.21 | 1.38 |
| Goven. | Rock Island No. 40. | do. | B. | 30270-76. | 7 | 1,910 | 2,070 | 2,030 | 6.96 | 1.46 |
| Wilburton. | Deguan-McConnell No. 5. | Upper Hartshorne. | B. | 27531-34. | 4 | 2,020 | 2,340 | 2,190 | 5.16 | .98 |
| LE FLORE COUNTY. | | | | | | | | | | |
| Bokoshe. | Slope No. 3. | Hartshorne. | S. B. | 26801-02. | 2 | 2,220 | 2,300 | 2,270 | 5.12 | .78 |
| Hughes (Lettimer County). | Turkey Creek No. 2. | do. | B. | 10065. | 1 | | | 2,060 | 6.46 | .89 |
| Williams. | Williams No. 1. | do. | S. B. | 19673-77. | 5 | 1,870 | 2,020 | 1,970 | 8.85 | 1.02 |
| OKMULGEE COUNTY. | | | | | | | | | | |
| Henryetta. | Victoria No. 1. | Henryetta. | B. | 10177-78. | 2 | 1,960 | 2,000 | 1,980 | 8.03 | 1.59 |
| PITTSBURG COUNTY. | | | | | | | | | | |
| Adams. | Eclipse No. 1. | Lower Hartshorne. | B. | 16143-44. | 2 | 2,030 | 2,080 | 2,060 | 4.16 | 1.42 |
| Do. | Rock Island No. 5. | McAlester. | B. | 11915-17, 30660-66. | 10 | 2,030 | 2,280 | 2,190 | 4.92 | .71 |
| Do. | Rock Island No. 38. | do. | B. | 11592-94, 11670, 15188-86, 30666-58. | 8 | 1,950 | 2,160 | 2,060 | 5.28 | .56 |
| Buck. | Buck No. 22. | Upper Hartshorne. | B. | 26150. | 1 | | | 2,200 | 4.78 | 1.68 |
| Craig. | Bolen-Darnall No. 4. | McAlester. | B. | 15104-06. | 2 | 2,200 | 2,280 | 2,250 | 5.39 | .87 |
| Halleysville. | Blue Creek No. 7. | Lower Hartshorne. | B. | 80401-02. | 2 | 2,000 | 2,060 | 2,030 | 6.02 | 2.31 |
| Do. | Bailey-Ola No. 2. | Upper Hartshorne. | B. | 30398-99. | 2 | 2,090 | 2,180 | 2,110 | 8.50 | 1.86 |
| Hartshorne. | Rock Island No. 8. | Lower Hartshorne. | B. | 10083, 19703. | 2 | 1,990 | 2,030 | 2,010 | 5.79 | 1.84 |
| McAlester. | Bushy No. 6. | McAlester. | B. | 11696, 86-90. | 6 | 2,190 | 2,270 | 2,230 | 4.40 | .68 |
| Pittsburg. | McAlester Edwards No. 1. | do. | B. | 10090. | 1 | | | 2,190 | 8.97 | 3.38 |
| Pocahontas No. 1. | McAlester No. 1. | Lower Hartshorne. | B. | 19728. | 1 | | | 2,060 | 5.12 | .81 |
| Rock Island No. 10. | do. | do. | B. | 30222-26. | 5 | 1,980 | 2,120 | 2,060 | 7.77 | 1.70 |

| ROGERS COUNTY. | Catale No. 1..... | Unnamed..... | B. | 20780..... | 1..... | | 2,160..... | 10.25..... | 5.40..... |
|-------------------|-----------------------------------|---------------|--------|---------------|--------|------------|------------|------------|-----------|
| | Chambers..... | do..... | B. | 20781..... | 1..... | | 2,520..... | 5.10..... | 4.86..... |
| | Collinsville..... | Dawson..... | B. | 20714..... | 1..... | | 1,860..... | 8.73..... | 4.09..... |
| SEQUOYAH COUNTY. | | | | | | | | | |
| | Brewer strip pit..... | Unnamed..... | S. B. | 26800..... | 1..... | | 2,170..... | 5.76..... | 1.82..... |
| TULSA COUNTY. | | | | | | | | | |
| | Southwestern No. 1..... | Dawson..... | B. | 20712..... | 1..... | | 1,920..... | 9.36..... | 3.62..... |
| | Hickory No. 2..... | do..... | B. | 20713..... | 1..... | | 1,970..... | 8.75..... | 4.01..... |
| WAGONER COUNTY. | | | | | | | | | |
| | Arkansas Valley strip pit..... | Unnamed..... | B. | 20832..... | 1..... | | 1,890..... | 7.25..... | 2.75..... |
| | Kirk strip pit..... | do..... | B. | 20343..... | 1..... | | 2,060..... | 14.21..... | 6.78..... |
| OREGON. | | | | | | | | | |
| CLACKAMAS COUNTY. | | | | | | | | | |
| | Prospect..... | Unnamed..... | B. (?) | 22492..... | 1..... | | 2,890..... | 22.04..... | .53..... |
| COOS COUNTY. | | | | | | | | | |
| | Beaver Hill..... | Newport..... | Sb. B. | 27951..... | 1..... | | 2,060..... | 11.59..... | .94..... |
| | Henryville..... | do..... | Sb. B. | 27956..... | 1..... | | 2,060..... | 4.79..... | .86..... |
| | Cabin prospect..... | Unnamed..... | B. | 17705..... | 1..... | | 2,780..... | 16.86..... | 1.46..... |
| | Donnell prospect..... | do..... | B. | 17708..... | 1..... | | 2,790..... | 26.71..... | 1.14..... |
| | Everett Association prospect..... | Meyers..... | B. | 17708..... | 1..... | | 2,500..... | 22.93..... | 1.62..... |
| | Hillis prospect..... | Anderson..... | B. | 14827..... | 1..... | | 2,480..... | 16.92..... | 1.70..... |
| | Meyers prospect..... | Meyers..... | B. | 17705..... | 1..... | | 2,480..... | 26.69..... | 1.86..... |
| | Pulford prospect..... | Anderson..... | B. | 19877-78..... | 2..... | 2,480..... | 2,590..... | 32.26..... | .93..... |
| | Reeves prospect..... | Carter..... | B. | 17704..... | 1..... | | 2,660..... | 40.70..... | .62..... |
| GRANT COUNTY. | | | | | | | | | |
| | Old prospect..... | Unnamed..... | L. | 18125..... | 1..... | | 2,260..... | 47.65..... | 2.16..... |
| MALEBUR COUNTY. | | | | | | | | | |
| | Honolulu prospect..... | Unnamed..... | L. | 12385..... | 1..... | | 2,420..... | 45.63..... | 3.82..... |
| WHEELER COUNTY. | | | | | | | | | |
| | Dry Hollow prospect..... | Unnamed..... | Sb. B. | 18128-27..... | 2..... | 2,480..... | 2,670..... | 37.60..... | .54..... |

TABLE I.—Softening temperatures of coal ash from coals of the United States—Continued.

| Locality. | Mine. | Coal bed. | Rank of coal. | Laboratory No. | Number of samples from mine. | Softening temperature ° F. | | | Average analysis of dry coal, percentage of— | |
|-------------------------------------|----------------------|--------------------------------|---------------|-----------------------------|------------------------------|----------------------------|----------|----------|--|----------|
| | | | | | | Lowest. | Highest. | Average. | Ash. | Sulphur. |
| 1 | | | | | | | | | | |
| PENNSYLVANIA. | | | | | | | | | | |
| ALLEGHENY COUNTY. | | | | | | | | | | |
| Bruceton..... | U. S. experimental. | Pittsburgh..... | B. | 11985, 12282, 12557, 22618. | 4 | 2, 280 | 2, 510 | 2, 400 | 6.82 | 1.23 |
| Campbell..... | New Field No. 1..... | Lower Freeport..... | B. | 32255 | 1 | | | 2, 440 | 6.16 | .52 |
| Do..... | do..... | Upper Freeport..... | B. | 32256 | 1 | | | 2, 180 | 9.04 | 1.72 |
| Do..... | do..... | Upper and Lower Freeport. | B. | 29329-59 | 2 | 2, 010 | 2, 150 | 2, 060 | 7.77 | 2.76 |
| Creighton..... | do..... | do..... | B. | 29329-34, 90-91. | 5 | 2, 350 | 2, 960 | 2, 590 | 10.66 | 1.25 |
| Elite..... | do..... | do..... | B. | 11130, 32 | 2 | 2, 410 | 2, 530 | 2, 470 | 6.73 | 1.01 |
| Elizabeth..... | do..... | do..... | B. | 20182, 34 | 2 | 2, 300 | 2, 580 | 2, 440 | 6.04 | .80 |
| Patterson No. 2..... | do..... | Pittsburgh..... | B. | 20182, 34 | 2 | 2, 300 | 2, 580 | 2, 440 | 6.13 | 1.22 |
| Oak Station..... | do..... | do..... | B. | 10845 | 1 | | | 2, 430 | | |
| ARMSTRONG COUNTY. | | | | | | | | | | |
| Snyder..... | do..... | Lower Kittanning..... | B. | 28694-95 | 2 | 2, 000 | 2, 220 | 2, 160 | 10.38 | 4.22 |
| Campbell..... | do..... | Upper Kittanning..... | B. | 30290-91 | 2 | 2, 000 | 2, 080 | 2, 020 | 17.75 | 5.04 |
| Logansport P. O. | do..... | Upper Freeport..... | B. | 28697-98 | 2 | 2, 140 | 2, 170 | 2, 160 | 10.80 | 3.20 |
| Adrian P. O. (Montgomeryville)..... | do..... | Middle Kittanning..... | B. | 28774-75 | 2 | 2, 300 | 2, 360 | 2, 330 | 28.80 | 3.72 |
| West Kittanning..... | do..... | Lower Freeport..... | B. | 28721-22 | 2 | 2, 070 | 2, 070 | 2, 070 | 10.06 | 3.28 |
| Yatesboro..... | do..... | do..... | B. | 28672-73 | 2 | 2, 060 | 2, 100 | 2, 080 | 10.83 | 3.20 |
| BEAVER COUNTY. | | | | | | | | | | |
| Cannelton..... | do..... | No. 4 (Brookville)..... | B. | 34650-51 | 2 | 1, 900 | 2, 080 | 2, 030 | 7.12 | 3.10 |
| Do..... | do..... | No. 7 (Upper Freeport)..... | B. | 34654-57 | 2 | 1, 980 | 2, 160 | 2, 070 | 6.57 | 2.10 |
| Do..... | do..... | No. 6 (Middle Kittanning)..... | B. | 34637-38 | 2 | 2, 130 | 2, 230 | 2, 180 | 5.12 | 1.03 |
| Do..... | do..... | No. 3 (Lower Mercer)..... | B. | 34653-54 | 2 | 1, 960 | 2, 080 | 2, 020 | 15.06 | 4.61 |
| Smiths Ferry..... | do..... | Upper Freeport..... | B. | 26587 | 1 | | | 2, 290 | 5.09 | 1.45 |
| BEDFORD COUNTY. | | | | | | | | | | |
| Hopewell..... | do..... | Upper Freeport..... | S. B. | 18065-69 | 5 | 2, 460 | 3, 010 | 2, 670 | 12.45 | 2.01 |

| BLAIR COUNTY. | Upper Freeport..... | 3 | 2, 120 | 2, 460 | 2, 240 | 6.87 | 1.96 |
|----------------------------|------------------------|----|---------|---------|---------|-------|------|
| Do | do | 1 | | | 2, 120 | 9.61 | 1.96 |
| BRADFORD COUNTY. | | | | | | | |
| Long Valley | Clarion..... | 2 | 3, 000 | +3, 000 | +3, 000 | 11.99 | 1.10 |
| BUTLER COUNTY. | | | | | | | |
| Bruin | Brookville..... | 2 | 2, 220 | 2, 480 | 2, 350 | 11.51 | 2.47 |
| Butler | Upper Freeport..... | 1 | | | 2, 080 | 7.75 | 3.07 |
| Do | Upper Kittanning..... | 1 | | | 2, 280 | 7.53 | 2.98 |
| Do | Do | 1 | | | 2, 060 | 4.50 | 3.73 |
| Do | Upper Kittanning..... | 2 | 3, 010 | +3, 010 | +3, 010 | 4.54 | 1.90 |
| Chicora | Upper Freeport..... | 1 | | | 2, 000 | 8.67 | 1.26 |
| Claytona | Middle Kittanning..... | 1 | | | 2, 130 | 12.77 | 3.89 |
| Evans City | Upper Freeport..... | 1 | | | 2, 110 | 8.55 | 3.71 |
| Young Shatt. | Upper Freeport..... | 4 | 2, 450 | 2, 580 | 2, 500 | 14.53 | 4.95 |
| Annandale No. 2. | Brookville and Clarion | 1 | | | 2, 060 | 11.37 | 4.54 |
| North Pittsburgh Realty | Middle Kittanning..... | 2 | 2, 020 | 2, 360 | 2, 190 | 11.56 | 2.98 |
| Victoria | Upper Freeport..... | 1 | | | 2, 150 | 8.41 | 1.32 |
| Reamer | do | 1 | | | 1, 960 | 10.77 | 3.92 |
| Jefferson Center | Middle Kittanning..... | 1 | | | 2, 220 | 6.50 | 2.67 |
| Nealy | Upper Freeport..... | | | | | | |
| Unlonville | Upper Freeport..... | | | | | | |
| Eagle | Upper Freeport..... | | | | | | |
| CAMBERIA COUNTY. | | | | | | | |
| Barnesboro | Lower Freeport..... | 5 | 2, 340 | 2, 570 | 2, 470 | 6.57 | 1.50 |
| Do | Upper Freeport..... | 4 | 2, 410 | 2, 490 | 2, 450 | 8.48 | 1.54 |
| Do | Lower Freeport..... | 5 | 2, 410 | 2, 540 | 2, 480 | 7.13 | 1.52 |
| Beaver Run | Upper Freeport..... | 6 | 2, 070 | 2, 980 | 2, 380 | 8.49 | 1.81 |
| Do | Lower Freeport..... | 4 | 2, 010 | 2, 380 | 2, 250 | 8.49 | 2.72 |
| Pennsylvania No. 15. | Upper Kittanning..... | 5 | 2, 280 | +3, 010 | +2, 440 | 7.26 | 1.92 |
| Do | Lower Kittanning..... | 5 | 2, 140 | 2, 470 | 2, 280 | 6.80 | 2.04 |
| Wilmore No. 1. | Upper Freeport..... | 5 | 1, 990 | 2, 450 | 2, 270 | 7.27 | 2.28 |
| Do | do | 9 | 2, 450 | +3, 010 | +2, 490 | 5.75 | 1.14 |
| Colver | Lower Kittanning..... | 9 | 3, 010 | +3, 010 | +3, 010 | 4.82 | .66 |
| Henricks | do | 5 | 2, 380 | +3, 010 | 2, 610 | 6.81 | 1.57 |
| Feeless No. 1. | do | 5 | 2, 380 | +3, 010 | 2, 380 | 10.06 | 1.78 |
| Feeless No. 4. | Upper Freeport..... | 2 | 2, 270 | 2, 330 | 2, 300 | 6.27 | 1.43 |
| Galitzin | do | 4 | 2, 280 | 2, 390 | 2, 350 | 6.27 | 1.47 |
| Do | do | 2 | 2, 280 | 2, 390 | 2, 350 | 6.27 | 1.47 |
| Galitzin (car sample) | do | 1 | | | 2, 490 | 8.79 | 1.47 |
| Pennsylvania No. 11 | Lower Freeport..... | 5 | 2, 430 | 2, 640 | 2, 540 | 7.14 | 1.46 |
| Do | do | 5 | 2, 200 | 2, 580 | 2, 380 | 7.63 | 1.86 |
| Pennsylvania No. 12. | do | 3 | 2, 160 | 2, 280 | 2, 210 | 9.71 | 2.19 |
| Smokeless No. 1. | Upper Kittanning..... | 4 | 2, 240 | 2, 450 | 2, 340 | 10.40 | 2.47 |
| Sunnyside | do | 3 | +3, 010 | +3, 010 | +3, 010 | 6.36 | .67 |
| Scap Level No. 2. | Lower Kittanning..... | 4 | 2, 960 | 3, 010 | 3, 000 | 4.93 | .72 |
| Lincoln No. 1 | do | 11 | 2, 160 | 2, 480 | 2, 370 | 6.47 | 1.84 |
| Lincoln No. 1 (car sample) | do | 1 | | | 2, 150 | 7.76 | 1.84 |
| Springfield No. 1 | do | 5 | 2, 080 | 2, 280 | 2, 110 | 7.65 | 2.18 |
| Miller No. 1 | do | 10 | 2, 260 | +3, 010 | +2, 730 | 6.68 | 2.18 |
| Puritan No. 1 | do | 4 | 2, 220 | 2, 580 | 2, 380 | 7.65 | 2.07 |

TABLE I.—Softening temperatures of coal ash from coals of the United States—Continued.

| Locality. | Mine. | Coal bed. | Rank of coal. | Laboratory No. | Number of samples from mine. | Softening temperature ° F. | | | Average analysis of dry coal, percentage of— | |
|----------------------------|---------------------------|---------------------------|---------------|--|------------------------------|----------------------------|----------|----------|--|----------|
| | | | | | | Lowest. | Highest. | Average. | Ash. | Sulphur. |
| 1 | | | | | | | | | | |
| PENNSYLVANIA—Continued. | | | | | | | | | | |
| CAMBERIA COUNTY—continued. | | | | | | | | | | |
| St. Michael..... | Maryland..... |do..... | S. B. | 25717-22, 30118-19, 27, 29, 68185, 87, 88, 68219-20. | 15 | 2, 630 | +3, 010 | +2, 890 | 6.13 | .93 |
| Sonman..... | Sonman Slope..... | Upper Freeport..... | S. B. | 26398-403. | 6 | 2, 050 | 2, 280 | 2, 170 | 6.44 | 1.78 |
| Spangler..... | Pennsylvania No. 21..... | Lower Freeport..... | B. | 12144-47. | 4 | 2, 240 | +3, 010 | +2, 550 | 6.49 | 1.32 |
| Do..... | Pennsylvania No. 22..... |do..... | B. | 12198-27, 40-42. | 5 | 2, 530 | 2, 880 | 2, 620 | 6.57 | 1.27 |
| Van Ormer..... | Peerless No. 1..... | Upper Freeport..... | B. | 10275-77. | 3 | 2, 420 | 3, 010 | 2, 760 | 8.94 | .86 |
| Do..... | Peerless No. 2..... | Lower Freeport..... | B. | 10272-74. | 3 | 2, 330 | 2, 500 | 2, 410 | 10.38 | 2.12 |
| Vintondale..... | Vinton No. 6..... | Lower Kittanning..... | S. B. | 10254, 56, 57. | 3 | 2, 100 | 2, 410 | 2, 310 | 6.95 | 2.08 |
| Windber..... | Eureka No. 40..... |do..... | S. B. | 68179, 80-91, 204, 206-08, 222. | 8 | 2, 520 | +3, 010 | +2, 650 | 7.33 | 1.22 |
| CAMBERON COUNTY. | | | | | | | | | | |
| Sterling Run..... | Berry..... | Freeport..... | B. | 75467-68. | 2 | 2, 850 | 2, 940 | 2, 900 | 6.45 | .70 |
| CENTER COUNTY. | | | | | | | | | | |
| Clarence..... | Lehigh Valley No. 22..... | Brookville..... | B. | 17447 | 1 | | | +3, 010 | 14.85 | 1.01 |
| Do..... | Peermansite..... | Lower Freeport..... | S. B. | 17445 | 1 | | | +2, 220 | 8.08 | 1.86 |
| Do..... |do..... | Lower Kittanning..... | B. | 17446 | 1 | | | +3, 010 | 12.83 | .91 |
| Gillingtown..... | Lehigh Valley No. 15..... | Lower Freeport..... | B. | 17448 | 1 | | | +2, 300 | 11.64 | 2.79 |
| Ocedus Mills..... | Electric..... | Lower Kittanning..... | S. B. | 12667-70. | 4 | 2, 370 | +3, 010 | +2, 680 | 7.95 | 2.07 |
| Do..... | Madison No. 10..... |do..... | S. B. | 12667-61. | 5 | 2, 350 | 2, 560 | 2, 470 | 7.52 | 2.11 |
| Do..... | Weston..... |do..... | S. B. | 12663-65. | 3 | 2, 450 | 2, 660 | 2, 560 | 7.93 | 2.11 |
| CLARION COUNTY. | | | | | | | | | | |
| Church Hill..... | Church Hill..... | Lower Kittanning..... | B. | 34525-26. | 3 | 2, 340 | 2, 330 | 2, 330 | 10.24 | 2.59 |
| Hillville..... | Blue Goose..... |do..... | B. | 34537-38. | 3 | 2, 210 | 2, 270 | 2, 240 | 8.94 | 3.56 |
| Monteary..... | Monteary..... | Clarion..... | B. | 34528-29. | 2 | 2, 120 | 2, 190 | 2, 160 | 7.63 | 3.58 |
| Parker..... | Clarion River..... | Upper Kittanning (?)..... | B. | 34522-23. | 2 | 2, 060 | 2, 270 | 2, 170 | 8.31 | 4.69 |
| Phillipston..... | James..... | Lower Kittanning..... | B. | 34531-32. | 2 | 2, 300 | 2, 310 | 2, 310 | 7.88 | 3.10 |
| Sarah Furnace..... | Sarah Furnace..... |do..... | B. | 34534-35. | 2 | 2, 140 | 2, 180 | 2, 160 | 8.45 | 3.37 |

| | | | | | | | | | | |
|---------------|-----------------|------------------|------|------------------|-----|-------|--------|--------|-------|-------|
| CLARK COUNTY. | Reed | Lower Freepoot. | B. | 22535-39 | 8 | 2,100 | 2,420 | 2,200 | 9.36 | 2.47 |
| Berwindale | Porta Run No. 3 | Lower Kittanning | S.B. | 12007-97 | 8 | 2,580 | +3,010 | +2,350 | 8.95 | 1.54 |
| Bartman | Leane No. 1 | Upper Freepoot | B. | 12046 | 1 | | | 2,540 | 8.04 | 1.61 |
| Do. | Leane No. 2 | do. | B. | 12046-15 | 2 | 2,450 | 2,540 | 2,500 | 8.49 | 1.02 |
| Do. | Leane No. 3 | do. | B. | 20342-43, 404-66 | 3 | 2,390 | 2,510 | 2,620 | 7.74 | 1.22 |
| Do. | Leane No. 4 | do. | B. | 20338-41 | 4 | 2,440 | 2,560 | 2,650 | 7.77 | 1.35 |
| Do. | Leane No. 5 | do. | B. | 20330-31 | 5 | 2,740 | 2,850 | 2,940 | 7.53 | 1.48 |
| Do. | Leane No. 6 | do. | B. | 75730-40, 42 | 6 | 2,170 | 2,280 | 2,370 | 7.58 | 1.61 |
| Do. | Leane No. 7 | do. | B. | 75628 | 7 | | | 2,460 | 7.67 | 1.74 |
| Do. | Leane No. 8 | do. | B. | 75625 | 8 | | | 2,550 | 7.77 | 1.87 |
| Do. | Leane No. 9 | do. | B. | 75622 | 9 | | | 2,640 | 7.87 | 1.99 |
| Do. | Leane No. 10 | do. | B. | 75619 | 10 | | | 2,730 | 7.97 | 2.12 |
| Do. | Leane No. 11 | do. | B. | 75616 | 11 | | | 2,820 | 8.07 | 2.25 |
| Do. | Leane No. 12 | do. | B. | 75613 | 12 | | | 2,910 | 8.17 | 2.38 |
| Do. | Leane No. 13 | do. | B. | 17443 | 13 | 2,740 | 2,840 | 2,930 | 8.27 | 2.51 |
| Do. | Leane No. 14 | do. | B. | 75809-10 | 14 | | | 3,000 | 8.37 | 2.64 |
| Do. | Leane No. 15 | do. | B. | 17441 | 15 | | | 3,090 | 8.47 | 2.77 |
| Do. | Leane No. 16 | do. | B. | 17442 | 16 | | | 3,180 | 8.57 | 2.90 |
| Do. | Leane No. 17 | do. | B. | 75674 | 17 | | | 3,270 | 8.67 | 3.03 |
| Do. | Leane No. 18 | do. | B. | 75671 | 18 | | | 3,360 | 8.77 | 3.16 |
| Do. | Leane No. 19 | do. | B. | 75668 | 19 | | | 3,450 | 8.87 | 3.29 |
| Do. | Leane No. 20 | do. | B. | 75665 | 20 | | | 3,540 | 8.97 | 3.42 |
| Do. | Leane No. 21 | do. | B. | 75662 | 21 | | | 3,630 | 9.07 | 3.55 |
| Do. | Leane No. 22 | do. | B. | 75659 | 22 | | | 3,720 | 9.17 | 3.68 |
| Do. | Leane No. 23 | do. | B. | 75656 | 23 | | | 3,810 | 9.27 | 3.81 |
| Do. | Leane No. 24 | do. | B. | 75653 | 24 | | | 3,900 | 9.37 | 3.94 |
| Do. | Leane No. 25 | do. | B. | 75650 | 25 | | | 3,990 | 9.47 | 4.07 |
| Do. | Leane No. 26 | do. | B. | 75647 | 26 | | | 4,080 | 9.57 | 4.20 |
| Do. | Leane No. 27 | do. | B. | 75644 | 27 | | | 4,170 | 9.67 | 4.33 |
| Do. | Leane No. 28 | do. | B. | 75641 | 28 | | | 4,260 | 9.77 | 4.46 |
| Do. | Leane No. 29 | do. | B. | 75638 | 29 | | | 4,350 | 9.87 | 4.59 |
| Do. | Leane No. 30 | do. | B. | 75635 | 30 | | | 4,440 | 9.97 | 4.72 |
| Do. | Leane No. 31 | do. | B. | 75632 | 31 | | | 4,530 | 10.07 | 4.85 |
| Do. | Leane No. 32 | do. | B. | 75629 | 32 | | | 4,620 | 10.17 | 4.98 |
| Do. | Leane No. 33 | do. | B. | 75626 | 33 | | | 4,710 | 10.27 | 5.11 |
| Do. | Leane No. 34 | do. | B. | 75623 | 34 | | | 4,800 | 10.37 | 5.24 |
| Do. | Leane No. 35 | do. | B. | 75620 | 35 | | | 4,890 | 10.47 | 5.37 |
| Do. | Leane No. 36 | do. | B. | 75617 | 36 | | | 4,980 | 10.57 | 5.50 |
| Do. | Leane No. 37 | do. | B. | 75614 | 37 | | | 5,070 | 10.67 | 5.63 |
| Do. | Leane No. 38 | do. | B. | 75611 | 38 | | | 5,160 | 10.77 | 5.76 |
| Do. | Leane No. 39 | do. | B. | 75608 | 39 | | | 5,250 | 10.87 | 5.89 |
| Do. | Leane No. 40 | do. | B. | 75605 | 40 | | | 5,340 | 10.97 | 6.02 |
| Do. | Leane No. 41 | do. | B. | 75602 | 41 | | | 5,430 | 11.07 | 6.15 |
| Do. | Leane No. 42 | do. | B. | 75599 | 42 | | | 5,520 | 11.17 | 6.28 |
| Do. | Leane No. 43 | do. | B. | 75596 | 43 | | | 5,610 | 11.27 | 6.41 |
| Do. | Leane No. 44 | do. | B. | 75593 | 44 | | | 5,700 | 11.37 | 6.54 |
| Do. | Leane No. 45 | do. | B. | 75590 | 45 | | | 5,790 | 11.47 | 6.67 |
| Do. | Leane No. 46 | do. | B. | 75587 | 46 | | | 5,880 | 11.57 | 6.80 |
| Do. | Leane No. 47 | do. | B. | 75584 | 47 | | | 5,970 | 11.67 | 6.93 |
| Do. | Leane No. 48 | do. | B. | 75581 | 48 | | | 6,060 | 11.77 | 7.06 |
| Do. | Leane No. 49 | do. | B. | 75578 | 49 | | | 6,150 | 11.87 | 7.19 |
| Do. | Leane No. 50 | do. | B. | 75575 | 50 | | | 6,240 | 11.97 | 7.32 |
| Do. | Leane No. 51 | do. | B. | 75572 | 51 | | | 6,330 | 12.07 | 7.45 |
| Do. | Leane No. 52 | do. | B. | 75569 | 52 | | | 6,420 | 12.17 | 7.58 |
| Do. | Leane No. 53 | do. | B. | 75566 | 53 | | | 6,510 | 12.27 | 7.71 |
| Do. | Leane No. 54 | do. | B. | 75563 | 54 | | | 6,600 | 12.37 | 7.84 |
| Do. | Leane No. 55 | do. | B. | 75560 | 55 | | | 6,690 | 12.47 | 7.97 |
| Do. | Leane No. 56 | do. | B. | 75557 | 56 | | | 6,780 | 12.57 | 8.10 |
| Do. | Leane No. 57 | do. | B. | 75554 | 57 | | | 6,870 | 12.67 | 8.23 |
| Do. | Leane No. 58 | do. | B. | 75551 | 58 | | | 6,960 | 12.77 | 8.36 |
| Do. | Leane No. 59 | do. | B. | 75548 | 59 | | | 7,050 | 12.87 | 8.49 |
| Do. | Leane No. 60 | do. | B. | 75545 | 60 | | | 7,140 | 12.97 | 8.62 |
| Do. | Leane No. 61 | do. | B. | 75542 | 61 | | | 7,230 | 13.07 | 8.75 |
| Do. | Leane No. 62 | do. | B. | 75539 | 62 | | | 7,320 | 13.17 | 8.88 |
| Do. | Leane No. 63 | do. | B. | 75536 | 63 | | | 7,410 | 13.27 | 9.01 |
| Do. | Leane No. 64 | do. | B. | 75533 | 64 | | | 7,500 | 13.37 | 9.14 |
| Do. | Leane No. 65 | do. | B. | 75530 | 65 | | | 7,590 | 13.47 | 9.27 |
| Do. | Leane No. 66 | do. | B. | 75527 | 66 | | | 7,680 | 13.57 | 9.40 |
| Do. | Leane No. 67 | do. | B. | 75524 | 67 | | | 7,770 | 13.67 | 9.53 |
| Do. | Leane No. 68 | do. | B. | 75521 | 68 | | | 7,860 | 13.77 | 9.66 |
| Do. | Leane No. 69 | do. | B. | 75518 | 69 | | | 7,950 | 13.87 | 9.79 |
| Do. | Leane No. 70 | do. | B. | 75515 | 70 | | | 8,040 | 13.97 | 9.92 |
| Do. | Leane No. 71 | do. | B. | 75512 | 71 | | | 8,130 | 14.07 | 10.05 |
| Do. | Leane No. 72 | do. | B. | 75509 | 72 | | | 8,220 | 14.17 | 10.18 |
| Do. | Leane No. 73 | do. | B. | 75506 | 73 | | | 8,310 | 14.27 | 10.31 |
| Do. | Leane No. 74 | do. | B. | 75503 | 74 | | | 8,400 | 14.37 | 10.44 |
| Do. | Leane No. 75 | do. | B. | 75500 | 75 | | | 8,490 | 14.47 | 10.57 |
| Do. | Leane No. 76 | do. | B. | 75497 | 76 | | | 8,580 | 14.57 | 10.70 |
| Do. | Leane No. 77 | do. | B. | 75494 | 77 | | | 8,670 | 14.67 | 10.83 |
| Do. | Leane No. 78 | do. | B. | 75491 | 78 | | | 8,760 | 14.77 | 10.96 |
| Do. | Leane No. 79 | do. | B. | 75488 | 79 | | | 8,850 | 14.87 | 11.09 |
| Do. | Leane No. 80 | do. | B. | 75485 | 80 | | | 8,940 | 14.97 | 11.22 |
| Do. | Leane No. 81 | do. | B. | 75482 | 81 | | | 9,030 | 15.07 | 11.35 |
| Do. | Leane No. 82 | do. | B. | 75479 | 82 | | | 9,120 | 15.17 | 11.48 |
| Do. | Leane No. 83 | do. | B. | 75476 | 83 | | | 9,210 | 15.27 | 11.61 |
| Do. | Leane No. 84 | do. | B. | 75473 | 84 | | | 9,300 | 15.37 | 11.74 |
| Do. | Leane No. 85 | do. | B. | 75470 | 85 | | | 9,390 | 15.47 | 11.87 |
| Do. | Leane No. 86 | do. | B. | 75467 | 86 | | | 9,480 | 15.57 | 12.00 |
| Do. | Leane No. 87 | do. | B. | 75464 | 87 | | | 9,570 | 15.67 | 12.13 |
| Do. | Leane No. 88 | do. | B. | 75461 | 88 | | | 9,660 | 15.77 | 12.26 |
| Do. | Leane No. 89 | do. | B. | 75458 | 89 | | | 9,750 | 15.87 | 12.39 |
| Do. | Leane No. 90 | do. | B. | 75455 | 90 | | | 9,840 | 15.97 | 12.52 |
| Do. | Leane No. 91 | do. | B. | 75452 | 91 | | | 9,930 | 16.07 | 12.65 |
| Do. | Leane No. 92 | do. | B. | 75449 | 92 | | | 10,020 | 16.17 | 12.78 |
| Do. | Leane No. 93 | do. | B. | 75446 | 93 | | | 10,110 | 16.27 | 12.91 |
| Do. | Leane No. 94 | do. | B. | 75443 | 94 | | | 10,200 | 16.37 | 13.04 |
| Do. | Leane No. 95 | do. | B. | 75440 | 95 | | | 10,290 | 16.47 | 13.17 |
| Do. | Leane No. 96 | do. | B. | 75437 | 96 | | | 10,380 | 16.57 | 13.30 |
| Do. | Leane No. 97 | do. | B. | 75434 | 97 | | | 10,470 | 16.67 | 13.43 |
| Do. | Leane No. 98 | do. | B. | 75431 | 98 | | | 10,560 | 16.77 | 13.56 |
| Do. | Leane No. 99 | do. | B. | 75428 | 99 | | | 10,650 | 16.87 | 13.69 |
| Do. | Leane No. 100 | do. | B. | 75425 | 100 | | | 10,740 | 16.97 | 13.82 |
| Do. | Leane No. 101 | do. | B. | 75422 | 101 | | | 10,830 | 17.07 | 13.95 |
| Do. | Leane No. 102 | do. | B. | 75419 | 102 | | | 10,920 | 17.17 | 14.08 |
| Do. | Leane No. 103 | do. | B. | 75416 | 103 | | | 11,010 | 17.27 | 14.21 |
| Do. | Leane No. 104 | do. | B. | 75413 | 104 | | | 11,100 | 17.37 | 14.34 |
| Do. | Leane No. 105 | do. | B. | 75410 | 105 | | | 11,190 | 17.47 | 14.47 |
| Do. | Leane No. 106 | do. | B. | 75407 | 106 | | | 11,280 | 17.57 | 14.60 |
| Do. | Leane No. 107 | do. | B. | 75404 | 107 | | | 11,370 | 17.67 | 14.73 |
| Do. | Leane No. 108 | do. | B. | 75401 | 108 | | | 11,460 | 17.77 | 14.86 |
| Do. | Leane No. 109 | do. | B. | 75398 | 109 | | | 11,550 | 17.87 | 14.99 |
| Do. | Leane No. 110 | do. | B. | 75395 | 110 | | | 11,640 | 17.97 | 15.12 |
| Do. | Leane No. 111 | do. | B. | 75392 | 111 | | | 11,730 | 18.07 | 15.25 |
| Do. | Leane No. 112 | do. | B. | 75389 | 112 | | | 11,820 | 18.17 | 15.38 |
| Do. | Leane No. 113 | do. | B. | 75386 | 113 | | | 11,910 | 18.27 | 15.51 |
| Do. | Leane No. 114 | do. | B. | 75383 | 114 | | | 12,000 | 18.37 | 15.64 |
| Do. | Leane No. 115 | do. | B. | 75380 | 115 | | | 12,090 | 18.47 | 15.77 |
| Do. | Leane No. 116 | do. | B. | 75377 | 116 | | | 12,180 | 18.57 | 15.90 |
| Do. | Leane No. 117 | do. | B. | 75374 | 117 | | | 12,270 | 18.67 | 16.03 |
| Do. | Leane No. 118 | do. | B. | 75371 | 118 | | | 12,360 | 18.77 | 16.16 |
| Do. | Leane No. 119 | do. | B. | 75368 | 119 | | | 12,450 | 18.87 | 16.29 |
| Do. | Leane No. 120 | do. | B. | 75365 | 120 | | | 12,540 | 18.97 | 16.42 |
| Do. | Leane No. 121 | do. | B. | 75362 | 121 | | | 12,630 | 19.07 | 16.55 |
| Do. | Leane No. 122 | do. | B. | 75359 | 122 | | | 12,720 | 19.17 | 16.68 |
| Do. | Leane No. 123 | do. | B. | 75356 | 123 | | | 12,810 | 19.27 | 16.81 |
| Do. | Leane No. 124 | do. | B. | 75353 | 124 | | | 12,900 | 19.37 | 16.94 |
| Do. | Leane No. 125 | do. | B. | 75350 | 125 | | | 12,990 | 19.47 | 17.07 |
| Do. | Leane No. 126 | do. | B. | 75347 | 126 | | | 13,080 | 19.57 | 17.20 |
| Do. | Leane No. 127 | do. | B. | 75344 | 127 | | | 13,170 | 19.67 | 17.33 |
| Do. | Leane No. 128 | do. | B. | 75341 | 128 | | | 13,260 | 19.77 | 17.46 |
| Do. | Leane No. 129 | do. | B. | 75338 | 129 | | | 13,350 | 19.87 | 17.59 |
| Do. | Leane No. 130 | do. | B. | 75335 | 130 | | | 13,440 | 19.97 | 17.72 |
| Do. | Leane No. 131 | do. | B. | 75332 | 131 | | | 13,530 | 20.07 | 17.85 |
| Do. | Leane No. 132 | do. | B. | 75329 | 132 | | | 13,620 | 20.17 | 17.98 |
| Do. | Leane No. 133 | do. | B. | 75326 | 133 | | | 13,710 | 20.27 | 18.11 |
| Do. | Leane No. 134 | do. | B. | 75323 | 134 | | | 13,800 | 20.37 | 18.24 |
| Do. | Leane No. 135 | do. | B. | 75320 | 135 | | | 13,890 | 20.47 | 18.37 |
| Do. | Leane No. 136 | do. | B. | 75317 | 136 | | | 13,980 | 20.57 | 18.50 |
| Do. | Leane No. 137 | do. | B. | 75314 | 137 | | | 14,070 | 20.67 | 18.63 |
| Do. | Le | | | | | | | | | |

TABLE I.—Softening temperatures of coal ash from coals of the United States—Continued.

| Locality. | Mine. | Coal bed. | Rank of coal. | Laboratory No. | Number of samples from mine. | Softening temperature ° F. | | | Average analysis of dry coal, percentage of— | |
|---------------------|----------------------------|--------------------|---------------|----------------|------------------------------|----------------------------|----------|----------|--|----------|
| | | | | | | Lowest. | Highest. | Average. | Ash. | Sulphur. |
| 1 | | | | | | | | | | |
| PENNSYLVANIA—Contd. | | | | | | | | | | |
| FAYETTE COUNTY. | | | | | | | | | | |
| East Millsboro. | Husted. | Pittsburgh. | B. | 23082-95. | 4 | 2,110 | 2,440 | 2,280 | 8.40 | 1.79 |
| HUNTINGDON COUNTY. | | | | | | | | | | |
| Jacobs. | Barnet. | Lower Kittanning. | S.B. | 10319. | 1 | | | +3,010 | 6.38 | .88 |
| Do. | Jacobs. | Fulton. | S.B. | 10315-17. | 3 | 2,770 | +3,010 | +2,890 | 8.64 | 1.63 |
| Do. | Start. | do. | S.B. | 10318. | 1 | | | +3,010 | 8.34 | 1.03 |
| Robertdale. | Robertdale. | do. | S.B. | 12115-19. | 5 | 2,370 | +3,010 | +2,890 | 6.55 | 1.05 |
| Woodvale. | Woodvale. | do. | S.B. | 12121-23. | 3 | +3,010 | +3,010 | +3,010 | 5.92 | .95 |
| INDIANA COUNTY. | | | | | | | | | | |
| Ernest. | Ernest No. 2. | Upper Freeport. | B. | 24191-92. | 2 | 2,190 | 2,240 | 2,220 | 10.70 | 2.04 |
| Glen Campbell. | Electric No. 8. | Upper Kittanning. | B. | 75812-13. | 2 | 2,510 | 2,630 | 2,570 | 10.13 | 1.58 |
| Do. | Falcon No. 9. | Lower Freeport. | B. | 75821. | 1 | | | 2,390 | 12.76 | 2.51 |
| Do. | Falcon No. 8. | Middle Kittanning. | B. | 75820. | 1 | | | 2,830 | 9.52 | 1.53 |
| Do. | Hillsdale No. 6. | Upper Freeport. | B. | 75819. | 1 | | | 2,450 | 10.78 | 1.75 |
| Do. | Indiana No. 6. | do. | B. | 12102-04, 05. | 4 | 2,410 | 2,550 | 2,490 | 8.58 | 1.28 |
| Gypsy. | Trojan. | Lower Freeport. | B. | 75818. | 1 | | | 2,250 | 5.79 | 2.24 |
| Homer City. | Lucerne No. 1. | Upper Freeport. | B. | 10305-08-12. | 6 | 2,080 | 2,560 | 2,290 | 7.97 | 2.12 |
| Do. | Lucerne No. 3. | do. | B. | 10309-05. | 3 | | | 2,190 | 8.28 | 1.93 |
| Locust. | Locust. | do. | B. | 75750-51. | 2 | 2,140 | 2,280 | 2,210 | 9.27 | 3.35 |
| Robindale. | Robindale. | Lower Kittanning. | S.B. | 25135-38. | 4 | 2,440 | 2,480 | 2,470 | 7.86 | 2.20 |
| Scott Glen. | Brush Valley. | do. | S.B. | 22501-05. | 5 | 2,200 | 2,380 | 2,280 | 8.09 | 2.08 |
| Do. | Brush Valley (car sample). | do. | S.B. | 22507. | 1 | | | 2,150 | 8.16 | 3.00 |
| JEFFERSON COUNTY. | | | | | | | | | | |
| Brookwayville. | West Clarion. | Lower Freeport. | B. | 17448. | 1 | | | 2,140 | 10.48 | 2.97 |
| Do. | do. | Lower Kittanning. | B. | 17455. | 1 | | | 2,120 | 10.66 | 3.45 |
| Do. | Upper Freeport. | Upper Freeport. | B. | 17449. | 1 | | | 2,300 | 8.98 | 1.74 |

| | | | | | | | |
|--------------------|---|-----------------------|-------|--------|--------|-------|------|
| Conifer..... | Brookville..... | 34910-11..... | 2 160 | 2 220 | 2 190 | 10 21 | 4 59 |
| Fuller..... | do..... | 75789-37..... | 2 130 | 2 180 | 2 160 | 12 48 | 5 15 |
| Black Prince..... | Upper Freeport..... | 75815-16..... | 2 400 | 2 510 | 2 460 | 8 76 | 1 91 |
| Hillman..... | Middle Kittanning..... | 34904-05..... | 2 910 | 2 970 | 2 940 | 10 27 | 1 59 |
| McDary..... | Upper Kittanning..... | 75729-29..... | 2 240 | 2 360 | 2 300 | 9 50 | 2 86 |
| Pancoat..... | Lower Freeport..... | 26527-31..... | 2 120 | 2 460 | 2 330 | 7 08 | 2 04 |
| Punxsutawney..... | do..... | 75754 53 55..... | 2 440 | 2 540 | 2 480 | 5 07 | 1 22 |
| Do..... | No. 4..... | 24685-57..... | 2 190 | 2 190 | 2 190 | 11 49 | 3 46 |
| Ramseytown..... | do..... | 12455, 57-58, 71..... | 2 330 | 2 720 | 2 510 | 12 08 | 2 28 |
| Sykesville..... | do..... | | | | | | |
| LACKAWANNA COUNTY. | | | | | | | |
| Dunmore..... | Sloan..... | 11440..... | | | +3,010 | 11 94 | .45 |
| Dunmore..... | Dunmore No. 2..... | | | | | | |
| LAWRENCE COUNTY. | | | | | | | |
| Wampum..... | Country bank..... | 34734-37..... | 2 130 | 2 190 | 2 160 | 5 85 | 2 02 |
| Do..... | Crescent..... | 34739-41..... | 2 410 | 2 540 | 2 460 | 9 03 | 2 28 |
| Do..... | Middle Kittanning..... | | | | | | |
| LUZERNE COUNTY. | | | | | | | |
| Pittston..... | Colliery No. 14..... | 11441..... | | | +3,010 | 6 03 | .58 |
| Pittston..... | Pittston..... | | | | | | |
| LYCOMING COUNTY. | | | | | | | |
| Balston..... | Red Run..... | 75520..... | | | 2 960 | 15 31 | .53 |
| Do..... | do..... | 75521..... | | | +3,000 | 17 11 | .57 |
| Do..... | Lower Kittanning (upper bench), Kittanning (lower bench), | | | | | | |
| CLERMONT COUNTY. | | | | | | | |
| Clermont..... | Lower Vein..... | 34932-84..... | 2 160 | 2 440 | 2 270 | 16 76 | 4 69 |
| Do..... | Upper Vein..... | 34932-86..... | 2 600 | 2 620 | 2 610 | 6 13 | 1 03 |
| Do..... | Coal Blossom..... | | | | | | |
| MERCER COUNTY. | | | | | | | |
| Drake..... | Sharon No. 5..... | 34897-08..... | 2 460 | 2 610 | 2 540 | 3 80 | .95 |
| Grove City..... | Diamond No. 3..... | 34810-17..... | 2 100 | 2 430 | 2 270 | 9 08 | 4 61 |
| Orcas Station..... | Pansey..... | 34894-06..... | 2 130 | 2 220 | 2 160 | 5 91 | 1 20 |
| Pardee..... | do..... | 34813-14..... | 2 040 | 2 280 | 2 170 | 7 80 | 3 95 |
| Stoneboro..... | Stoneboro No. 7..... | 34510-11..... | 2 660 | 2 160 | 2 100 | 7 27 | 3 26 |
| SOMERSET COUNTY. | | | | | | | |
| Bakersville..... | Wyand..... | 17689..... | | | +3,010 | 12 05 | .70 |
| Calmarbrook..... | Hitchew or Fiegle..... | 19651..... | | | +3,010 | 12 60 | 1 43 |
| Do..... | Loyal Hanna No. 6..... | 19648-49..... | | | +3,010 | 6 75 | .86 |
| Confidence..... | Immer..... | 19631..... | 3 010 | +3,010 | 2 190 | 8 44 | 2 43 |
| Edle..... | Levi Berkey..... | 17691..... | 1 1 | 1 | 2 680 | 11 23 | 1 57 |
| Engle..... | Upper Freeport..... | 19663..... | 1 1 | 1 | 2 460 | 8 94 | 1 99 |
| Elk Lick..... | Lower Freeport..... | 19663..... | 1 1 | 1 | 2 460 | 8 94 | 1 99 |
| Do..... | Grassy Run No. 1..... | 19664..... | 1 1 | 1 | 2 460 | 13 01 | 2 36 |
| Do..... | Four-foot..... | | | | | | |

TABLE I.—Softening temperatures of coal ash from coals of the United States—Continued.

| Locality. | Mine. | Coal bed. | Rank of coal. | Laboratory No. | Number of samples from mine. | Softening temperature ° F. | | | Average analysis of dry coal, percentage of— | |
|-------------------------|--|----------------------|---------------|----------------|------------------------------|----------------------------|----------|----------|--|----------|
| | | | | | | Lowest. | Highest. | Average. | Ash. | Sulphur. |
| I | | | | | | | | | | |
| PENNsylvania—continued. | Gillette. | Unamed. | B. (?) | 17602 | 1 | | | | 11.25 | 1.19 |
| | Hawes No. 3. | Lower Kittanning. | S. B. | 22541-45. | 5 | | | | 8.00 | 2.43 |
| | Hawes No. 3 (run-of-mine, car sample). |do..... | S. B. | 22547 | 1 | 2,180 | 2,440 | 2,370 | 9.07 | 2.71 |
| | Lenore. |do..... | S. B. | 15078-81 | 4 | 2,210 | 2,600 | 2,360 | 8.00 | 2.88 |
| | Onoda. | Upper Kittanning. | S. B. | 15073-76 | 4 | 2,120 | 2,460 | 2,280 | 11.06 | 2.81 |
| | Consolidation No. 112. | "Little Pittsburgh." | S. B. | 13981-82 | 2 | 2,350 | 2,420 | 2,360 | 8.13 | 1.70 |
| | Consolidation No. 113. | "Stoner." | S. B. | 13984 | 1 | | | 2,470 | 11.20 | 1.96 |
| | Eureka No. 39. | Upper Kittanning. | S. B. | 68200, 05, 14 | 3 | 1,980 | 2,360 | 2,190 | 9.20 | 2.24 |
| | Nava. | Upper Freeport. | S. B. | 17603 | 1 | | | 2,270 | 12.07 | 2.49 |
| | Samer & Shaffer. |do..... | S. B. | 17604 | 1 | | | 2,460 | 9.62 | 1.69 |
| Do. | Do. | S. B. | 17605 | 1 | | | +3,010 | 5.34 | 1.65 | |
| Stanley No. 3. | Lower Freeport. | S. B. | 20292-95, 98. | 5 | 2,320 | +3,010 | +3,010 | 6.79 | 1.44 | |
| Windber. | Lower Kittanning. | S. B. | 22534 | 1 | | | 2,460 | 10.25 | 2.16 | |
| Do. | Lochrie Arrow (run-of-mine, car sample). |do..... | S. B. | 17606 | 1 | | | 2,600 | 15.61 | 1.52 |
| Zimmerman. | Ralphdon No. 6. | Upper Freeport. | S. B. | | | | | | | |
| SULLIVAN COUNTY. | | | | | | | | | | |
| Bernice. | Randall & Shadd. | Lower Kittanning. | S. A. | 9652 | 1 | | | +3,010 | 12.09 | .84 |
| TIOGA COUNTY. | | | | | | | | | | |
| Antrim. | Anna S. | Bloss. | S. B. | 17453 | 1 | | | 2,500 | 12.78 | 2.92 |
| Landrus. | Bear Run. |do..... | S. B. | 17462 | 1 | | | 2,500 | 12.99 | 2.83 |
| Morris Run. | New. |do..... | S. B. | 17454 | 1 | | | 2,900 | 10.11 | 1.31 |
| Do. |do. | Morgan. | S. B. | 17451 | 1 | | | 2,450 | 9.39 | 1.76 |

| | | | | | | | | | |
|-----------------------|--|----|---|---|-------|-------|-------|-------|------|
| WASHINGTON COUNTY. | | | | | | | | | |
| Avella..... | Panobocott..... | B. | 13673-74..... | 2 | 2,040 | 2,110 | 2,080 | 6.96 | 2.12 |
| Baird..... | Schoenberger..... | B. | 11164-69..... | 6 | 2,310 | 2,520 | 2,440 | 6.98 | 1.08 |
| Finleyville..... | Cincinnati..... | B. | 17982-84..... | 3 | 2,360 | 2,660 | 2,420 | 6.98 | 1.30 |
| Greer Station..... | Renderson No. 1..... | B. | 27901-68..... | 5 | 2,010 | 2,220 | 2,120 | 8.12 | 2.36 |
| Monongahela City..... | Catsburg..... | B. | 11157-62..... | 6 | 2,140 | 2,610 | 2,420 | 6.98 | 1.17 |
| WESTMORELAND COUNTY. | | | | | | | | | |
| Greensburg..... | Jamison No. 4 (car sample, lump coal)..... | B. | 12853..... | 1 | | | 2,550 | 10.66 | 1.41 |
| Ligonier..... | John Dyer..... | B. | 17901..... | 1 | | | 2,450 | 11.90 | 1.99 |
| Lucesco..... | Lucesco..... | B. | 12688-96..... | 4 | 2,000 | 2,140 | 2,090 | 10.79 | 3.27 |
| SOUTH DAKOTA. | | | | | | | | | |
| HARDING COUNTY. | | | | | | | | | |
| Buffalo..... | Hilton..... | L. | 13221..... | 1 | | | 2,400 | 18.40 | 1.59 |
| Do..... | Mendenhall..... | L. | 13220..... | 1 | | | 2,140 | 17.36 | 1.04 |
| Ralph..... | Newcomb..... | L. | 15062..... | 1 | | | 2,140 | 13.98 | 1.46 |
| PERKINS COUNTY. | | | | | | | | | |
| Lodgepole..... | Nelson..... | L. | 14854..... | 1 | | | 2,180 | 15.73 | 1.14 |
| Strool..... | Phillips..... | L. | 12488..... | 1 | | | 2,130 | 13.44 | 2.02 |
| Do..... | Jones..... | L. | 12483..... | 1 | | | 2,280 | 13.72 | 3.65 |
| Do..... | Knudson..... | L. | 12454..... | 1 | | | 2,120 | 17.23 | 1.65 |
| TENNESSEE. | | | | | | | | | |
| ANDERSON COUNTY. | | | | | | | | | |
| Coal Creek..... | Black Diamond No. 1..... | B. | 21645-46..... | 4 | 2,180 | 2,400 | 2,310 | 5.14 | 1.30 |
| Do..... | Cross Mountain No. 1..... | B. | 13237-38, 64-65, 90, 92, 97, 71-73..... | 8 | 2,120 | 3,010 | 2,550 | 8.18 | 1.00 |
| Do..... | Fraterville..... | B. | 21636-38..... | 3 | 2,300 | 2,600 | 2,440 | 5.90 | 1.04 |
| Do..... | Klondike or No. 5..... | B. | 21640-43..... | 4 | 2,330 | 2,460 | 2,400 | 5.81 | 1.04 |
| Do..... | Middle Ridge..... | B. | 21680..... | 1 | | | 2,390 | 4.33 | .91 |
| Do..... | Smith..... | B. | 21627..... | 1 | | | 2,320 | 2.99 | .87 |
| Do..... | Tennessee..... | B. | 21632-34..... | 3 | 2,350 | 2,460 | 2,410 | 8.63 | 1.67 |
| Do..... | Thistle..... | B. | 21628-30..... | 3 | 2,210 | 2,320 | 2,260 | 8.29 | 1.60 |
| Devonia..... | Buffalo..... | B. | 77135-36..... | 2 | 2,240 | 2,260 | 2,150 | 4.11 | 3.14 |
| Oliver Springs..... | Big Mary..... | B. | 21427..... | 1 | 2,000 | 2,230 | 2,100 | 4.52 | 1.63 |
| Do..... | Coal Creek..... | B. | 21451-54..... | 4 | 2,260 | 2,310 | 2,280 | 5.10 | 1.68 |
| Do..... | Piedmont..... | B. | 32688-91..... | 4 | | | 2,260 | 5.10 | .88 |
| Petros..... | Windrock..... | B. | 32701..... | 1 | | | 2,350 | 8.75 | 2.72 |
| Rosedale..... | Windrock (Dean)..... | B. | 32701..... | 1 | | | 2,350 | 8.75 | 2.09 |
| Do..... | Jellison..... | B. | 32701..... | 1 | | | 2,370 | 9.15 | 2.43 |
| Do..... | Charley's Branch..... | B. | 32703..... | 1 | | | 2,370 | 9.15 | 2.43 |
| Do..... | New River..... | B. | 32703..... | 1 | | | 2,370 | 9.15 | 2.43 |
| Do..... | Do..... | B. | 32697..... | 1 | | | 2,320 | 8.18 | 2.32 |
| Stalinville..... | Main Mountain..... | B. | 32697..... | 1 | | | 2,320 | 8.18 | 2.32 |

TABLE I.—Softening temperatures of coal ash from coals of the United States—Continued.

| Locality. | Mine. | Coal bed. | Rank of coal. | Laboratory No. | Number of samples from mine. | Softening temperature ° F. | | | Average analysis of dry coal, percentage of— | |
|----------------------|----------------------|---------------------|---------------|----------------|------------------------------|----------------------------|----------|----------|--|----------|
| | | | | | | Lowest. | Highest. | Average. | Ash. | Sulphur. |
| 1 | | | | | | | | | | |
| TENNESSEE—Continued. | | | | | | | | | | |
| BLUESOE COUNTY. | | | | | | | | | | |
| Alpintley..... | Alpintley No. 6..... | Sewanee..... | B. | 2259-60..... | 2 | 2,370 | 2,440 | 2,410 | 8.47 | 1.06 |
| Harbert..... | do. (?)..... | do. (?)..... | B. | 10915..... | 1 | | | 2,600 | 8.10 | .88 |
| Do..... | do..... | do. (?)..... | B. | 10916..... | 1 | | | 2,600 | 6.60 | .84 |
| Lifton..... | Hale..... | Angel..... | B. | 12584..... | 1 | | | 2,160 | 5.80 | 1.94 |
| Do..... | New Opening..... | Richland..... | B. | 10799..... | 1 | | | 2,760 | 9.87 | .88 |
| Do..... | Prospect..... | Unnamed..... | B. | 10803..... | 1 | | | 2,910 | 8.06 | .62 |
| Do..... | Thurman..... | Richland..... | B. | 10801..... | 1 | | | 2,610 | 7.52 | .88 |
| Pileville..... | McFarland..... | Morgan Springs..... | B. | 10731..... | 1 | | | 2,210 | 10.87 | 3.61 |
| Do..... | Vaughn prospect..... | Sewanee (?)..... | B. | 11049..... | 1 | | | 2,720 | 8.96 | .97 |
| Do..... | do..... | do..... | B. | 11060..... | 1 | | | 2,640 | 6.89 | .84 |
| CAMPBELL COUNTY. | | | | | | | | | | |
| Anthras..... | Anthras..... | Jellico..... | B. | 21853-55..... | 2 | 2,400 | 2,480 | 2,420 | 3.15 | .85 |
| Black..... | Monarch..... | Monarch..... | B. | 22023-31..... | 4 | 2,060 | 2,470 | 2,220 | 11.29 | 2.77 |
| Do..... | Fee Wee..... | Red Ash..... | B. | 22025-26..... | 2 | 2,940 | 2,970 | 2,960 | 5.72 | 1.07 |
| Caryville..... | Bear Wallow..... | Coal Creek..... | B. | 21672-76..... | 3 | 2,800 | 2,610 | 2,570 | 4.39 | .94 |
| Do..... | do..... | do..... | B. | 22660-62..... | 3 | 2,300 | 2,660 | 2,450 | 7.13 | 1.28 |
| Do..... | Caryville..... | Red Ash..... | B. | 21659-62..... | 4 | 2,220 | 2,480 | 2,370 | 6.08 | 1.24 |
| Do..... | Fee Wee..... | Lower Dean..... | B. | 21677-78..... | 2 | 2,240 | 2,480 | 2,340 | 3.69 | .72 |
| Do..... | Red Ash..... | Red Ash..... | B. | 21669-71..... | 3 | 2,400 | 2,450 | 2,420 | 6.48 | 1.02 |
| Do..... | Sum..... | do..... | B. | 21664-67..... | 4 | 2,400 | 2,560 | 2,510 | 6.24 | 1.18 |
| Do..... | Chaska..... | Rich Mountain..... | B. | 21813..... | 1 | | | 2,220 | 1.53 | .90 |
| Coal Creek..... | Cambria..... | Coal Creek..... | B. | 21650-62..... | 3 | 2,350 | 2,500 | 2,400 | 3.95 | .83 |
| Cottula..... | Jellico..... | Jellico..... | B. | 22685..... | 1 | | | 2,380 | 5.07 | 1.41 |
| Do..... | Southern..... | Jordan..... | B. | 21687-88..... | 2 | 2,080 | 2,340 | 2,210 | 3.68 | 1.22 |
| Do..... | Wynn..... | Rich Mountain..... | B. | 21690-92..... | 3 | 2,440 | 2,460 | 2,450 | 2.10 | .86 |
| Elk Valley..... | Elkhart..... | Blue Gum..... | B. | 22016..... | 1 | | | 2,000 | 2.69 | 1.37 |
| Do..... | Elk Valley..... | do..... | B. | 22018-19..... | 2 | 2,080 | 2,090 | 2,070 | 2.39 | 1.03 |
| Do..... | Perkins Branch..... | do..... | B. | 22012-13..... | 2 | 1,960 | 2,130 | 2,040 | 2.04 | 1.04 |
| Do..... | Split..... | do..... | B. | 32641-42..... | 2 | 2,270 | 2,360 | 2,320 | 8.63 | 1.68 |
| Habersham..... | Old Davis Creek..... | Jellico..... | B. | 21820..... | 1 | | | 2,390 | 6.83 | 1.50 |
| Do..... | Remy..... | Rich Mountain..... | B. | 21802, 04..... | 2 | 2,440 | 2,740 | 2,590 | 2.13 | .72 |

| | | | | | | | | |
|-------------------|------------------------|--------------------|----------------------|--------|---------|--------|-------|------|
| Do. | Rich Mountain. | do. | 2190, 06-09. | 2, 160 | 2, 330 | 2, 230 | 4.65 | 2.10 |
| Jellco. | Black. | Blue Gem. | 2200. | 1, 900 | 2, 140 | 2, 080 | 4.15 | 1.16 |
| Do. | Blue Gem. | do. | 21924-26. | 2, 040 | 2, 130 | 2, 080 | 2.14 | 1.02 |
| Do. | Braughton. | do. | 21896-97. | 2, 040 | 2, 130 | 2, 080 | 2.12 | .98 |
| Do. | Evans. | do. | 22001-02. | 2, 080 | 2, 120 | 2, 100 | 1.98 | 1.88 |
| Do. | Indian Mountain No. 3. | Jellco. | 21831-34. | 2, 080 | 2, 270 | 2, 160 | 3.14 | 1.33 |
| Do. | Jameson Blue Gem. | Blue Gem. | 21999. | 2, 160 | 2, 810 | 2, 690 | 2.49 | 1.96 |
| Kilguth. | Big Creek. | Kent. | 32967-69. | 2, 040 | 2, 360 | 2, 060 | 3.06 | 1.00 |
| Do. | Jordan. | do. | 21998-97. | 2, 040 | 2, 360 | 2, 540 | 3.28 | .79 |
| Do. | Gem No. 2. | do. | 21994-96. | 2, 040 | 2, 660 | 2, 500 | 3.13 | 1.80 |
| Do. | Gem No. 4. | do. | 21821-23. | 2, 120 | 2, 480 | 2, 380 | 4.76 | 1.01 |
| Kimberly. | Kimberly. | Rich Mountain. | 21969-702, 17555-57. | 2, 260 | 2, 260 | 2, 200 | 6.43 | 1.13 |
| Lafolette. | Rex No. 1. | do. | 17544, 52-53. | 2, 080 | 2, 450 | 2, 260 | 6.74 | 1.13 |
| Do. | Rex No. 2. | Jellco. | 21827-29. | 2, 070 | 2, 160 | 2, 100 | 4.61 | 1.80 |
| Do. | Meadow Branch. | do. | 32952-54. | 2, 220 | 2, 450 | 2, 300 | 5.07 | 1.13 |
| Do. | Red Moon. | do. | 22004-06. | 2, 220 | 2, 440 | 2, 310 | 6.13 | 5.21 |
| Newcomb. | Marion-Anna. | do. | 21923-25. | 1, 900 | 2, 130 | 2, 110 | 7.06 | 2.27 |
| Do. | Italian Blue Gem. | Blue Gem. | 32944-46. | 1, 900 | 1, 990 | 1, 990 | 1.83 | .96 |
| Do. | Regina Slope. | Swamp Angel. | 21931-32. | 1, 970 | 1, 990 | 1, 980 | 5.10 | .87 |
| Do. | Washington Blue Gem. | Blue Gem. | 22008-10. | 2, 270 | 2, 390 | 2, 330 | 2.11 | .96 |
| Do. | Zechini. | Jellco. | 21839-40. | 1, 960 | 2, 040 | 2, 300 | 2.00 | .88 |
| Do. | Falls Branch. | do. | 21871-72. | 2, 010 | 2, 420 | 2, 220 | 7.72 | 2.55 |
| Do. | Powhatan. | Blue Gem. | 32964-65. | 2, 170 | 2, 400 | 2, 730 | 10.25 | 1.04 |
| Do. | Pioneer. | Unnamed. | 22021-23. | 2, 170 | 2, 400 | 2, 290 | 12.02 | 2.29 |
| Do. | Pioneer Jellco. | Rock Springs. | 21964. | 2, 170 | 2, 320 | 2, 340 | 3.96 | .73 |
| Turley. | Disney. | Frozen Head (?). | 21955-66. | 1, 960 | 2, 380 | 2, 240 | 4.24 | 1.19 |
| Vaspar. | Vaspar. | Coal Creek. | 32949-50. | 2, 010 | 2, 180 | 2, 210 | 6.55 | 1.47 |
| Do. | Westbourne No. 2. | Jellco. | 32959-53. | 2, 010 | 2, 180 | 2, 100 | 2.71 | .99 |
| Do. | White Oak. | Log Mountain. | | | | | | |
| CLAIBORNE COUNTY. | | | | | | | | |
| Anthrac. | Tackett Creek. | Jellco. | 32975-77. | 2, 280 | 2, 620 | 2, 490 | 3.08 | .98 |
| Bosworth (Ky.). | Yellow Creek No. 3. | Billygoat. | 22067. | 2, 460 | 2, 530 | 2, 600 | 2.28 | 1.13 |
| Bryson. | Bryson Mountain No. 1. | Mingo. | 22106-08. | 2, 460 | 2, 530 | 2, 480 | 4.95 | 1.82 |
| Do. | Bryson Mountain No. 2. | Poplar Lick. | 21960-61. | 2, 040 | 2, 460 | 2, 360 | 7.77 | 2.70 |
| Clairfield. | King Mountain. | Jellco. | 21867-68. | 2, 110 | 2, 250 | 2, 180 | 4.72 | 1.54 |
| Do. | Standard. | do. | 21962-65. | 2, 110 | 2, 250 | 2, 250 | 3.08 | 1.52 |
| Do. | Buffalo. | do. | 21869-70. | 2, 110 | 2, 250 | 2, 310 | 2.53 | 1.19 |
| Egan. | Clear Fork. | do. | 32979-82. | 2, 080 | 2, 570 | 2, 660 | 4.16 | .76 |
| Fonde. | Fork Ridge No. 1. | do. | 22114-17. | 2, 080 | 2, 570 | 2, 380 | 6.80 | 1.22 |
| Fork Ridge. | Fork Ridge No. 4. | do. | 22110-12. | 2, 080 | 2, 570 | 2, 790 | 6.86 | .83 |
| Do. | Fork Ridge No. 3. | Poplar Lick. | 32957-58. | 2, 440 | 2, 680 | 2, 560 | 7.89 | 1.96 |
| Do. | Jack Rock No. 3. | Mingo. | 83460. | 2, 460 | 2, 500 | 2, 910 | 4.80 | 1.06 |
| Do. | Mingo No. 4. | Poplar Lick. | 32961-63. | 2, 460 | 2, 500 | 2, 490 | 8.45 | 1.03 |
| Do. | Nicholson No. 2. | Mingo. | 22067-68, 22126. | 2, 300 | 2, 440 | 2, 370 | 9.56 | 2.87 |
| Do. | Mingo No. 5. | Poplar Lick. | 22123-25. | 2, 460 | 2, 470 | 2, 470 | 4.02 | 1.29 |
| Do. | Reliance No. 1. | Mingo. | 22120-21. | 2, 080 | 2, 400 | 2, 240 | 3.96 | 1.09 |
| Do. | Reliance No. 2. | do. | 22119. | 2, 080 | 2, 400 | 2, 380 | 10.34 | 1.26 |
| Do. | Sandstone Parting. | Sandstone Parting. | 22100-03. | 2, 850 | +3, 010 | 2, 930 | 9.54 | .96 |
| Do. | Sterling. | Poplar Lick. | 21857-60. | 2, 310 | 2, 600 | 2, 400 | 3.44 | 1.23 |
| Pruden. | High Cliff. | Mingo. | 21944-47. | 2, 060 | 2, 500 | 2, 360 | 4.73 | 1.45 |
| Do. | Pruden. | do. | 32964. | 2, 060 | 2, 500 | 2, 330 | 2.15 | .90 |
| Rogerton. | Rogerton. | Jellco. | | | | | | |

TABLE I.—Softening temperatures of coal ash from coals of the United States—Continued.

| Locality. | Mine. | Coal bed. | Rank of coal. | Laboratory No. | Number of samples from mine. | Softening temperature ° F. | | | Average analysis of dry coal, percentage of— | |
|----------------------|-----------------------|-----------------|---------------|----------------|------------------------------|----------------------------|----------|----------|--|----------|
| | | | | | | Lowest. | Highest. | Average. | Ash. | Sulphur. |
| 1 | | | | | | | | | | |
| TENNESSEE—Continued. | | | | | | | | | | |
| CUMBERLAND COUNTY. | | | | | | | | | | |
| Crab Orchard. | Haley Mountain No. 1. | Sewanee. | B. | 33471-72. | 2 | 2,500 | 2,610 | 2,660 | 7.50 | 0.70 |
| Litton. | Hale Prospect. | Morgan Springs. | B. | 10802. | 1 | | | 2,310 | 11.23 | 3.31 |
| Do. | do. | Sewanee (?) | B. | 10800. | 1 | | | 2,850 | 7.81 | .99 |
| Orone. | For Den. | Sewanee. | B. | 33474. | 1 | | | 2,570 | 9.35 | .62 |
| Do. | Nelson. | Nelson. | B. | 33475. | 1 | | | 2,640 | 6.77 | .66 |
| FENTRESS COUNTY. | | | | | | | | | | |
| Davidson. | Davidson. | Bon Air No. 2. | B. | 20987-88. | 2 | 2,080 | 2,080 | 2,070 | 10.54 | 3.14 |
| Do. | Highland No. 2. | do. | B. | 20982-84, 86. | 4 | 2,080 | 2,190 | 2,120 | 9.91 | 3.25 |
| Gerrit. | East Laurel. | White Oak. | B. | 32789. | 1 | | | 2,360 | 6.53 | 1.26 |
| Wildor. | Big Laurel No. 2. | Wildor. | B. | 33476. | 1 | | | 2,260 | 11.18 | 3.24 |
| Do. | Wildor No. 3. | Bon Air No. 2. | B. | 24845-49. | 5 | 2,210 | 2,350 | 2,240 | 10.28 | 2.69 |
| Zenith. | White Oak. | White Oak. | B. | 32798-800. | 3 | 2,460 | 2,550 | 2,500 | 5.60 | .42 |
| GRUNDY COUNTY. | | | | | | | | | | |
| Coalmont. | Coalmont E. | Sewanee. | B. | 22384. | 1 | | | 2,530 | 9.55 | 1.20 |
| Do. | Coalmont I. | do. | B. | 22383. | 1 | | | 2,630 | 12.61 | .90 |
| Do. | Coalmont New A. | do. | B. | 22385. | 1 | | | 2,460 | 13.19 | 1.21 |
| Do. | Coalmont Old A. | do. | B. | 22386. | 1 | | | 2,530 | 10.06 | 1.06 |
| Do. | Coalmont S. Old Hill. | do. | B. | 22374. | 1 | | | 2,890 | 9.40 | .85 |
| Do. | Dixie No. 4. | do. | B. | 32584. | 1 | | | 2,750 | 9.78 | .62 |
| Do. | Flanagan. | do. | B. | 22373. | 1 | | | 2,420 | 9.15 | .74 |
| Do. | Mill Creek prospects. | do. | B. | 22379, 81. | 2 | 2,210 | 2,690 | 2,450 | 9.54 | 1.84 |
| Freemont. | Tracy City No. 3. | do. | B. | 32582. | 1 | | | 2,640 | 9.06 | .65 |
| Do. | Tracy City No. 4. | do. | B. | 32583. | 1 | | | 2,400 | 8.33 | .62 |
| Palmer. | Tennessee No. 8. | do. | B. | 32586. | 1 | | | 2,610 | 8.63 | .71 |
| Do. | Tennessee No. 16. | do. | B. | 32587. | 1 | | | 2,690 | 9.45 | .60 |
| Tracy City. | East Fork. | do. | B. | 22391. | 1 | | | 2,300 | 10.29 | 2.05 |

| | | | | | | | |
|-----------------------|-------------------------|----------------|---|-------|-------|-------|------|
| Do..... | East Staub..... | 22383..... | 1 | | 2,500 | 11.22 | .62 |
| Do..... | Old Clouse Hill..... | 22372..... | 1 | | 2,590 | 9.03 | .80 |
| Do..... | Do..... | 22371..... | 1 | | 2,390 | 9.18 | .57 |
| Do..... | Old Staub..... | 22404..... | 1 | | 2,930 | 8.30 | .58 |
| Do..... | Do..... | 22380..... | 1 | | 1,970 | 11.97 | 3.81 |
| Do..... | Reed Hill No. 2..... | 22403..... | 1 | | 2,690 | 8.07 | .66 |
| Do..... | West Ramsey..... | | | | | | |
| HAMILTON COUNTY. | | | | | | | |
| Daisy..... | Abel..... | 22249..... | 1 | | 2,100 | 8.95 | 1.53 |
| Montlake..... | Montlake No. 10..... | 22243-45..... | 3 | 1,980 | 2,280 | 11.42 | 3.14 |
| Do..... | Unamed..... | 22247..... | 1 | | 2,620 | 11.32 | 1.66 |
| Rathbun..... | Soddy..... | 22248..... | 1 | | 2,420 | 5.17 | 1.67 |
| Do..... | Do..... | 22211-12..... | 2 | 2,570 | 3,010 | 6.33 | .71 |
| Do..... | Old Bunker..... | 22190..... | 1 | | 2,790 | 4.13 | .71 |
| Do..... | Do..... | 22187-88..... | 2 | 2,720 | 2,980 | 10.48 | 1.43 |
| Do..... | Sheephead..... | 22186..... | 2 | 2,180 | 2,280 | 7.79 | 1.28 |
| Do..... | Soddy No. 1..... | | 3 | | | | |
| MARION COUNTY. | | | | | | | |
| Orme..... | Battle Creek No. 3..... | 22231..... | 1 | | 2,620 | 8.77 | 1.09 |
| Do..... | Battle Creek No. 4..... | 22230, 33..... | 2 | 2,400 | 2,410 | 10.59 | 1.04 |
| Troy City..... | Long Ridge..... | 22270..... | 1 | | 2,530 | 8.86 | 1.19 |
| Do..... | Pryor Ridge No. 1..... | 22286-401..... | 4 | 1,980 | 2,670 | 10.64 | 2.30 |
| Whiteside..... | Castle Rock..... | 22284..... | 1 | | 2,300 | 10.31 | 2.41 |
| Do..... | Do..... | 22287..... | 1 | | 2,210 | 11.24 | 2.65 |
| Do..... | Clements prospect..... | 22285..... | 1 | | 2,600 | 8.70 | 1.29 |
| Do..... | New Etina No. 1..... | 22287..... | 1 | | 2,400 | 8.56 | 1.36 |
| Do..... | Do..... | 22281..... | 1 | | 2,140 | 8.63 | .76 |
| Do..... | Old Etina..... | 22287..... | 1 | | 2,360 | 8.53 | .60 |
| Whitwell..... | Sewares..... | 22287..... | 1 | | 2,360 | 8.53 | .60 |
| Do..... | Whitwell No. 5..... | 22286-70..... | 3 | 2,400 | 2,490 | 8.68 | .93 |
| MORGAN COUNTY. | | | | | | | |
| Blue Gem Siding..... | Bottomlee..... | 21150..... | 1 | | 2,110 | 4.18 | 1.66 |
| Do..... | Coal Cut..... | 21145..... | 1 | | 2,460 | 4.35 | 1.37 |
| Catoosa..... | Catoosa No. 1..... | 32743-44..... | 2 | 2,210 | 2,360 | 5.36 | 3.37 |
| Do..... | Unamed..... | 32756..... | 1 | | 2,940 | 7.21 | 1.45 |
| Do..... | Cow Ford No. 4..... | 32757..... | 1 | | 2,960 | 5.03 | 1.17 |
| Do..... | Do..... | 25556-58..... | 3 | 2,240 | 2,280 | 7.11 | 2.99 |
| Do..... | Flatrock No. 3..... | 32746-47..... | 2 | 2,270 | 2,380 | 8.50 | 2.51 |
| Do..... | Catoosa..... | 32758..... | 3 | | 2,370 | 8.06 | 2.99 |
| Do..... | Unamed..... | | 1 | | 2,470 | 1.87 | 1.87 |
| Do..... | Red Eye..... | 21151..... | 1 | | 2,600 | 5.54 | .69 |
| Christmas Siding..... | Grassy Ridge..... | 21152..... | 1 | | 2,990 | 5.54 | .90 |
| Do..... | Hooper..... | 21153..... | 1 | | 2,130 | 7.97 | 4.13 |
| Do..... | Hooper (?)..... | 21104-05..... | 2 | 2,080 | 2,170 | 4.14 | 4.13 |
| Coalfield..... | Coal Creek..... | 21107-08..... | 3 | 2,150 | 2,190 | 3.28 | 4.13 |
| Do..... | Do..... | 21147..... | 1 | | 2,390 | 4.65 | 1.47 |
| Do..... | Bowing..... | 21099..... | 1 | | 2,200 | 4.89 | 3.40 |
| Do..... | Davis..... | 21146..... | 1 | | 2,140 | 11.62 | 4.56 |
| Do..... | Slope..... | 21146..... | 1 | | 2,230 | 2.44 | 1.85 |
| Do..... | Coal Creek (?)..... | | 1 | | 2,360 | 4.83 | 1.32 |
| Do..... | Blue Gem..... | | 1 | | | | |
| Do..... | Hooper..... | | 1 | | | | |
| Do..... | Unamed..... | | 1 | | | | |
| Launcing..... | Summers..... | | 1 | | | | |

TABLE I.—Softening temperatures of coal ash from coals of the United States—Continued.

| Locality. | Mine. | Coal bed. | Rank of coal. | Laboratory No. | Number of samples from mine. | Softening temperature ° F. | | | Average analysis of dry coal, percentage of— | |
|--------------------------|-----------------------|---------------------|---------------|----------------|------------------------------|----------------------------|----------|----------|--|----------|
| | | | | | | Lowest. | Highest. | Average. | Ash. | Sulphur. |
| 1 | | | | | | | | | | |
| TENNESSEE—Continued. | | | | | | | | | | |
| MORGAN COUNTY—continued. | | | | | | | | | | |
| Nemo..... | Lower Seam..... | do..... | B. | 32760..... | 1 | | | 3,000 | 15.64 | 0.86 |
| Do..... | Upper Seam..... | do..... | B. | 32759..... | 1 | | | 2,540 | 9.92 | 1.89 |
| Do..... | Catoosa..... | do..... | B. | 21428..... | 1 | | | 2,430 | 7.49 | 1.33 |
| Oakdale..... | Babatchle..... | Walden Ridge..... | B. | 32755..... | 1 | | | 2,620 | 16.50 | 1.85 |
| Oliver Springs..... | Big Mountain..... | Unnamed..... | B. | 21424..... | 1 | | | 2,040 | 6.75 | 4.24 |
| Do..... | Jackson..... | Coal Creek..... | B. | 21426..... | 1 | | | 2,270 | 2.77 | 1.01 |
| Do..... | Levan..... | Coal Creek (?)..... | B. | 21420..... | 1 | | | 2,290 | 3.57 | 1.39 |
| Do..... | New Eagle No. 1..... | Old Eagle..... | B. | 32690..... | 1 | | | 2,380 | 6.20 | 3.54 |
| Do..... | Old Mount Carbon..... | Poplar Creek..... | B. | 21421..... | 1 | | | 2,200 | 3.29 | 1.21 |
| Do..... | Poplar Creek..... | Coal Creek..... | B. | 21423..... | 1 | | | 2,040 | 7.16 | 3.97 |
| Do..... | Prudential..... | do..... | B. | 21412-14..... | 3 | 1,990 | 2,010 | 2,000 | 9.51 | 4.61 |
| Do..... | Reed..... | Unnamed..... | B. | 21425..... | 1 | | | 2,300 | 4.25 | 7.72 |
| Do..... | Richards..... | Coal Creek..... | B. | 21416-17..... | 2 | 2,050 | 2,060 | 2,060 | 9.63 | 6.50 |
| Do..... | Signal Mountain..... | do..... | B. | 21422..... | 1 | | | 2,080 | 6.08 | 5.31 |
| Do..... | Union..... | Poplar Creek..... | B. | 32702..... | 1 | | | 2,230 | 5.24 | 1.79 |
| Do..... | Williams..... | Coal Creek..... | B. | 21419..... | 1 | | | 2,490 | 6.41 | 1.13 |
| Petros..... | Big Brushy..... | Jellico..... | B. | 32693-95..... | 3 | 2,520 | 2,520 | 2,520 | 9.51 | 3.75 |
| Do..... | Frozen Head..... | Frozen Head..... | B. | 21083..... | 1 | | | 2,580 | 6.92 | 9.92 |
| Do..... | Petros No. 5..... | Jellico..... | B. | 21088-90..... | 3 | 2,330 | 2,430 | 2,370 | 7.36 | 2.55 |
| Do..... | State No. 3..... | do..... | B. | 21092-95..... | 4 | 2,130 | 2,490 | 2,300 | 8.30 | 2.72 |
| Stephens..... | Layman..... | Unnamed..... | B. | 21149..... | 1 | | | 2,070 | 7.26 | 2.80 |
| Do..... | Little Brushy..... | Jellico..... | B. | 21084-86..... | 3 | 2,130 | 2,350 | 2,210 | 5.14 | 2.68 |
| OVERTON COUNTY. | | | | | | | | | | |
| Davidson..... | Briar Hill No. 1..... | Wilder..... | B. | 32468-69..... | 3 | 2,270 | 2,360 | 2,330 | 8.78 | 2.54 |
| Do..... | Briar Hill No. 2..... | do..... | B. | 32465-66..... | 3 | 2,270 | 2,270 | 2,270 | 9.19 | 2.79 |
| Do..... | Buckeye..... | do..... | B. | 32441-43..... | 3 | 2,210 | 2,370 | 2,270 | 10.01 | 3.09 |
| Do..... | Goch..... | do..... | B. | 32445-47..... | 3 | 2,220 | 2,310 | 2,260 | 11.20 | 3.00 |
| Do..... | Tunnel..... | do..... | B. | 32437-39..... | 3 | 2,300 | 2,330 | 2,310 | 9.61 | 2.63 |

| | | | | | | | | | | |
|----------------------|----------------------|--------------------|----|-------------------------|---|-------|-------|-------|-------|-------|
| Highland Junction. | Overtown..... | Bon Air No. 2..... | B. | 20978-80..... | 3 | 2,120 | 2,300 | 2,190 | 10.57 | 3.15 |
| Lovejoy..... | Bill's Branch..... | Wilder..... | B. | 33454-55..... | 2 | 2,010 | 2,000 | 2,040 | 12.89 | 6.93 |
| Obey City..... | Bon Air No. 2..... | Bon Air No. 2..... | B. | 20991..... | 1 | | | 2,180 | 12.90 | 3.60 |
| Do..... | Do..... | Do..... | B. | 20992..... | 1 | | | 2,160 | 10.29 | 3.40 |
| Wilder..... | Wilder No. 3..... | Wilder..... | B. | 33449-52..... | 4 | 2,060 | 2,230 | 2,180 | 10.52 | 3.50 |
| FUTNAM COUNTY. | | | | | | | | | | |
| Monterey..... | Monterey..... | Bon Air No. 2..... | B. | 20990..... | 1 | | | 2,100 | 10.07 | 3.55 |
| REHA COUNTY. | | | | | | | | | | |
| Dayton..... | New Prospect..... | Nelson..... | B. | 10895-99..... | 4 | 2,460 | 2,600 | 2,600 | 18.46 | .49 |
| Do..... | North Pole..... | Richard..... | B. | 10898-90, 92-93..... | 4 | 2,250 | 2,490 | 2,380 | 14.21 | 1.40 |
| Graysville..... | Montague No. 1..... | Nelson..... | B. | 22159, 65..... | 2 | 2,010 | 2,300 | 2,150 | 11.79 | 2.24 |
| Do..... | Montague No. 3..... | Sewanee..... | B. | 22180-83..... | 4 | 2,070 | 2,710 | 2,400 | 12.35 | 1.28 |
| Do..... | Montague No. 6..... | Nelson..... | B. | 22158..... | 1 | | | 2,280 | 23.95 | .59 |
| ROANE COUNTY. | | | | | | | | | | |
| De Armond..... | Wilcox..... | Rockwood..... | B. | 32764..... | 1 | | | 2,140 | 8.55 | .67 |
| Emory Gap..... | Emory Gap..... | do..... | B. | 32762..... | 1 | | | 2,000 | 8.83 | .55 |
| Hartman..... | Blizard..... | do..... | B. | 32763..... | 1 | | | 2,440 | 6.80 | .50 |
| Do..... | Walden Ridge..... | Walden Ridge..... | B. | 21082..... | 1 | | | 2,790 | 8.85 | .80 |
| McLean..... | McLean..... | Sewanee..... | B. | 21024..... | 1 | | | 2,400 | 9.81 | .81 |
| Rockwood..... | Rockwood..... | do..... | B. | 18995-97, 21018-18..... | 6 | 2,250 | 2,460 | 2,240 | 9.50 | .54 |
| SCOTT COUNTY. | | | | | | | | | | |
| Bear Creek Junction. | Phillips..... | No. 4..... | B. | 21154-55..... | 3 | 2,190 | 2,590 | 2,360 | 6.84 | 2.02 |
| Do..... | Wilson..... | do..... | B. | 21218-20..... | 3 | 2,050 | 2,000 | 2,050 | 11.82 | 5.22 |
| Helenwood..... | Fiat Creek..... | Glen Mary..... | B. | 32750..... | 1 | | | 2,220 | 8.45 | 3.54 |
| Do..... | Helenwood..... | do..... | B. | 32740-41..... | 2 | 2,300 | 2,380 | 2,270 | 8.61 | 4.65 |
| Do..... | Wagon..... | do..... | B. | 32749..... | 1 | | | 2,300 | 13.61 | 4.10 |
| Huntsville..... | Cross..... | Paint Rock..... | B. | 21223..... | 1 | | | 2,050 | 9.05 | 3.78 |
| Jake's Tank..... | Jake's Branch..... | do..... | B. | 21226..... | 1 | | | 2,680 | 8.83 | .82 |
| Do..... | Opossum Jaw..... | do..... | B. | 21223..... | 1 | | | 2,430 | 5.57 | 3.47 |
| Newland..... | Arch Mountain..... | Jellico..... | B. | 21273..... | 1 | | | 2,490 | 9.35 | 1.92 |
| Do..... | Arch Mountain..... | Big Mary..... | B. | 32795-96..... | 2 | 2,460 | 2,520 | 2,410 | 7.65 | 1.76 |
| Do..... | Pemberton No. 1..... | Jellico..... | B. | 32788..... | 1 | | | 1,990 | 3.85 | 3.85 |
| Robbins..... | Clay No. 1..... | Blue Gem (?)..... | B. | 21228..... | 1 | | | 2,640 | 4.21 | .92 |
| Do..... | Hughett..... | Mud Slip..... | B. | 21314..... | 1 | | | 2,580 | 4.21 | 1.06 |
| Do..... | do..... | Blue Gem (?)..... | B. | 21299..... | 1 | | | 2,210 | 4.26 | 1.25 |
| Do..... | Long prospect..... | do..... | B. | 21316..... | 1 | | | 2,500 | 5.71 | 1.08 |
| Do..... | Newman prospect..... | do..... | B. | 21315..... | 1 | | | 2,060 | 10.44 | 5.35 |
| Do..... | Southern No. 8..... | Glen Mary..... | B. | 32751..... | 1 | | | 2,580 | 4.43 | .80 |
| Stanley Junction. | Keaton..... | Paint Rock..... | B. | 32791..... | 1 | | | 2,220 | 4.58 | .78 |
| Do..... | Morning Glory..... | do..... | B. | 32790..... | 1 | | | 2,360 | 3.44 | .84 |
| Do..... | Paint Rock..... | do..... | B. | 21225..... | 1 | | | 2,400 | 5.08 | .65 |
| Do..... | Pumpkin Hollow..... | do..... | B. | 21224, 32792-93..... | 3 | 2,330 | 2,580 | 2,420 | 4.18 | .84 |
| Do..... | Saxon..... | do..... | B. | 32787..... | 1 | | | 2,610 | 6.54 | .72 |
| Do..... | Terry No. 5..... | do..... | B. | | 1 | | | | | |

TABLE I.—Softening temperatures of coal ash from coals of the United States—Continued.

| Locality. | Mine. | Coal bed. | Rank of coal. | Laboratory No. | Number of samples from mine. | Softening temperature ° F. | | | Average analysis of dry coal, percentage of— | |
|----------------------|-----------------------|----------------------------|---------------|-------------------------|------------------------------|----------------------------|----------|----------|--|----------|
| | | | | | | Lowest. | Highest. | Average. | Ash. | Sulphur. |
| 1 | | | | | | | | | | |
| TENNESSEE—Continued. | | | | | | | | | | |
| SEQUATCHIE COUNTY. | | | | | | | | | | |
| Dunlap..... | Douglas No. 2..... | Sewanee..... | B. | 22239-41..... | 3 | 2,330 | 2,460 | 2,390 | 10.12 | 1.35 |
| WHITE COUNTY. | | | | | | | | | | |
| Bon Air..... | Bon Air..... | Bon Air (lower bench)..... | B. | 22345-48..... | 4 | 2,300 | 2,420 | 2,350 | 7.87 | 2.87 |
| Clifty..... | Clifty No. 1..... | Sewanee (T)..... | B. | 12622-26, 22382-91..... | 8 | 1,990 | 2,090 | 2,040 | 13.15 | 4.24 |
| Ravenscroft..... | Ravenscroft..... | Bon Air (T)..... | B. | 22383-96..... | 4 | 2,050 | 2,210 | 2,140 | 10.78 | 4.19 |
| TEXAS. | | | | | | | | | | |
| WEBB COUNTY. | | | | | | | | | | |
| Dolores..... | Dolores..... | San Pedro..... | L. | 20022..... | 1 | | | 2,330 | 12.75 | 2.04 |
| Do..... | do..... | Santo Tomas..... | L. | 20021..... | 1 | | | 2,590 | 19.89 | 2.17 |
| Santo Tomas..... | do..... | do..... | L. | 20024..... | 1 | | | 2,570 | 18.52 | 1.78 |
| UTAH. | | | | | | | | | | |
| CARBON COUNTY. | | | | | | | | | | |
| Carbon..... | Panther..... | Castlegate..... | B. | 19881..... | 1 | | | 2,110 | 5.59 | .80 |
| Castlegate..... | Cameron No. 1..... | "No. 1"..... | B. | 19880..... | 1 | | | 2,220 | 6.40 | .41 |
| Do..... | Cameron No. 2..... | "No. 2"..... | B. | 19879..... | 1 | | | 2,170 | 6.78 | .48 |
| Do..... | Castlegate No. 1..... | "A"..... | B. | 19847..... | 1 | | | 2,100 | 6.46 | .51 |
| Do..... | do..... | "C"..... | B. | 19846..... | 1 | | | 2,080 | 5.02 | .49 |
| Do..... | do..... | "D"..... | B. | 19844..... | 1 | | | 2,130 | 7.37 | .63 |
| Do..... | Castlegate No. 2..... | "B"..... | B. | 19845..... | 1 | | | 2,210 | 9.44 | .68 |
| Do..... | do..... | "D"..... | B. | 19843..... | 1 | | | 2,150 | 5.51 | .34 |
| Helper..... | Prospect..... | Spring Canyon No. 1..... | B. | 19837..... | 1 | | | 2,360 | 7.98 | .80 |
| Do..... | do..... | do..... | B. | 19838..... | 1 | | | 2,280 | 9.09 | .72 |
| Hiawatha..... | Hiawatha No. 1..... | Hiawatha..... | B. | 10044-46..... | 2 | 2,140 | 2,430 | 2,280 | 6.16 | .74 |
| Do..... | do..... | do..... | B. | 10047-48..... | 2 | 2,220 | 2,220 | 2,220 | 7.50 | .58 |

| | | | | | | | | |
|-----------------|------------------------|----------------------|--------------------|---|--------|---------|-------|------|
| Kenilworth. | Aberdeen. | Book Cliffs (?) | 10044, 46. | 2 | 2, 150 | 2, 150 | 4.64 | .35 |
| Do. | Aberdeen prospect. | Aberdeen. | 19702. | 1 | 2, 150 | 2, 150 | 6.68 | .43 |
| Do. | Kenilworth. | do. | 19080, 711. | 2 | 2, 150 | 2, 150 | 6.65 | .46 |
| Do. | do. | Kenilworth. | 19080, 712. | 2 | 2, 150 | 2, 150 | 6.63 | .50 |
| Do. | do. | Royal Blue. | 19082, 710. | 2 | 2, 150 | 2, 150 | 4.90 | .70 |
| Do. | Milburn prospect. | Kenilworth. | 19708. | 2 | 2, 150 | 2, 150 | 6.86 | .56 |
| Do. | Royal Blue. | Book Cliffs. | 10045. | 1 | 1 | 2, 150 | 4.57 | .65 |
| Do. | Jesse Knight prospect. | Unnamed. | 14678. | 1 | 1 | 2, 220 | 6.05 | .61 |
| Price. | Standard. | Castlegate. | 19088-89. | 2 | 2, 150 | 2, 150 | 6.26 | .40 |
| Standardville. | Spring Canyon No. 1. | Spring Canyon No. 1. | 19088. | 2 | 2, 150 | 2, 150 | 8.83 | .68 |
| Storrs. | Spring Canyon No. 2. | Spring Canyon No. 2. | 19087. | 1 | 1 | 2, 110 | 8.25 | .90 |
| Do. | Spring Canyon No. 3. | Spring Canyon No. 3. | 19090. | 1 | 1 | 2, 190 | 8.57 | .41 |
| Do. | Prospect. | Unnamed. | 14801. | 1 | 1 | 2, 210 | 4.20 | .53 |
| Sunnyvale. | do. | do. | 17762. | 1 | 1 | 2, 620 | 6.58 | 1.11 |
| Do. | Utah No. 1. | Lower. | 12632. | 1 | 1 | 2, 300 | 7.01 | 1.84 |
| Do. | Utah No. 3. | Utah No. 3. | 12631, 17604. | 2 | 2, 550 | 2, 760 | 8.04 | 1.32 |
| Do. | do. | Upper. | 12646, 630, 17605. | 3 | 2, 620 | +2, 880 | 7.69 | .67 |
| Do. | do. | do. | 22345-53, 55, 414. | 6 | 2, 040 | 2, 430 | 6.34 | .78 |
| EMERY COUNTY. | Black Hawk. | Hiawatha. | 12627. | 1 | 1 | 2, 130 | 6.17 | .41 |
| Black Hawk. | Browning. | "C". | 12642. | 1 | 1 | 2, 400 | 18.50 | .80 |
| Emery. | Casper. | "I". | 14008. | 1 | 1 | 2, 100 | 8.52 | 1.34 |
| Do. | Surface prospect. | "I". | 12613. | 1 | 1 | 2, 000 | 12.10 | 4.85 |
| Do. | Williams. | do. | 17577. | 1 | 1 | 2, 960 | 11.24 | .71 |
| GRAND COUNTY. | No. 1-A. | "B". | 17578. | 1 | 1 | 2, 670 | 11.28 | .65 |
| Thompson. | Prospect. | "A". | 19799. | 1 | 1 | 2, 500 | 23.47 | .77 |
| Do. | do. | do. | 19800. | 1 | 1 | 2, 560 | 21.56 | .56 |
| MORGAN COUNTY. | Heber Robinson. | Unnamed. | 17715. | 1 | 1 | 2, 040 | 23.28 | 7.05 |
| Devil's Slide. | Lucas & Smith. | do. | 17717. | 1 | 1 | 2, 210 | 19.66 | 3.50 |
| Do. | do. | do. | 17718. | 1 | 1 | 2, 130 | 10.17 | 4.76 |
| SANPETE COUNTY. | Coal Creek. | Unnamed. | 15061. | 1 | 1 | 2, 470 | 6.53 | .50 |
| Wales. | do. | do. | 15060. | 1 | 1 | 2, 380 | 13.53 | 3.77 |
| Do. | North Tunnel. | do. | 15045. | 1 | 1 | 2, 320 | 13.07 | .47 |
| Do. | Thomas. | do. | 15045. | 1 | 1 | 2, 320 | 13.07 | .47 |
| SEVIER COUNTY. | Surface prospect. | "A". | 20805. | 1 | 1 | 2, 710 | 15.88 | .57 |
| Emery. | Hogan prospect. | "A". | 20894. | 1 | 1 | 2, 150 | 10.55 | 4.28 |
| Fremont. | Keams & Duggins. | "Duggins". | 13216. | 1 | 1 | 2, 420 | 3.86 | 2.16 |
| Salina. | do. | do. | 13217. | 1 | 1 | 2, 470 | 4.72 | 1.85 |
| Do. | do. | do. | 13218. | 1 | 1 | 2, 300 | 4.72 | 1.59 |
| SUMMIT COUNTY. | Prospect. | Unnamed. | 20805. | 1 | 1 | 2, 710 | 15.88 | .57 |
| Carters. | do. | do. | 20894. | 1 | 1 | 2, 150 | 10.55 | 4.28 |
| Do. | Rees-Grass Creek. | Wasatch. | 13216. | 1 | 1 | 2, 420 | 3.86 | 2.16 |
| Coalville. | do. | do. | 13217. | 1 | 1 | 2, 470 | 4.72 | 1.85 |
| Do. | Superior. | do. | 13218. | 1 | 1 | 2, 300 | 4.72 | 1.59 |
| Do. | Wasatch. | do. | 13218. | 1 | 1 | 2, 300 | 4.72 | 1.59 |

LEE COUNTY.

| | | | | | | | | |
|---------------------|------------------------------|-------------------------|---------------|--------|------------|------------|------------|-----------|
| Keokee..... | Mohawk No. 1..... | Kelly (?)..... | 75748..... | 1..... | | 2,630..... | 6.81..... | .80..... |
| Do..... | Do..... | Do..... | 75747..... | 1..... | | 2,650..... | 6.76..... | .91..... |
| Do..... | Stonaga No. 1..... | McConnell..... | 75746-79..... | 2..... | 2,450..... | 2,510..... | 6.62..... | .73..... |
| Do..... | Do..... | Wilson..... | 75745-76..... | 2..... | 2,020..... | 2,070..... | 6.43..... | 2.38..... |
| Pennington Gap..... | Emerald No. 3..... | No. 3 (Wilson)..... | 75737..... | 1..... | | 2,280..... | 10.23..... | 3.86..... |
| Do..... | Stone Creek No. 1..... | Stone Creek..... | 75805..... | 1..... | | 2,190..... | 4.73..... | 1.98..... |
| Do..... | Virginia Blue Gem No. 1..... | North Fork..... | 75807..... | 1..... | | 2,060..... | 2.29..... | 1.75..... |
| Do..... | Penn Lee No. 1..... | No. 1..... | 54978..... | 1..... | | 2,800..... | 6.74..... | .79..... |
| Do..... | Do..... | Do..... | 54979..... | 1..... | | 2,710..... | 3.98..... | .72..... |
| Do..... | Fickett No. 1..... | No. 3 (Wilson)..... | 75788..... | 1..... | | 2,360..... | 8.87..... | 2.94..... |
| Pockett..... | Reed Creek..... | Unnamed..... | 25684-85..... | 2..... | 2,140..... | 2,250..... | 8.14..... | 3.69..... |
| St. Charles..... | Benedict No. 1..... | No. 7..... | 75789-90..... | 2..... | 2,060..... | 2,960..... | 8.46..... | 3.78..... |
| Do..... | Black Diamond No. 1..... | Jack Rock No. 3..... | 54981..... | 1..... | | 2,470..... | 14.22..... | 2.77..... |
| Do..... | Frederick No. 1..... | 2A..... | 75792..... | 1..... | | 2,800..... | 11.83..... | .63..... |
| Do..... | Do..... | No. 5..... | 75908..... | 2..... | 2,110..... | 2,110..... | 10.23..... | .71..... |
| Do..... | Old Virginia No. 3..... | 2A..... | 75906..... | 1..... | | 2,110..... | 5.88..... | .64..... |
| Do..... | St. Charles No. 1..... | No. 2..... | 75909..... | 1..... | | 2,110..... | 10.23..... | 2.12..... |
| Do..... | Tomlinson-Pearly No. 1..... | No. 5..... | 54980..... | 1..... | | 2,100..... | 3.43..... | 1.55..... |
| Do..... | Virginia Lee No. 1..... | No. 6..... | 75907..... | 1..... | | 2,640..... | 9.70..... | 1.06..... |
| Do..... | Virginia Iron No. 2..... | No. 6 (low splint)..... | 75906..... | 1..... | | 2,630..... | 7.15..... | 1.14..... |
| Do..... | Virginia Iron No. 3..... | do..... | | 1..... | | | | |

MONTGOMERY COUNTY.

| | | | | | | | | |
|---------------------|------------------------|---------------|----------------------|--------|------------|-------------|------------|----------|
| Blacksburg..... | Clements Hollow..... | Small..... | 10359..... | 1..... | | +3,010..... | 42.98..... | .34..... |
| Do..... | Plunkett & Wall..... | Large..... | 10357..... | 1..... | | +3,010..... | 18.06..... | .52..... |
| Do..... | Seymour-Price..... | do..... | 10360..... | 1..... | | 2,960..... | 19.34..... | .69..... |
| Do..... | Slusser..... | do..... | 10358, 22929-30..... | 3..... | 2,360..... | 2,410..... | 17.55..... | .60..... |
| Do..... | do..... | do..... | 30689..... | 1..... | | 2,260..... | 15.64..... | .66..... |
| Christiansburg..... | Lyken Hill..... | do..... | 19403..... | 1..... | | +3,010..... | 22.70..... | .72..... |
| Merrimac Mines..... | Merrimac..... | "Big"..... | 30692..... | 1..... | | 2,650..... | 20.07..... | .46..... |
| Do..... | Merrimac prospect..... | "Little"..... | 30693..... | 1..... | | +3,010..... | 21.29..... | .49..... |

FULASKI COUNTY.

| | | | | | | | | |
|------------------|--------------|--------------|------------|--------|-------|------------|------------|----------|
| Dublin..... | Cloyd..... | "Upper"..... | 20722..... | 1..... | | 3,010..... | 24.23..... | .42..... |
| Gunton Park..... | Summit..... | do..... | 30696..... | 1..... | | 3,010..... | 25.99..... | .78..... |
| Parrott..... | Parrott..... | "Big"..... | 30694..... | 1..... | | 2,350..... | 23.97..... | .68..... |
| Do..... | do..... | do..... | 19431..... | 1..... | | 3,010..... | 23.23..... | .69..... |

RUSSELL COUNTY.

| | | | | | | | | |
|------------|--------------------------|-------------------|--------------------------------|--------|------------|------------|------------|-----------|
| Dante..... | Clinchfield..... | Lower Banner..... | 10385..... | 1..... | | 2,350..... | 6.69..... | .66..... |
| Do..... | Do..... | Upper Banner..... | 10735, 18128-30, 32168-71..... | 8..... | 2,080..... | 2,210..... | 6.88..... | .63..... |
| Do..... | Clinchfield No. 2..... | do..... | 10387, 32529-27.... | 3..... | 2,200..... | 2,350..... | 7.88..... | .63..... |
| Do..... | Clinchfield No. 52..... | Lower Banner..... | 18122-23, 32529-30..... | 4..... | 2,110..... | 2,570..... | 6.61..... | .69..... |
| Do..... | Clinchfield No. 103..... | Kennedy..... | 18121..... | 2..... | | 2,220..... | 7.64..... | 1.83..... |
| Do..... | Laurel Branch..... | do..... | 32452-53..... | 1..... | 2,400..... | 2,400..... | 25.12..... | 2.02..... |
| Do..... | Open exposure..... | Kelly..... | 32454..... | 1..... | | 2,290..... | 4.52..... | .93..... |

TABLE I.—Softening temperatures of coal ash from coals of the United States—Continued.

| Locality. | Mine. | Coal bed. | Rank of coal. | Laboratory No. | Number of samples from mine. | Softening temperature ° F. | | | Average analysis of dry coal, percentage of— | |
|---------------------------|--------------------------|-----------------------|---------------|--|------------------------------|----------------------------|----------|----------|--|----------|
| | | | | | | Lowest. | Highest. | Average. | Ash. | Sulphur. |
| 1 | | | | | | | | | | |
| VIRGINIA—Contd. | | | | | | | | | | |
| RUSSELL COUNTY—continued. | | | | | | | | | | |
| Della..... | Flat Rock..... | Kennedy..... | B. | 25367-68..... | 2 | 2,060 | 2,160 | 2,130 | 9.58 | 0.84 |
| Do..... | Jackson..... | Unnamed..... | B. | 22346..... | 1 | | | 2,070 | 12.96 | 0.66 |
| Drill..... | Do..... | Kennedy..... | B. | 19484..... | 1 | | | 2,260 | 6.94 | 1.08 |
| Do..... | Sandy Ridge..... | do..... | B. | 19628, 22245..... | 2 | 2,070 | 2,220 | 2,150 | 5.81 | 1.00 |
| St. Paul..... | Russell Fork..... | do..... | B. | 32513..... | 1 | | | 2,130 | 3.98 | 0.49 |
| Stump..... | Clinchfield No. 201..... | Unnamed..... | B. | 18230-41..... | 3 | 2,160 | 2,200 | 2,180 | 4.58 | 0.49 |
| Wilder..... | Clinchfield No. 6..... | Upper Banner..... | B. | 18235-37..... | 3 | 2,260 | 2,360 | 2,300 | 5.86 | 0.77 |
| Do..... | Clinchfield No. 56..... | Lower Banner..... | B. | 18243-45..... | 3 | 2,300 | 2,450 | 2,350 | 5.92 | 0.82 |
| SCOTT COUNTY. | | | | | | | | | | |
| Adams..... | Hagan..... | Unnamed..... | B. | 10359..... | 1 | | | 2,530 | 29.61 | 1.01 |
| Dungannon..... | J. S. T..... | do..... | B. | 75743-44..... | 2 | 2,740 | 2,780 | 2,760 | 7.12 | 0.91 |
| Ka..... | Miner prospect..... | Miller..... | B. | 10358..... | 1 | | | 2,150 | 5.89 | 1.69 |
| Do..... | Hagan prospect..... | Duncan..... | B. | 10361..... | 1 | | | 2,160 | 6.65 | 0.88 |
| TAYLOR COUNTY. | | | | | | | | | | |
| Bandy..... | Christian..... | Middle Seaboard..... | B. | 25913..... | 1 | | | 2,360 | 4.15 | 1.03 |
| Do..... | Patrick..... | Lower Seaboard..... | B. | 25912..... | 1 | | | 2,260 | 3.67 | 1.14 |
| Bellevue..... | Big Vein No. 2..... | Poconantas No. 3..... | S. B. | 67627, 67664..... | 2 | 2,410 | 2,500 | 2,450 | 4.40 | 0.60 |
| Do..... | Bellevue..... | do..... | S. B. | 67857-58, 62, 99, 901, 92-94, 67922-23, 26, 57, 60, 71, 26101..... | 13 | 2,060 | 2,560 | 2,270 | 4.05 | 0.51 |
| Paradey..... | Altizer..... | Poconantas No. 5..... | S. B. | 26101..... | 1 | | | 2,090 | 5.19 | 0.82 |
| Jewell..... | Jewell Ridge No. 1..... | Raven (Red Ash)..... | S. B. | 26651-52..... | 2 | 2,500 | 2,640 | 2,570 | 5.94 | 0.78 |
| Poconantas..... | Baby..... | Poconantas No. 3..... | S. B. | 67921, 24, 26..... | 3 | 2,490 | 2,580 | 2,530 | 4.33 | 0.83 |
| Do..... | Big Vein No. 1..... | do..... | S. B. | 67896, 89, 902, 984, 951..... | 5 | 2,380 | 2,640 | 2,520 | 3.98 | 0.62 |
| Do..... | West Poconantas..... | do..... | S. B. | 67910, 53-54, 58..... | 4 | 2,190 | 2,500 | 2,320 | 4.52 | 0.46 |

| Do. | West Pocahontas (run-of-mine, 4 cars). | do. | S. B. | 30264. | 1 | 2,190 | 5.60 | .67 |
|----------------|--|-------------------------------|-------|---------------|---|--------|-------|------|
| Red Ash. | Raven Red Ash. | do. | B. | 25830-31. | 2 | 2,240 | 5.90 | .64 |
| Richlands. | East. | do. | B. | 25768-69. | 2 | 2,500 | 8.40 | .66 |
| Do. | Richlands No. 2. | do. | B. | 31114-15. | 2 | 2,450 | 12.92 | .62 |
| Do. | Richlands No. 4. | do. | B. | 31113. | 2 | 2,640 | 9.85 | .61 |
| Do. | West. | do. | B. | 25717-18. | 2 | 2,190 | 11.94 | .46 |
| Seaboard. | Empire No. 63. | Upper Seaboard. | B. | 26760-61. | 2 | 2,320 | 6.34 | .60 |
| WISE COUNTY. | | | | | | | | |
| Arno. | Rock Heading No. 2. | Imboden. | B. | 76864-66. | 3 | 2,670 | 4.19 | .66 |
| Banner. | Lower Banner No. 1. | Lower Banner. | B. | 32398-10. | 2 | 2,420 | 7.65 | .72 |
| Do. | No. 1. | Upper Banner. | B. | 32312. | 2 | 2,370 | 4.66 | .91 |
| Dooley. | Ontcrop. | Lee. | B. | 76963. | 1 | 2,910 | 43.59 | .62 |
| Dorchester. | Haskell No. 3. | Upper Banner (Haskell No. 3). | B. | 76440-41. | 1 | 2,220 | 5.03 | 1.46 |
| Dunbar. | Stonew. | Rhodes Marker. | B. | 32362. | 2 | 2,330 | 2.78 | .77 |
| Do. | do. | Taggart. | B. | 32363. | 2 | 2,370 | 3.06 | .63 |
| Easerville. | Essex No. 4. | Upper Banner (?). | B. | 75359-61. | 1 | 2,500 | 5.31 | 1.00 |
| Flat Gap. | Reuben Bolling. | Lower Banner. | B. | 15174. | 1 | 2,720 | 8.74 | 1.12 |
| Georgel. | Swansea. | Upper Banner. | B. | 10266. | 1 | 2,530 | 5.65 | .83 |
| Glamorgan. | Glamorgan No. 3. | Glamorgan. | B. | 15101-01. | 2 | 2,160 | 5.40 | 1.22 |
| Do. | do. | do. | B. | 32842-43. | 2 | 2,370 | 5.11 | 1.17 |
| Imboden. | Imboden No. 2. | Imboden. | B. | 76973-74. | 2 | 2,570 | 7.13 | .84 |
| Inman. | Imboden No. 2. | do. | B. | 76969-70. | 2 | 2,380 | 10.25 | .90 |
| Josephine. | Virginia Iron No. 2. | do. | B. | 32367-68. | 2 | 2,640 | 6.94 | 2.29 |
| Norton. | Norton No. 2. | do. | B. | 75363-65. | 2 | 2,190 | 3.89 | .98 |
| Do. | Unnamed. | Imboden. | B. | 10890. | 1 | 2,900 | 6.49 | .72 |
| Ocala. | Ocala. | do. | B. | 33183-94. | 3 | 2,920 | 9.52 | 1.23 |
| Pardee. | Pardee. | Pardee (Parsons). | B. | 33140-41. | 2 | 2,460 | 8.04 | 1.59 |
| Do. | Pardee No. 1. | do. | B. | 15099, 22279. | 2 | 2,500 | 6.53 | .94 |
| Roaring Fork. | Roaring Fork. | Kelly. | B. | 33143-44. | 2 | 2,370 | 2.28 | .83 |
| Roda. | Stone No. 3. | Taggart. | B. | 33212-06. | 2 | 2,330 | 12.69 | 2.22 |
| St. Paul. | Twin City No. 1. | Imboden. | B. | 32276. | 1 | 2,240 | 19.96 | 1.03 |
| Do. | Twin City No. 2. | Jawbone. | B. | 32276. | 1 | 2,910 | 4.96 | .77 |
| Do. | Twin City No. 3. | do. | B. | 32277. | 1 | 2,960 | 8.03 | .77 |
| Stonew. | Stonew No. 2. | Imboden. | B. | 33188. | 2 | 2,920 | 6.04 | 2.37 |
| Do. | Stonew No. 3. | do. | B. | 33186-87. | 3 | 2,180 | 10.50 | .96 |
| Sutherland. | Clark No. 2. | Haskell No. 3. | B. | 76436-38. | 2 | 2,910 | 6.70 | .94 |
| Taomsa. | Beaver. | Lower Banner. | B. | 76446-47. | 2 | 2,560 | 5.31 | .57 |
| Do. | Bolling. | Imboden. | B. | 76443-44. | 2 | 2,510 | 5.23 | .56 |
| Turns Creek. | Caney. | Upper Banner. | B. | 32407-08. | 3 | 2,660 | 6.25 | .51 |
| Do. | Cranehead No. 1. | do. | B. | 15226-28. | 2 | 2,370 | 14.84 | .62 |
| Do. | Do. | do. | B. | 32410-11. | 2 | 2,380 | 8.90 | .56 |
| Do. | Lee No. 6. | do. | B. | 32413. | 1 | 2,370 | 17.42 | 1.28 |
| Do. | Sexton No. 2. | do. | B. | 32404-05. | 2 | 2,470 | 6.45 | .78 |
| Do. | Thelma No. 1. | do. | B. | 32503-06. | 4 | 2,520 | 7.76 | 1.47 |
| Virginia City. | Virginia Iron. | Jawbone. | B. | 76746. | 1 | 2,460 | 35.20 | .34 |
| Wise. | Dean Brothers. | Glamorgan (?). | B. | 76367-69. | 3 | 2,370 | 35.20 | .34 |
| Do. | Gladeville No. 3. | Yellow Creek. | B. | 76367-69. | 3 | 2,370 | 35.20 | .34 |
| WYTHE COUNTY. | | | | | | | | |
| Max Meadows. | Ellison & Johnson prospect. | Upper. | B. B. | 20721. | 1 | +3,010 | 35.20 | .34 |

TABLE I.—Softening temperatures of coal ash from coals of the United States—Continued.

| Locality. | Mine. | Coal bed. | Rank of coal. | Laboratory No. | Number of samples from mine. | Softening temperature ° F. | | | Average analysis of dry coal, percentage of— | | |
|------------------|---------------------------|--------------|---------------|----------------|------------------------------|----------------------------|----------|----------|--|----------|--|
| | | | | | | Lowest. | Highest. | Average. | Ash. | Sulphur. | |
| 1 | | | | | | | | | | | |
| WASHINGTON. | | | | | 2 | 3 | 4 | 5 | 6 | 7 | |
| CLALLAM COUNTY. | | | | | | | | | | | |
| Clallam..... | Fuca..... | Unnamed..... | B. | 10080..... | 1 | | | 1,870 | 14.16 | 5.75 | |
| KING COUNTY. | | | | | | | | | | | |
| Cumberland..... | Rose-Marshall..... | Unnamed..... | B. | 10512..... | 1 | | | 2,980 | 17.24 | .63 | |
| Durham..... | Prospect..... | No. 1..... | B. | 32664..... | 1 | | | 3,000 | 17.51 | .54 | |
| Do..... | do..... | No. 2..... | B. | 32665..... | 1 | | | 2,730 | 21.42 | .88 | |
| Grand Ridge..... | do..... | No. 3..... | Sb. B. | 11736..... | 1 | | | 2,610 | 11.33 | .88 | |
| Do..... | do..... | No. 4..... | Sb. B. | 11737..... | 1 | | | 2,400 | 24.44 | 2.69 | |
| Do..... | do..... | No. 7..... | Sb. B. | 11738..... | 1 | | | 2,640 | 15.01 | .45 | |
| Ravensdale..... | do..... | No. 3..... | B. | 22465..... | 1 | | | 2,380 | 6.91 | .37 | |
| Snoqualmie..... | Niblock..... | do..... | B. | 10031..... | 1 | | | 2,960 | 11.65 | .51 | |
| Do..... | do..... | No. 4..... | B. | 10032..... | 1 | | | 2,980 | 13.21 | .94 | |
| Do..... | do..... | No. 5..... | B. | 10033..... | 1 | | | 2,910 | 25.58 | 1.56 | |
| KITTITAS COUNTY. | | | | | | | | | | | |
| Ellensburg..... | Prospect..... | No. 7..... | B. | 13266..... | 1 | | | 2,760 | 25.47 | 1.26 | |
| Do..... | do..... | No. 9C..... | B. | 13268..... | 1 | | | 2,250 | 7.73 | .66 | |
| Do..... | do..... | No. 10D..... | B. | 13267..... | 1 | | | 2,380 | 28.83 | 1.20 | |
| Do..... | do..... | Unnamed..... | B. | 26223..... | 1 | | | 2,580 | 17.73 | 1.59 | |
| Roslyn..... | Roslyn No. 1..... | Roslyn..... | B. | 11522-24..... | 3 | 2,350 | 2,590 | 2,440 | 12.99 | .40 | |
| Do..... | Roslyn No. 2..... | do..... | B. | 11526..... | 1 | | | 2,320 | 11.94 | .48 | |
| Thorp..... | Wilson (run-of-mine)..... | Unnamed..... | B. | 13269..... | 1 | | | 2,250 | 17.52 | .47 | |
| LEWIS COUNTY. | | | | | | | | | | | |
| Centralia..... | Fords Prairie..... | Unnamed..... | Sb. B. | 26564..... | 1 | | | 2,200 | 11.14 | .87 | |
| Do..... | Monarch..... | do..... | Sb. B. | 26570..... | 1 | | | 1,990 | 12.16 | 1.37 | |

| | | | | | | | | |
|----------------|-------------------|-----------------|-----------------|---|-------|-------|-------|------|
| Chahalls | Chahalls | do. | 9944 | 1 | 2,340 | 2,390 | 10.83 | 2.50 |
| Do | Sheldon | do. | 9943, 29599 | 2 | 2,390 | 2,390 | 8.30 | 8.75 |
| Do | Superior | Superior No. 2 | 29598 | 1 | 1 | 2,150 | 12.18 | 1.08 |
| Do | Superior No. 1 | do. | 9942 | 1 | 1 | 2,340 | 14.99 | 4.45 |
| Do | Superior No. 2 | do. | 9941 | 1 | 1 | 2,170 | 7.12 | 1.90 |
| Do | Twin City | do. | 9945 | 1 | 1 | 2,270 | 14.04 | 1.39 |
| Lead | East Creek Lead | No. 2 | 9952 | 1 | 1 | 2,240 | 18.05 | 1.31 |
| Do | do. | No. 3 | 9890-81 | 2 | 2,570 | 2,670 | 22.54 | 7.73 |
| Do | do. | No. 4 | 9879 | 1 | 1 | 2,620 | 26.78 | 0.88 |
| Do | Prospect | No. 1 | 33073 | 1 | 1 | 2,710 | 28.73 | 1.17 |
| Do | do. | No. 5 | 33072 | 1 | 1 | 2,530 | 50.34 | 7.73 |
| Do | do. | do. | 9940 | 1 | 1 | 2,150 | 12.97 | 4.36 |
| Littell | Crescent | do. | 10323-24, 29655 | 3 | 2,230 | 2,350 | 15.97 | 1.51 |
| Mendota | Mendota | do. | 12563 | 1 | 1 | 2,290 | 25.06 | 0.84 |
| Morton | Prospect | Edlund | 12564 | 1 | 1 | 2,760 | 17.40 | 7.73 |
| Do | do. | do. | 12565 | 1 | 1 | 2,970 | 25.48 | 0.90 |
| Do | do. | do. | 12565 | 1 | 1 | 2,660 | 25.48 | 0.90 |
| PIERCE COUNTY. | | | | | | | | |
| Ashtord | Mashel | do. | 9884-85 | 2 | 2,710 | 2,860 | 33.77 | 5.59 |
| Do | Prospect | Misqually Chief | 12581 | 1 | 1 | 2,650 | 25.72 | 4.44 |
| Burnett | Burnett | No. 2 | 9891 | 1 | 1 | 2,320 | 8.44 | 7.79 |
| Do | do. | No. 3 | 9890-90 | 2 | 2,350 | 2,420 | 13.74 | 3.89 |
| Carbonado | Carbon Hill | No. 1 | 9572 | 1 | 1 | 2,720 | 15.40 | 4.47 |
| Do | do. | No. 1 Coking | 9569 | 1 | 1 | 2,210 | 18.86 | 8.31 |
| Do | do. | No. 2 Coking | 9557 | 1 | 1 | 2,640 | 15.98 | 3.36 |
| Do | do. | No. 3 Coking | 9555, 65 | 2 | 2,640 | 2,740 | 17.54 | 3.33 |
| Do | do. | No. 4 | 9562 | 1 | 1 | 2,350 | 10.77 | 5.58 |
| Do | do. | No. 5 | 9564 | 1 | 1 | 2,890 | 16.90 | 5.54 |
| Do | do. | No. 9 | 9558 | 1 | 1 | 2,840 | 16.10 | 4.41 |
| Do | do. | No. 11 | 9570 | 1 | 1 | 2,740 | 20.42 | 8.22 |
| Do | do. | Wingsale | 9590, 9601 | 2 | 2,180 | 2,190 | 9.66 | 5.54 |
| Do | do. | do. | 9558 | 1 | 1 | 2,260 | 6.68 | 3.34 |
| Do | Carbon Hill No. 6 | Wilkeson | 10573-74 | 2 | 2,610 | 2,730 | 13.16 | 7.70 |
| Do | Carbonado No. 4N | do. | 9609 | 1 | 1 | 2,240 | 10.51 | 5.54 |
| Fairfax | Fairfax | No. 3 | 9607 | 1 | 1 | 2,740 | 34.22 | 4.48 |
| Do | do. | No. 7 | 9608 | 1 | 1 | 2,910 | 13.50 | 1.03 |
| Do | do. | No. 1 | 9602 | 1 | 1 | 2,570 | 23.44 | 7.73 |
| Do | Montezuma | No. 2 | 9603 | 1 | 1 | 2,890 | 20.21 | 5.57 |
| Do | do. | No. 3 | 9605 | 1 | 1 | 2,430 | 11.07 | 8.22 |
| Do | do. | No. 4 | 9606 | 1 | 1 | 2,290 | 35.15 | 5.57 |
| Do | do. | do. | 12478 | 1 | 1 | 2,980 | 20.46 | 1.19 |
| Do | Prospect | Prospect No. 1 | 12496 | 1 | 1 | 2,370 | 19.49 | 7.73 |
| Do | Prospect No. 2 | No. 1 | 12495 | 1 | 1 | 2,880 | 18.28 | 4.48 |
| Do | do. | No. 2 | 9576, 80 | 2 | 2,850 | 2,910 | 14.70 | 2.74 |
| Melmont | Melmont | No. 1 | 9576-79 | 2 | 2,410 | 2,750 | 24.19 | 4.48 |
| Do | do. | No. 2 | 9892 | 1 | 1 | 2,760 | 19.79 | 7.73 |
| Do | do. | No. 3 | 9895 | 1 | 1 | 2,960 | 21.33 | 5.58 |
| Pittsburg | Pittsburg | Black Carbon | 9894 | 1 | 1 | 2,300 | 21.87 | 4.42 |
| Do | do. | Lady Wellington | 9894 | 1 | 1 | 2,300 | 21.87 | 4.42 |
| Do | do. | Pittsburg | 9894 | 1 | 1 | 2,300 | 21.87 | 4.42 |
| Do | do. | Windsor | 9896 | 1 | 1 | 2,300 | 21.87 | 4.42 |
| South Willis | South Willis | Windsor | 9896 | 1 | 1 | 2,300 | 21.87 | 4.42 |

TABLE I.—Softening temperatures of coal ash from coals of the United States—Continued.

| Locality. | Mine. | Coal bed. | Rank of coal. | Laboratory No. | Number of samples from mine. | Softening temperature • F. | | | Average analysis of dry coal, percentage of— | |
|--------------------------|-----------------------|--------------|---------------|----------------|------------------------------|----------------------------|----------|----------|--|----------|
| | | | | | | Lowest. | Highest. | Average. | Ash. | Sulphur. |
| 1 | | | | | | | | | | |
| WASHINGTON—Continued. | | | | | 2 | 3 | 4 | 5 | 6 | 7 |
| PIECES COUNTY—continued. | | | | | | | | | | |
| Wilkeson..... | Briar Hill..... | Unnamed..... | B. | 9887..... | 1 | | | 2,700 | 29.91 | 1.21 |
| Do..... | Gale Creek..... | No. 1..... | B. | 9908..... | 1 | | | 2,300 | 8.53 | .85 |
| Do..... | do..... | No. 2..... | B. | 9909..... | 1 | | | 2,390 | 6.22 | 1.00 |
| Do..... | do..... | Queen..... | B. | 9910..... | 1 | | | 2,220 | 9.83 | 1.04 |
| Do..... | do..... | Snell..... | B. | 9886..... | 1 | | | 2,800 | 18.72 | .84 |
| Do..... | Wilkeson..... | No. 2..... | B. | 9903-05..... | 3 | 2,720 | 2,850 | 2,780 | 17.99 | .47 |
| Do..... | do..... | No. 3..... | B. | 9900-02..... | 3 | 2,240 | 2,970 | 2,570 | 14.27 | .47 |
| Do..... | do..... | No. 7..... | B. | 9889..... | 1 | | | 2,380 | 10.38 | .44 |
| STEVENS COUNTY. | | | | | | | | | | |
| Valley..... | Valley..... | Unnamed..... | Sb. B. | 34155-57..... | 2 | 2,040 | 2,070 | 2,060 | 22.73 | 2.82 |
| THURSTON COUNTY. | | | | | | | | | | |
| Hurn..... | Hannaford No. 1..... | Unnamed..... | Sb. B. | 9572..... | 1 | | | 2,150 | 7.69 | .55 |
| Do..... | Black Bear..... | do..... | Sb. B. | 9689..... | 1 | | | 2,570 | 27.61 | 1.79 |
| Do..... | King..... | do..... | Sb. B. | 9687..... | 1 | | | 2,210 | 14.12 | 3.09 |
| Do..... | Hannaford..... | do..... | Sb. B. | 29366-67..... | 2 | 2,330 | 2,340 | 2,340 | 12.28 | 1.19 |
| WELATON COUNTY. | | | | | | | | | | |
| Bellingham..... | Bellingham..... | No. 1..... | B. | 32084..... | 1 | | | 2,460 | 18.01 | .28 |
| Glacier..... | Discovery tunnel..... | Unnamed..... | S. A. | 1972-23..... | 2 | 2,430 | 2,510 | 2,470 | 11.55 | 1.04 |
| Do..... | Prospect..... | do..... | S. A. | 1972..... | 1 | | | 2,440 | 8.35 | 1.05 |
| Do..... | do..... | do..... | S. A. | 1972..... | 1 | | | 2,590 | 9.95 | 1.11 |
| Do..... | Smith tunnel..... | do..... | A. | 19724..... | 1 | | | 3,000 | 9.56 | .97 |

TABLE I.—Softening temperatures of coal ash from coals of the United States—Continued.

| Locality. | Mine. | Coal bed. | Rank of coal. | Laboratory No. | Number of samples from mine. | Softening temperature ° F. | | | Average analysis of dry coal, percentage of— | |
|---------------------------|--|------------------|---------------|-----------------|------------------------------|----------------------------|----------|----------|--|----------|
| | | | | | | Lowest. | Highest. | Average. | Ash. | Sulphur. |
| 1 | | | | | | | | | | |
| WEST VIRGINIA—Continued. | | | | | | | | | | |
| PAYETTE COUNTY—continued. | | | | | | | | | | |
| Minden | Minden No. 2 | Sewell | B. | 68113, 137, 133 | 3 | 2, 280 | 2, 640 | 2, 450 | 3.23 | 0.65 |
| Do. | Minden No. 3 | do. | B. | 68117 | 1 | | | 2, 600 | 2.86 | .56 |
| Do. | Minden No. 4 | do. | B. | 68124-25, 131 | 3 | 2, 270 | 2, 710 | 2, 550 | 2.57 | .43 |
| Do. | Minden No. 5 | do. | B. | 68116, 119, 121 | 4 | 2, 080 | 2, 620 | 2, 340 | 3.13 | .88 |
| Do. | Rock Lick No. 4 | do. | B. | 68114, 126, 132 | 3 | 2, 370 | 2, 620 | 2, 520 | 2.43 | .45 |
| Mount Hope | Catherine (run-of-mine, 25 tons) | do. | S. B. | 76383 | 1 | | | 2, 450 | 7.13 | 1.03 |
| Do. | Catherine | do. | S. B. | 77168-70 | 2 | 2, 350 | 2, 620 | 2, 490 | 4.38 | 1.12 |
| Do. | Hi-Top (run-of-mine, 1 car) | do. | S. B. | 76390 | 1 | | | 2, 430 | 4.35 | .83 |
| Do. | Mill Creek No. 1 | do. | S. B. | 76928-29 | 2 | 2, 250 | 2, 370 | 2, 310 | 3.60 | 1.56 |
| Do. | Minnie Bell | do. | S. B. | 77173-74 | 2 | 2, 450 | 2, 660 | 2, 560 | 3.66 | .91 |
| Do. | Minnie Bell (run-of-mine, ½ car) | do. | S. B. | 76377 | 1 | | | 2, 450 | 4.74 | 1.32 |
| Do. | Pack's Branch | Eagle | B. | 77203-04 | 2 | 2, 980 | 3, 000 | 3, 000 | 6.96 | .76 |
| Do. | Sugar Creek (run-of-mine, 5 cars) | Sewell | S. B. | 76805 | 1 | | | 2, 410 | 5.25 | 1.04 |
| Do. | Warner (run-of-mine, 30 tons) | do. | S. B. | 76392 | 1 | | | 2, 270 | 9.56 | 1.86 |
| Do. | Warner | do. | S. B. | 77171-72 | 2 | 2, 230 | 2, 800 | 2, 520 | 4.60 | 1.02 |
| Oakwood | Oakwood (run-of-mine, 10 cars) | do. | S. B. | 77080 | 2 | | | 2, 450 | 5.83 | 1.01 |
| Page | Ansted (car sample, run-of-mine) | No. 2 | B. | 9673 | 1 | | | 2, 430 | 6.61 | 1.81 |
| Do. | Page (run-of-mine, 12 cars) | Eagle and Ansted | B. | 76451 | 1 | | | 2, 690 | 9.81 | 1.41 |
| Do. | Page No. 1 | No. 1 Gas | B. | 76935-66 | 2 | 2, 850 | 2, 960 | 2, 910 | 5.83 | .88 |
| Do. | Page No. 2 Gas | No. 2 Gas | B. | 77200-02 | 3 | 2, 340 | 2, 400 | 2, 360 | 6.94 | 2.25 |
| Pax | Welwood | Sewell | S. B. | 32598-99 | 2 | 2, 400 | 2, 400 | 2, 400 | 6.31 | .95 |
| Do. | Pax Branch (run-of-mine, approximately 100 tons) | Upper Eagle | B. | 76487 | 1 | | | 2, 740 | 12.51 | 1.15 |
| Scarbro | Scarbro (run-of-mine, 8 cars) | Sewell | S. B. | 76992 | 1 | | | 2, 450 | 5.05 | .95 |
| South Nuttal | Brown | do. | S. B. | 76630-22 | 3 | 2, 580 | 2, 830 | 2, 730 | 2.55 | |

| | | | | | | | | |
|---------------------|---|-------|--------------------------|---|-------|--------|-------|------|
| Sumnerlee..... | Sumnerlee (run-of-mine, 7 cars)..... | B. | 77001..... | 1 | | 2,310 | 4.23 | .70 |
| Do..... | Sumnerlee..... | B. | 77179-82..... | 4 | 2,360 | 2,600 | 2.60 | .81 |
| Sun..... | New River (run-of-mine, 13 cars)..... | S. B. | 76817..... | 1 | | 2,440 | 6.16 | 1.08 |
| Do..... | Sun No. 1..... | S. B. | 9741-44, 47, 50, 53..... | 5 | 2,070 | 2,550 | 6.14 | 1.60 |
| Do..... | Sunset..... | S. B. | 77175-76..... | 2 | 2,360 | 2,420 | 3.97 | .92 |
| Do..... | Sunset (run-of-mine, 1 car)..... | S. B. | 76807..... | 1 | | 2,360 | 5.91 | .74 |
| Thaver..... | Rock Lick No. 2..... | S. B. | 10029, 10864, 730..... | 3 | 2,180 | 2,580 | 7.46 | .78 |
| Whipple..... | Buffalo..... | S. B. | 68129-30..... | 3 | 2,310 | 2,370 | 4.90 | .68 |
| Willis Branch..... | Whipple (run-of-mine, 7 cars, 1 car)..... | S. B. | 76991..... | 1 | | 2,310 | 4.53 | .74 |
| Do..... | Willis Branch..... | B. | 77165..... | 1 | | 2,850 | 10.60 | .71 |
| GRANT COUNTY. | Willis Branch..... | B. | 76831-33..... | 3 | 2,870 | 2,910 | 5.55 | .76 |
| Henry..... | Henry or No. 22..... | S. B. | 26579..... | 1 | | 2,190 | 6.17 | 1.97 |
| GREENBRIER COUNTY. | | | | | | | | |
| Richwood..... | Spruce Knob..... | B. | 13547..... | 1 | | 2,960 | 6.04 | .66 |
| HANCOCK COUNTY. | | | | | | | | |
| Chester..... | Alison coal bank..... | B. | 25633..... | 1 | | 2,130 | 6.04 | 1.88 |
| Do..... | Lower Kittanning..... | B. | 25635..... | 1 | | 2,150 | 6.15 | 4.64 |
| Do..... | Lower Kittanning..... | B. | 25591..... | 1 | | 2,110 | 10.93 | 4.06 |
| Do..... | Lower Kittanning..... | B. | 25534..... | 1 | | 2,070 | 12.55 | 4.29 |
| New Cumberland..... | Lower Kittanning..... | B. | 25716..... | 1 | | 2,280 | 13.65 | 9.20 |
| Do..... | Lower Kittanning..... | B. | 25714..... | 1 | | 2,300 | 4.13 | 1.46 |
| Do..... | Mahoning..... | B. | 25715..... | 1 | | 2,140 | 6.13 | 2.15 |
| Do..... | McNeal-Herron coal bank..... | B. | 25713..... | 1 | | 2,080 | 5.28 | 2.05 |
| HARRISON COUNTY. | | | | | | | | |
| Lost Creek..... | Calif. Righter No. 1..... | B. | 22321-23, 25..... | 4 | 2,000 | 2,140 | 8.08 | 2.24 |
| Do..... | Red Stone..... | B. | 22332-36..... | 5 | 2,030 | 2,370 | 6.07 | 1.66 |
| KANAWHA COUNTY. | | | | | | | | |
| Putney..... | Putney No. 1..... | B. | 23892-96..... | 5 | 2,250 | +3,010 | 9.87 | 1.11 |
| Ward..... | Kelly's Creek No. 3..... | B. | 23919-24..... | 6 | 2,700 | +3,010 | 8.51 | .74 |
| LOGAN COUNTY. | | | | | | | | |
| Acoville..... | Big Eagle..... | B. | 23407-12..... | 6 | 2,730 | 2,990 | 6.08 | .66 |
| Do..... | Island Creek..... | B. | 23414-18..... | 5 | 2,520 | +3,010 | 5.82 | 1.28 |
| Cranecko..... | Lorado No. 1..... | B. | 23509-12..... | 4 | 2,470 | +3,010 | 5.65 | .75 |
| Earling..... | Earling No. 1..... | B. | 23339, 41-42..... | 3 | 2,850 | 2,960 | 5.07 | .70 |
| Omar..... | Main Island Creek No. 5..... | B. | 23414-18..... | 5 | 2,390 | 2,900 | 5.62 | .88 |

TABLE I.—Softening temperatures of coal ash from coals of the United States—Continued.

| Locality. | Mine. | Coal bed. | Rank of coal. | Laboratory No. | Number of samples from mine. | Softening temperature ° F. | | | Average analysis of dry coal, percentage of— | |
|--------------------------|--|-----------------------|---------------|-------------------------------------|------------------------------|----------------------------|----------|----------|--|----------|
| | | | | | | Lowest. | Highest. | Average. | Ash. | Sulphur. |
| 1 | | | | | | | | | | |
| WEST VIRGINIA—Continued. | | | | | | | | | | |
| M'DOWELL COUNTY. | | | | | | | | | | |
| Piney..... | Algonia..... | Pocahontas No. 3..... | S. B. | 14154-55, 58-61..... | 7 | 2, 180 | 2, 300 | 2, 240 | 4.79 | 0.56 |
| Arlington..... | Arlington..... | do..... | S. B. | 68026, 81-82..... | 3 | 2, 460 | 2, 530 | 2, 520 | 3.77 | .40 |
| Bear Wallow..... | Roanoke..... | do..... | S. B. | 30232, 6787-88, 983..... | 4 | 2, 190 | 2, 350 | 2, 270 | 6.58 | .44 |
| Berwind..... | Berwind No. 1..... | do..... | S. B. | 68086, 91-92..... | 3 | 2, 280 | 2, 450 | 2, 360 | 6.17 | .90 |
| Do..... | Berwind No. 2..... | do..... | S. B. | 68081-82, 87..... | 3 | 2, 170 | 2, 340 | 2, 250 | 7.64 | .87 |
| Do..... | Berwind No. 3..... | do..... | S. B. | 68070, 73, 77..... | 3 | 2, 380 | 2, 510 | 2, 460 | 6.90 | .86 |
| Do..... | Berwind No. 4..... | do..... | S. B. | 68087-94, 74..... | 3 | 2, 310 | 2, 370 | 2, 340 | 5.1 | 1.00 |
| Do..... | Curtis No. 2..... | do..... | S. B. | 14218-21..... | 4 | 2, 200 | 2, 340 | 2, 280 | 4.86 | .63 |
| Do..... | do..... | Pocahontas No. 4..... | S. B. | 14223..... | 1 | | | 2, 770 | 3.85 | .57 |
| Big Sandy..... | Big Sandy..... | Sewell..... | S. B. | 68015, 21..... | 2 | 2, 260 | 2, 370 | 2, 330 | 3.60 | .53 |
| Coalwood..... | No. 1..... | Pocahontas No. 4..... | S. B. | 20300-01, 686-98..... | 5 | 2, 100 | 2, 220 | 2, 170 | 6.29 | .67 |
| Do..... | Olga..... | do..... | S. B. | 68094, 97-98..... | 3 | 2, 030 | 2, 140 | 2, 080 | 6.43 | .77 |
| Do..... | Nora or No. 3..... | Sewell..... | S. B. | 20307-10..... | 4 | 2, 350 | 2, 490 | 2, 420 | 2.95 | .74 |
| Do..... | Thelma or No. 6..... | do..... | S. B. | 20317-23..... | 7 | 2, 300 | 2, 650 | 2, 440 | 4.32 | 1.08 |
| Cooper..... | Tug River..... | Pocahontas No. 3..... | S. B. | 67896, 896, 933..... | 3 | 2, 370 | 2, 690 | 2, 530 | 3.0 | .47 |
| Davy..... | Blackstone..... | Sewell..... | S. B. | 20889..... | 1 | | | 2, 430 | 3.46 | .54 |
| Do..... | Cletus..... | do..... | S. B. | 20892..... | 1 | | | 2, 430 | 2.67 | .51 |
| Do..... | Helena..... | do..... | S. B. | 20896..... | 1 | | | 2, 400 | 3.18 | .58 |
| Do..... | Pulaski No. 2..... | do..... | S. B. | 20615-16..... | 10 | 2, 250 | 2, 500 | 2, 330 | 4.53 | .56 |
| Do..... | Pulaski No. 3..... | do..... | S. B. | 67929, 86, 68011..... | 4 | 2, 170 | 2, 270 | 2, 210 | 4.31 | .63 |
| Do..... | Shawnee..... | do..... | S. B. | 20610-13..... | 3 | 2, 300 | 2, 380 | 2, 340 | 5.67 | .52 |
| Elkhorn..... | Houston No. 1..... | do..... | S. B. | 67933, 37-38..... | 2 | 2, 190 | 2, 240 | 2, 220 | 4.90 | .48 |
| Do..... | Houston No. 2..... | do..... | S. B. | 67935, 59..... | 3 | 2, 120 | 2, 780 | 2, 380 | 5.00 | .45 |
| Elk Ridge..... | do..... | do..... | S. B. | 67932, 69-70..... | 1 | | | 2, 410 | 3.86 | .59 |
| Fareday..... | Elk Ridge..... | do..... | S. B. | 10033..... | 1 | | | 2, 380 | 3.99 | .55 |
| Gilliam..... | Presley..... | Pocahontas No. 5..... | S. B. | 26159..... | 1 | | | 2, 300 | 5.4 | .45 |
| Do..... | Gilliam..... | Pocahontas No. 3..... | S. B. | 68000, 623-23..... | 3 | 2, 180 | 2, 300 | 2, 250 | 3.06 | .44 |
| Do..... | Gilliam (run-of-mine, 4 cars)..... | do..... | S. B. | 30238..... | 1 | | | 2, 260 | 4.17 | .83 |
| Do..... | Berwind No. 6..... | do..... | S. B. | 68096, 72, 76..... | 3 | 2, 190 | 2, 410 | 2, 260 | 4.17 | .83 |
| Do..... | Berwind No. 6 (run-of-mine, 4 cars)..... | do..... | S. B. | 30283..... | 1 | | | 2, 310 | 9.55 | .96 |
| Havaco..... | Havaco..... | do..... | S. B. | 10034, 13775, 67999, 68019-626..... | 5 | 2, 190 | 2, 640 | 2, 350 | 5.96 | .74 |

| | | | | | | |
|--------------|--|--------------------------|---------------------------------|----|-------|-------|
| Do. | Hasaco (run-of-mine, 4 cars). | do. | 30227. | 1 | 2,240 | 8.19 |
| Hemphill. | Welch. | do. | 20552-55, 57. | 5 | 2,840 | 7.41 |
| Huger. | Pocahontas No. 1. | do. | 20548-53. | 6 | 2,370 | 4.98 |
| Do. | Lake Superior No. 3. | do. | 14221. | 1 | 2,420 | 4.73 |
| Do. | Middle States. | do. | 14418-19, 22-23. | 6 | 2,190 | 6.09 |
| Jenkintones. | do. | do. | 14185-87. | 14 | 2,170 | 5.04 |
| | Jenkintones No. 6. | Pocahontas No. 3. | 20518-23, 68063, 90, 96. | | 2,250 | 5.03 |
| Do. | Jenkintones No. 6 (car sample, 8 cars). | do. | 30258. | 1 | 2,280 | 5.9 |
| Do. | Jenkintones No. 7. | do. | 20507-11, 67965, 87, 68029, 92. | 9 | 2,270 | 3.98 |
| Do. | Jenkintones No. 8. | do. | 20531-35, 68069-89, 93, 95. | 9 | 2,320 | 5.16 |
| Do. | Jenkintones No. 8 (run-of-mine, 34 cars). | do. | 30229. | 1 | 2,280 | 6.08 |
| Kimball. | Carwell. | Pocahontas Nos. 3 and 4. | 68033, 34-35. | 3 | 2,270 | 6.4 |
| Leckie. | Leckie No. 1. | Pocahontas No. 3. | 14271-76. | 6 | 2,710 | 3.32 |
| Do. | Leckie No. 2. | do. | 14277-81. | 5 | 2,510 | 3.89 |
| Lex. | Central West. | Pocahontas No. 6. | 17469-70. | 2 | 2,790 | 3.64 |
| Do. | Lick Branch. | Pocahontas No. 3. | 67789, 94, 997. | 3 | 2,870 | 7.80 |
| Do. | Delta (car sample, 11 cars). | do. | 30260. | 1 | 2,210 | 4.30 |
| McDowell. | McDowell. | do. | 68006-07, 10. | 3 | 2,370 | 7.71 |
| Marytown. | Marytown. | Sewell. | 68002-03. | 3 | 2,490 | 3.98 |
| Newhall. | Berwind No. 6. | Pocahontas No. 3. | 68066, 75. | 2 | 2,670 | 8.96 |
| Do. | Berwind No. 6 (run-of-mine, 9 cars). | do. | 30235. | 1 | 2,540 | 4.90 |
| Do. | Berwind No. 7. | do. | 68146, 49, 64. | 3 | 2,690 | 9.03 |
| Do. | Berwind No. 8. | do. | 68071, 78-79. | 3 | 2,610 | 4.27 |
| Do. | Berwind No. 8 (run-of-mine, 8 cars). | do. | 30262. | 1 | 2,970 | 6.55 |
| Roderfield. | Davy Pocahontas No. 1. | Beckley (War Creek). | 14265-68. | 3 | 2,700 | 7.69 |
| Do. | Davy Pocahontas No. 2. | Sewell. | 14262-64. | 3 | 2,820 | 11.31 |
| Rolle. | Rolle. | Pocahontas No. 3. | 14394-96, 400. | 3 | 2,430 | 6.43 |
| Susana. | Yukon-Pocahontas. | Beckley (War Creek). | 14428-29. | 6 | 2,400 | 4.43 |
| Twin Branch. | do. | Sewell. | 68001. | 4 | 2,640 | 8.50 |
| Do. | J. B. B. No. 1. | do. | 68004. | 1 | 2,440 | 3.1 |
| Do. | J. B. B. No. 2. | do. | 29851. | 1 | 3,010 | 4.4 |
| Do. | J. B. B. No. 3. | do. | 67999. | 1 | 2,540 | 5.27 |
| Do. | J. B. B. No. 4. | do. | 67997-98. | 1 | 2,550 | 6.7 |
| Do. | J. B. B. No. 5. | do. | 22480-85, 89. | 2 | 2,500 | 3.0 |
| Do. | J. B. B. (or Maher) No. 4. | do. | 22487-88. | 2 | 2,680 | 4.61 |
| Do. | J. B. B. (or Maher) No. 4 (car sample, run-of-mine). | do. | 28314-16. | 2 | 2,690 | 4.23 |
| Vivian. | Carwell. | Pocahontas No. 4. | 68024-25, 77, 80. | 3 | 2,850 | 6.81 |
| Do. | King. | Pocahontas No. 3. | 20261. | 4 | 2,160 | 4.44 |
| Do. | King (car sample, 11 cars). | do. | 24800. | 2 | 2,280 | 4.85 |
| Do. | King No. 98. | do. | 11247, 780. | 1 | 2,260 | 8.05 |
| Do. | Norfolk (car sample). | do. | 68006, 08-09. | 1 | 2,940 | 4.03 |
| Do. | Tidewater. | do. | 30226. | 2 | 2,730 | 10.33 |
| Do. | Tidewater (run-of-mine, 15 cars). | do. | | 3 | 2,300 | 5.55 |
| Do. | | do. | | 1 | 2,330 | 8.55 |

TABLE I.—Softening temperatures of coal ash from coals of the United States—Continued.

| Locality. | Mine. | Coal bed. | Rank of coal. | Laboratory No. | Number of samples from mine. | Softening temperature ° F. | | | Average analysis of dry coal, percentage of— | |
|----------------------|---------------------------|--------------------------|---------------|--------------------------|------------------------------|----------------------------|----------|----------|--|----------|
| | | | | | | Lowest. | Highest. | Average. | Ash. | Sulphur. |
| 1 | | | | | | | | | | |
| WEST VIRGINIA— | | | | | | | | | | |
| Continued. | | | | | | | | | | |
| M'DOWELL COUNTY— | | | | | | | | | | |
| continued. | | | | | | | | | | |
| War..... | John's Branch..... | Beckley (War Creek)..... | S. B. | 14468-71..... | 4 | 2,650 | +3,010 | +2,980 | 8.99 | 0.63 |
| Do..... | War Creek..... | do..... | S. B. | 14408-07..... | 3 | 2,780 | 2,750 | 2,740 | 8.62 | .72 |
| Welch..... | Oregon No. 2..... | Pocahontas No. 4..... | S. B. | 14215-16..... | 2 | 3,010 | 3,010 | 3,010 | 7.52 | .53 |
| Do..... | Oregon No. 3..... | do..... | S. B. | 14212-14..... | 3 | 2,960 | +3,010 | +2,980 | 7.92 | .63 |
| Do..... | Standard Pocahontas..... | Pocahontas No. 3..... | S. B. | 12512-13..... | 2 | 2,430 | 2,470 | 2,450 | 7.36 | .97 |
| MARION COUNTY. | | | | | | | | | | |
| Chickton..... | Consolidation No. 47..... | Pittsburgh..... | B. | 28570-75..... | 6 | 2,040 | 2,170 | 2,090 | 7.41 | 2.97 |
| Farmount..... | Consolidation No. 38..... | do..... | B. | 28106-07, 11..... | 3 | 2,110 | 2,180 | 2,130 | 6.97 | 2.29 |
| Farmington..... | Consolidation No. 87..... | do..... | B. | 28505-08..... | 4 | 2,010 | 2,330 | 2,220 | 6.61 | 1.23 |
| Hutchinson..... | Consolidation No. 84..... | do..... | B. | 28432-35, 66-70..... | 6 | 2,010 | 2,310 | 2,260 | 7.12 | 1.98 |
| Monongah..... | Consolidation No. 43..... | do..... | B. | 28209-11, 25-27..... | 6 | 2,170 | 2,360 | 2,270 | 6.27 | 1.07 |
| Do..... | Consolidation No. 63..... | do..... | B. | 11431-32, 28323-26..... | 6 | 2,200 | 2,330 | 2,270 | 5.84 | 1.07 |
| Montana Station..... | Parker Run..... | Sewickley..... | B. | 15067-71..... | 5 | 2,090 | 2,200 | 2,090 | 9.61 | 3.99 |
| Watson..... | Consolidation No. 28..... | Pittsburgh..... | B. | 28138-42, 44..... | 5 | 2,000 | 2,200 | 2,110 | 6.65 | 1.87 |
| Worthington..... | Annabelle..... | do..... | B. | 32839..... | 1 | 2,240 | 2,410 | 2,200 | 6.04 | .82 |
| Do..... | Consolidation No. 36..... | do..... | B. | 28535-38..... | 4 | 2,070 | 2,170 | 2,120 | 6.15 | .80 |
| Do..... | Hutchinson..... | do..... | B. | 11163-65..... | 3 | 2,070 | 2,170 | 2,130 | 8.70 | 2.25 |
| MARSHALL COUNTY. | | | | | | | | | | |
| Benwood..... | Hitchman..... | Pittsburgh..... | B. | 25605-10..... | 5 | 1,940 | 1,990 | 1,970 | 8.36 | 4.64 |
| Moundsville..... | Mound..... | do..... | B. | 16422-23..... | 2 | 2,020 | 2,130 | 2,080 | 12.44 | 6.09 |
| Do..... | Panama..... | do..... | B. | 14456-57..... | 2 | 2,070 | 2,090 | 2,080 | 7.29 | 3.38 |
| MERCER COUNTY. | | | | | | | | | | |
| Algonquin..... | Algonquin..... | Pocahontas No. 3..... | S. B. | 77208-08..... | 3 | 2,720 | 2,940 | 2,770 | 4.60 | .56 |
| Coaldale..... | Coaldale..... | do..... | S. B. | 67980, 97, 905..... | 3 | 2,300 | 2,540 | 2,410 | 4.07 | .70 |
| Crystal..... | Crystal No. 1..... | do..... | S. B. | 29653, 67988, 88..... | 3 | 2,580 | 2,610 | 2,590 | 6.27 | .55 |
| Do..... | Crystal No. 2..... | do..... | S. B. | 67984, 68014-15, 17..... | 4 | 2,410 | 2,530 | 2,480 | 4.50 | .64 |
| Do..... | Godfrey..... | do..... | S. B. | 14284-59, 61..... | 3 | 2,370 | 2,580 | 2,490 | 3.88 | .73 |

TABLE I.—Softening temperatures of coal ash from coals of the United States—Continued.

| Locality. | Mine. | Coal bed. | Rank of coal. | Laboratory No. | Number of samples from mine. | Softening temperature ° F. | | | Average analysis of dry coal, percentage of— | |
|--------------------------|---|--|---------------|-----------------------------------|------------------------------|----------------------------|----------|----------|--|----------|
| | | | | | | Lowest. | Highest. | Average. | Ash. | Sulphur. |
| 1 | | | | | | | | | | |
| WEST VIRGINIA—Continued. | | | | | | | | | | |
| PRESTON COUNTY. | | | | | | | | | | |
| Corinth..... | Wills No. 3..... | Lower Kittanning..... | B. | 26516..... | 1 | | | 2,080 | 9.55 | 3.15 |
| Crallin (Md.)..... | Arnold Run No. 2..... | do..... | B. | 33875, 34124..... | 2 | | | 2,440 | 12.00 | 2.18 |
| Do..... | Banner..... | do..... | B. | 34123, 26-28..... | 3 | 2,440 | 2,440 | 2,500 | 11.66 | 2.02 |
| Do..... | Frederick..... | do..... | S. B. | 33871-73..... | 3 | 2,380 | 2,480 | 2,440 | 13.91 | 2.40 |
| Do..... | Guthrie..... | do..... | B. | 33888..... | 1 | | | 2,440 | 15.70 | 3.13 |
| Do..... | Kildow..... | do..... | B. | 33900-91..... | 2 | | | 2,470 | 15.39 | 3.02 |
| Do..... | Prudergast..... | do..... | B. | 33889..... | 1 | 2,440 | 2,500 | 2,780 | 19.02 | 2.18 |
| Do..... | Kempton or No. 42..... | Middle Kittanning..... | S. B. | 26578..... | 1 | | | 2,430 | 5.55 | .83 |
| FULASKI COUNTY. | | | | | | | | | | |
| Pulaski..... | Virginia Anthracite No. 1..... | Little Bed..... | S. A. | 75888-89..... | 2 | 2,530 | 2,980 | 2,730 | 14.93 | .60 |
| PUTNAM COUNTY. | | | | | | | | | | |
| Black Betsey..... | Black Betsey No. 3..... | Pittsburgh..... | B. | 23641-46..... | 6 | 1,930 | 2,370 | 2,150 | 7.40 | 1.92 |
| BALEIGH COUNTY. | | | | | | | | | | |
| Abney..... | Boone No. 1..... | Berkley..... | S. B. | 76881-82..... | 2 | 2,540 | 2,900 | 2,720 | 2.73 | .88 |
| Do..... | Boone Smokeless (run-of-mine, 1 car)..... | Fire Creek..... | S. B. | 76196..... | 1 | | | 2,370 | 11.03 | 1.64 |
| Affinity..... | Affinity..... | Berkley..... | S. B. | 26991-93, 95, 68115, 122-128..... | 7 | 2,660 | +3,000 | +3,000 | 4.19 | .56 |
| Do..... | Affinity (run-of-mine, 2 cars)..... | do..... | S. B. | 76104..... | 1 | | | 2,800 | 7.45 | .75 |
| Amigo..... | Amigo (run-of-mine, 2 cars)..... | Peachontas No. 3..... | S. B. | 76110..... | 1 | | | 2,730 | 8.66 | .71 |
| Do..... | Kirby No. 1..... | do..... | S. B. | 76215-16..... | 2 | | | 2,670 | 5.20 | .95 |
| Do..... | Kirby No. 1 (run-of-mine, 75 tons)..... | Berkley, 75 per cent; Peachontas No. 3, 25 per cent..... | S. B. | 76254..... | 1 | 2,570 | 2,760 | 2,660 | 8.10 | .64 |
| Bacantown..... | Bacantown (run-of-mine, 4 cars)..... | Fire Creek..... | S. B. | 77721..... | 1 | | | 2,510 | 5.27 | .92 |
| Do..... | Do..... | do..... | S. B. | 76922..... | 1 | | | 2,510 | 5.25 | 1.26 |

| | | | | | | | |
|-----------------|--|-------|------------------------------------|---|--------|---------|------|
| Do..... | BattleShip (run-of-mine, 3 cars)..... | S. B. | 76197 | 1 | 2, 610 | 6.71 | .88 |
| Do..... | Wilton No. 1..... | S. B. | 76917-19..... | 3 | 2, 460 | 2, 860 | .94 |
| Beckley..... | Beckley (run-of-mine, 11 cars)..... | S. B. | 77414..... | 1 | 2, 370 | 2, 710 | 1.04 |
| Do..... | Do..... | S. B. | 77187-88..... | 2 | 2, 450 | 2, 450 | .70 |
| Do..... | City No. 1 (run-of-mine, 15 tons)..... | S. B. | 76899..... | 1 | 2, 430 | 2, 430 | .88 |
| Do..... | City No. 2 (run-of-mine, 1 car)..... | S. B. | 77415..... | 1 | 2, 530 | 2, 530 | .95 |
| Besco..... | Beckley Smokeless..... | S. B. | 77187-89..... | 3 | 2, 520 | 2, 740 | .63 |
| Do..... | Clyde Pocahontas..... | S. B. | 76285-86..... | 2 | 2, 800 | 2, 660 | .66 |
| Do..... | Clyde Pocahontas (run-of-mine, 2 cars)..... | S. B. | 76822..... | 1 | 2, 960 | 2, 960 | .65 |
| Big Stick..... | Big Stick..... | S. B. | 26684-86, 88-89, 68118, 34-35..... | 8 | 2, 700 | +3, 010 | .55 |
| Do..... | Big Stick (run-of-mine, 12 cars)..... | S. B. | 76899..... | 1 | 2, 570 | 2, 570 | .57 |
| Cepoco..... | Callaway (run-of-mine, 2 cars)..... | S. B. | 76378..... | 1 | 2, 340 | 2, 340 | .99 |
| Craberry..... | Craberry..... | S. B. | 26274-75..... | 2 | 2, 460 | 2, 480 | .88 |
| East Gulf..... | Mead No. 1..... | S. B. | 76290-71..... | 3 | 2, 250 | 2, 890 | .55 |
| Eccles..... | Eccles No. 3 (run-of-mine, 6 cars)..... | S. B. | 76921..... | 1 | 2, 700 | 2, 700 | 1.08 |
| Do..... | Eccles No. 5..... | S. B. | 19774-78..... | 5 | 2, 580 | +3, 010 | .78 |
| Do..... | Eccles No. 5 (run-of-mine, 8 cars)..... | S. B. | 76922..... | 1 | 2, 800 | 2, 800 | .79 |
| Do..... | Eccles No. 6 (run-of-mine, 9 cars)..... | S. B. | 77316..... | 1 | 2, 350 | 2, 350 | 1.10 |
| Fireco..... | Douglas No. 2..... | S. B. | 76387-90..... | 4 | 2, 570 | 2, 750 | .84 |
| Do..... | Douglas No. 2 (run-of-mine, 10 cars)..... | S. B. | 77723..... | 1 | 2, 570 | 2, 570 | .95 |
| Do..... | Leckie No. 1..... | S. B. | 76914-15..... | 2 | 2, 780 | 2, 750 | .76 |
| Do..... | Leckie No. 2..... | S. B. | 76907-09..... | 3 | 2, 730 | 2, 800 | .80 |
| Do..... | Leckie No. 2 (run-of-mine, 6 cars)..... | S. B. | 77596..... | 1 | 2, 570 | 2, 570 | .92 |
| Do..... | Lillybrook No. 2..... | S. B. | 76481-83..... | 3 | 2, 540 | 2, 940 | .91 |
| Do..... | Lillybrook No. 2 (run-of-mine, 6 cars)..... | S. B. | 77722..... | 1 | 2, 600 | 2, 600 | 1.09 |
| Glen White..... | Glen White (run-of-mine, 27 cars)..... | S. B. | 76289..... | 1 | 2, 710 | 2, 710 | .97 |
| Helen..... | East Gulf No. 3..... | S. B. | 76278-80..... | 3 | 2, 660 | 2, 790 | .90 |
| Do..... | East Gulf No. 4..... | S. B. | 76283-85..... | 3 | 2, 570 | 2, 960 | .88 |
| Do..... | East Gulf No. 5..... | S. B. | 76301-03..... | 3 | 2, 850 | 2, 970 | .68 |
| Do..... | Helen Nos. 3 and 4 (run-of-mine, 32 cars)..... | S. B. | 76229..... | 1 | 2, 790 | 2, 790 | .63 |
| Do..... | Helen No. 5 (run-of-mine, 3 cars)..... | S. B. | 76290..... | 1 | 2, 790 | 2, 790 | .73 |
| Hotoal..... | Big Stick..... | S. B. | 14575-77..... | 3 | 2, 960 | +3, 010 | .99 |
| Do..... | Gulf (Hotoal)..... | S. B. | 76924-27, 68..... | 5 | 2, 660 | +3, 000 | .72 |
| Do..... | Gulf (Hotoal, run-of-mine, 10 cars)..... | S. B. | 76868..... | 1 | 2, 960 | 2, 960 | .63 |
| Lego..... | Fire Creek..... | S. B. | 77191-93..... | 3 | 2, 689 | 2, 910 | .92 |
| Do..... | Laurel Smokeless..... | S. B. | 76542-43..... | 2 | 2, 730 | 2, 830 | .82 |
| Do..... | Laurel Smokeless (run-of-mine, 1 car)..... | S. B. | 76863..... | 1 | 2, 780 | 2, 820 | .74 |

TABLE I.—Softening temperatures of coal ash from coals of the United States—Continued.

| Locality. | Mine. | Coal bed. | Rank of coal. | Laboratory No. | Number of samples from mine. | Softening temperature ° F. | | | Average analysis of dry coal, percentage of— | |
|--------------------------|--|-------------------|---------------|-------------------------------|------------------------------|----------------------------|----------|----------|--|----------|
| | | | | | | Lowest. | Highest. | Average. | Ash. | Sulphur. |
| 1 | | | | | | | | | | |
| WEST VIRGINIA—Continued. | | | | | | | | | | |
| BALCON COUNTY—continued. | | | | | | | | | | |
| Lester. | Lester Smokeless (run-of-mine, 13 tons). | Sewell. | S. B. | 77266. | 1 | | | 2,440 | 10.46 | 1.63 |
| Do. | Neal. | do. | S. B. | 76951-52. | 2 | 2,390 | 2,780 | 2,570 | 4.88 | 1.00 |
| Do. | Neal (run-of-mine, 2 cars). | do. | S. B. | 76951. | 1 | | | 2,510 | 12.43 | 1.37 |
| Lillybrook. | Lillybrook No. 1. | Fire Creek. | S. B. | 76399-401. | 4 | 2,410 | 2,800 | 2,660 | 4.79 | .97 |
| Do. | Lillybrook No. 1 (run-of-mine, 8 cars). | do. | S. B. | 76241. | 1 | | | 2,770 | 6.22 | .88 |
| Do. | Lilly and Hornbrook No. 3. | Pocahontas No. 6. | S. B. | 76351-52. | 2 | 2,740 | 2,740 | 2,740 | 5.13 | .83 |
| Do. | Lilly and Hornbrook No. 3 (run-of-mine, 2 cars). | do. | S. B. | 76183. | 1 | | | 2,670 | 9.95 | .90 |
| McAlpin. | McAlpin. | Beckley | S. B. | 68299-50. | 2 | 2,780 | +3,010 | +2,870 | 2.65 | .65 |
| Do. | McAlpin (run-of-mine, 21 cars). | do. | S. B. | 77720. | 1 | | | 2,790 | 7.62 | .64 |
| McQuade Station. | Blue Jay No. 4. | do. | S. B. | 14288-50. | 3 | 2,730 | +3,020 | +2,880 | 4.38 | .75 |
| Mabecott. | Mabecott (run-of-mine, 9 cars). | Sewell. | S. B. | 77413. | 1 | | | 2,890 | 6.70 | 1.23 |
| Metakon. | Summit (run-of-mine, 6 cars). | do. | S. B. | 76312. | 1 | | | 2,860 | 6.91 | .89 |
| Mistake. | Lynwin. | Beckley | S. B. | 14319-23, 63130-37, 76896-98. | 10 | 2,620 | +3,010 | +2,900 | 3.99 | .74 |
| Do. | Lynwin (run-of-mine, 6 cars). | do. | S. B. | 76954. | 1 | | | 2,740 | 8.33 | .67 |
| Morco. | Morco (run-of-mine, 1 car). | Pocahontas No. 3. | S. B. | 75768. | 1 | | | 2,730 | 15.06 | .50 |
| Mount Hope. | Cepecoe No. 1. | Sewell. | S. B. | 76946-49. | 2 | 2,000 | 2,200 | 2,150 | 4.92 | 2.37 |
| Oswald. | Oswald (run-of-mine, 4 cars). | do. | S. B. | 77134. | 1 | | | 2,460 | 4.59 | .83 |
| Do. | Oswald No. 3. | do. | S. B. | 63166, 69-70. | 3 | 2,190 | 2,590 | 2,360 | 4.07 | .80 |
| Pemberton. | Pemberton. | Beckley | S. B. | 76962-64. | 3 | 2,810 | +3,000 | +2,920 | 5.43 | .67 |
| Do. | Pemberton (run-of-mine, 44 cars). | do. | S. B. | 76967. | 1 | | | 2,890 | 7.34 | .76 |
| Do. | Regland. | do. | S. B. | 77223-25. | 3 | 2,800 | +3,000 | +2,980 | 6.20 | .69 |
| Do. | Regland (run-of-mine, 5 cars). | do. | S. B. | 77181. | 1 | | | 2,900 | 7.46 | .68 |
| Price Hill. | Price Hill (run-of-mine, 7 cars). | Sewell. | S. B. | 76379. | 1 | | | 2,280 | 5.37 | 1.37 |
| Price Wick. | Price Wick. | Beckley | S. B. | 76317-20. | 4 | 2,150 | 2,990 | 2,590 | 3.33 | .6 |
| Prosperity. | Granberry No. 1 (run-of-mine, 16 cars). | Sewell. | S. B. | 77412. | 1 | | | 2,520 | 4.93 | .69 |

| | | | | | | | | | |
|-------------|---|--|-------|-----------------------------|---|-------|--------|-------|------|
| Raleigh | Raleigh No. 1 | Bectley | S. B. | 10830 | 1 | | +3,010 | 3.70 | .65 |
| Do. | Raleigh No. 1 and 4 (run-of-mine, 3 cars) | do. | S. B. | 77499 | 1 | | 2,730 | 4.45 | .69 |
| Do. | Raleigh No. 3 | do. | S. B. | 10824-27 | | 2,620 | 2,760 | 3.67 | .75 |
| Do. | Raleigh No. 3 (run-of-mine, 8 cars) | do. | S. B. | 77493 | 1 | | 2,720 | 4.32 | .71 |
| Do. | Raleigh No. 6 (run-of-mine, 14 cars) | do. | S. B. | 77500 | 1 | | 2,780 | 5.26 | .86 |
| Rhodell | Rhodell No. 1 | Pocahontas No. 3 | S. B. | 76226-37 | 2 | 2,910 | 2,910 | 4.45 | .88 |
| Do. | Rhodell No. 1 and 2 (run-of-mine, 3 cars) | Bectley, 80 per cent; and Pocahontas No. 3, 20 per cent. | S. B. | 76386 | 1 | | 2,910 | 4.45 | .87 |
| Do. | Rhodell No. 2 | Bectley | S. B. | 76229-30 | 2 | 2,800 | 2,850 | 5.68 | .82 |
| Stanton | Cranberry No. 2 | Sewell | S. B. | 76572-76 | 4 | 2,460 | 2,860 | 2.99 | .73 |
| Do. | Cranberry No. 2 (run-of-mine, 9 cars) | do. | S. B. | 76874 | 1 | | 2,740 | 5.45 | .81 |
| Slab Fork | Slab Fork No. 1 | Bectley | S. B. | 14337 | 1 | | 2,580 | 2.87 | .61 |
| Do. | Slab Fork No. 1 (run-of-mine, 16 cars) | do. | S. B. | 77328 | 1 | | 2,500 | 5.53 | .71 |
| Do. | Slab Fork No. 2 | do. | S. B. | 14325-26 | 8 | 2,450 | +3,010 | 3.69 | .50 |
| Do. | Slab Fork No. 2 (run-of-mine, 34 cars) | do. | S. B. | 68164, 64, 61 | 1 | | +2,740 | 5.87 | .69 |
| Do. | Slab Fork No. 3 | do. | S. B. | 77339 | 1 | | 2,800 | 4.08 | .54 |
| Do. | Slab Fork No. 4 | do. | S. B. | 14332-34, 68155, 57, 60, 64 | 7 | 2,370 | +3,010 | 2.39 | .55 |
| Do. | Slab Fork No. 6 | do. | S. B. | 14336 | 1 | | 2,960 | 3.49 | .50 |
| Do. | Slab Fork No. 6 | do. | S. B. | 14313-16, 18, 68150-52 | 8 | 2,530 | +3,010 | 4.45 | .83 |
| Sprague | Cranberry No. 3 (run-of-mine, 12 cars) | Sewell | S. B. | 77411 | 1 | | 2,530 | 2.92 | 1.22 |
| Stanford | Elkhorn Piney No. 6 | do. | S. B. | 77226-28 | 3 | 2,130 | 2,780 | 4.70 | .60 |
| Statesbury | White No. 3 | Bectley | S. B. | 76942-46 | 5 | 2,800 | 2,910 | 5.99 | .66 |
| Do. | Statesbury (run-of-mine, 21 cars) | do. | S. B. | 77786 | 1 | | 2,760 | 6.44 | 1.25 |
| Sullivan | Piney Creek No. 1 | do. | S. B. | 76900-01 | 2 | 2,510 | 2,580 | 6.42 | 1.62 |
| Do. | Sullivan (run-of-mine, 4 cars) | do. | S. B. | 76945 | 1 | | 2,570 | 1.97 | .60 |
| Tamroy | Tamroy | Sewell | S. B. | 68166-68 | 3 | 2,640 | +3,010 | 4.93 | .95 |
| Do. | Tamroy (run-of-mine, 4 cars) | do. | S. B. | 77132 | 1 | | 2,440 | 4.99 | .87 |
| Tams | Tams (run-of-mine, 18 cars) | Bectley | S. B. | 76869 | 1 | | 2,790 | 6.2 | .60 |
| Do. | Tams No. 1 | do. | S. B. | 68086 | 1 | | +3,010 | 4.3 | .70 |
| Do. | Tams No. 3 | do. | S. B. | 68084 | 1 | | 2,010 | 7.60 | .83 |
| Do. | Tams No. 4 | do. | S. B. | 68080 | 1 | | 2,530 | 7.43 | .72 |
| Tabert | Cabb Orchard (run-of-mine, 185 tons) | Sewell | S. B. | 76798 | 1 | | 2,780 | 3.82 | .64 |
| Tommy Creek | Tommy Creek (run-of-mine, 1 car) | Pocahontas No. 3 | S. B. | 76108 | 1 | | 2,630 | 7.24 | .63 |
| Vanwood | Bectley No. 1 | Bectley | S. B. | 76338-40 | 3 | 2,510 | 2,880 | 4.53 | .71 |
| Do. | Wood-Sullivan (run-of-mine, 14 cars) | Bectley and Pocahontas No. 3 | S. B. | 76109 | 1 | | 2,740 | 16.09 | .63 |
| Do. | Wood-Sullivan No. 3 | Pocahontas No. 3 | S. B. | 76334-35 | 2 | 2,590 | 2,890 | 12.96 | .54 |
| Viscova | Viscova (run-of-mine, 2 cars) | Sewell | S. B. | 76380 | 1 | | 2,910 | | |
| Do. | Viscova | do. | S. B. | 77183-84 | 2 | 2,910 | | | |

TABLE I.—Softening temperatures of coal ash from coals of the United States—Continued.

| Locality. | Mine. | Coal bed. | Rank of coal. | Laboratory No. | Number of samples from mine. | Softening temperature ° F. | | | Average analysis of dry coal, percentage of— | |
|---------------------------|--|--|---------------|-------------------------|------------------------------|----------------------------|----------|----------|--|----------|
| | | | | | | Lowest. | Highest. | Average. | Ash. | Sulphur. |
| 1 | | | | | | | | | | |
| WEST VIRGINIA—Continued. | | | | | | | | | | |
| RALEIGH COUNTY—continued. | | | | | | | | | | |
| Whitby..... | Bowyer No. 1..... | Fire Creek No. 1..... | S. B. | 76884-85..... | 2 | 2,700 | 2,800 | 2,750 | 6.92 | 0.83 |
| Do..... | Bowyer No. 2..... | Beckley..... | S. B. | 76854..... | 1 | | | 2,820 | 10.46 | .75 |
| Do..... | Bowyer (run-of-mine, 3 cars)..... | Fire Creek, 80 per cent; Beckley, 20 per cent. | S. B. | 76816..... | 1 | | | 2,580 | 6.91 | .86 |
| Wiley..... | Pinney Creek (run-of-mine, 2 cars)..... | Beckley..... | S. B. | 76376..... | 1 | | | 2,460 | 10.76 | 1.42 |
| Winding Gulf..... | Winding Gulf No. 1..... | do..... | S. B. | 14368-70, 76035-37..... | 6 | 2,670 | +3,020 | +2,860 | 3.14 | .54 |
| Do..... | Winding Gulf No. 1 (run-of-mine, 1 car)..... | do..... | S. B. | 77637..... | 1 | | | 2,640 | 5.92 | .60 |
| Do..... | Winding Gulf No. 2..... | do..... | S. B. | 14372-73..... | 2 | 2,960 | 2,980 | 2,960 | 2.78 | .61 |
| Do..... | Winding Gulf No. 3..... | do..... | S. B. | 76928-30..... | 2 | 2,720 | 2,960 | 2,850 | 3.53 | .70 |
| Woodley..... | McAlpin No. 1..... | do..... | S. B. | 14301-03..... | 3 | 2,560 | +3,010 | +2,860 | 3.60 | .60 |
| Do..... | Woodley (run-of-mine, 12 cars)..... | do..... | S. B. | 77724..... | 1 | | | 2,910 | 6.91 | .60 |
| RANDOLPH COUNTY. | | | | | | | | | | |
| Mill Creek..... | Talbert & Spitzer..... | Sewell..... | B. | 13651..... | 1 | | | +3,010 | 11.43 | .62 |
| Spring..... | Hopkins..... | do..... | B. | 13601..... | 1 | | | +3,010 | 13.85 | .99 |
| WYOMING COUNTY. | | | | | | | | | | |
| Alpena..... | Alpena..... | Beckley..... | S. B. | 68147, 49, 69-68..... | 4 | 2,120 | 2,540 | 2,360 | 4.45 | 1.05 |
| Angelo..... | Angelo..... | Poconchos No. 3..... | S. B. | 76219-21..... | 3 | 2,760 | 2,960 | 2,860 | 5.47 | .78 |
| Beckley..... | Thermo Poconchos..... | Beckley..... | S. B. | 76300-02..... | 3 | 2,450 | 2,770 | 2,610 | 5.09 | 1.25 |
| Do..... | Thermo Poconchos (run-of-mine, 4 cars)..... | do..... | S. B. | 76301..... | 1 | | | 2,440 | 5.22 | 1.57 |
| Do..... | Smith Poconchos..... | Poconchos No. 2..... | S. B. | 76302..... | 3 | 2,550 | 2,900 | 2,720 | 7.06 | .95 |
| Do..... | Smith Poconchos (run-of-mine, 1 car)..... | do..... | S. B. | 76303..... | 1 | | | 2,610 | 11.61 | .71 |

| | | | | | | | | | | |
|------------------|--|---|--------|------------|---|-------|-------|-------|-------|------|
| Do. | Miller Pocahontas. | do. | S. B. | 76157-59. | 3 | 2,810 | 2,860 | 2,840 | 5.44 | .66 |
| Do. | Virginia Smokeless. | do. | S. B. | 76151-54. | 4 | 2,820 | 2,910 | 2,850 | 6.21 | .72 |
| Do. | Virginia Smokeless (run-of-mine, tippie sample). | do. | S. B. | 76917. | 1 | | | 2,960 | 11.85 | .65 |
| Osbb Orchard. | | | | | | | | | | |
| Devil's Fork. | Sewell | do. | S. B. | 76911-12. | 2 | 2,440 | 2,620 | 2,530 | 4.67 | 1.04 |
| Do. | Pocahontas No. 3. | do. | S. B. | 76913. | 1 | | | 2,910 | 6.53 | .70 |
| Do. | Devil's Fork. | do. | S. B. | 76938-40. | 3 | 2,920 | 2,910 | 2,910 | 6.41 | .65 |
| Herndon. | Clagette. | Pocahontas No. 6. | S. B. | 76937-38. | 2 | 2,450 | 2,700 | 2,580 | 5.98 | .68 |
| Do. | Flat Top (run-of-mine, 2 cars). | Pocahontas Nos. 3 and 6. | S. B. | 76946. | 1 | | | 2,740 | 10.66 | .76 |
| Do. | Flat Top. | Pocahontas No. 3. | S. B. | 76936-37. | 3 | 2,740 | 2,800 | 2,760 | 6.01 | .52 |
| Iroquois. | Iroquois (run-of-mine, 5 cars). | do. | S. B. | 76934. | 1 | | | 2,900 | 8.12 | .66 |
| Do. | Morris Smokeless No. 1. | do. | S. B. | 76924-26. | 3 | 2,800 | 2,900 | 2,860 | 6.43 | .69 |
| Mercer. | Mullens (run-of-mine, 1 tippie sample car). | do. | S. B. | 76162-63. | 2 | 2,800 | 2,800 | 2,800 | 6.36 | .57 |
| Mullens. | Mullens Smokeless. | do. | S. B. | 76913. | 1 | | | 2,850 | 11.84 | .79 |
| Do. | Otego. | do. | S. B. | 76876-77. | 2 | 2,850 | 3,000 | 2,930 | 6.43 | .71 |
| Do. | Otego. | do. | S. B. | 76940-42. | 3 | 2,680 | 2,940 | 2,880 | 9.22 | .82 |
| Do. | Sabine. | Beckley | S. B. | 76900-02. | 3 | 2,260 | 2,460 | 2,350 | 4.41 | .88 |
| Do. | Sabine (run-of-mine tippie sample). | do. | S. B. | 76918. | 1 | | | 2,560 | 10.17 | .73 |
| Tracal. | Trace Fork. | Pocahontas No. 3. | S. B. | 76901-03. | 3 | 2,910 | 3,010 | 2,940 | 6.67 | .72 |
| Do. | Trace Fork (run-of-mine, 6 cars). | do. | S. B. | 76938. | 1 | | | 2,840 | 12.27 | .94 |
| Do. | Barker's Creek No. 1. | do. | S. B. | 76948-46. | 3 | 2,570 | 2,620 | 2,600 | 4.81 | .71 |
| Do. | Barker's Creek No. 2. | Pocahontas No. 6. | S. B. | 76914-15. | 2 | 2,390 | 2,570 | 2,480 | 6.63 | .63 |
| Do. | Barker's Creek No. 2 (run-of-mine, 1 car). | do. | S. B. | 76876. | 1 | | | 2,510 | 6.38 | .51 |
| Do. | Barker's Creek No. 3. | Beckley | S. B. | 76908-40. | 3 | 2,570 | 2,620 | 2,600 | 5.37 | 1.20 |
| Do. | Barker's Creek No. 3 (run-of-mine, 1 car). | Pocahontas No. 6. | S. B. | 76991. | 1 | | | 2,880 | 10.01 | .49 |
| Do. | Cooper No. 1 (run-of-mine, 8 cars). | Beckley, 27 percent, and Pocahontas No. 3, 73 per cent. | S. B. | 76768. | 1 | | | 2,620 | 9.88 | .86 |
| Do. | Mead-Pocahontas. | Pocahontas No. 3. | S. B. | 76871-873. | 3 | 2,620 | 2,980 | 2,770 | 6.18 | .78 |
| Do. | Mead - Pocahontas (run-of-mine, 5 cars). | do. | S. B. | 76962. | 1 | | | 2,800 | 6.60 | .74 |
| Do. | Trail No. 2. | Pocahontas No. 3. | S. B. | 76917-19. | 3 | 2,740 | 2,860 | 2,800 | 5.93 | .64 |
| Do. | Trail No. 2 (run-of-mine, 9 cars). | do. | S. B. | 76960. | 1 | | | 2,620 | 7.93 | .67 |
| Wyco. | Wyco. | do. | S. B. | 76938-47. | 5 | 2,850 | 3,000 | 2,940 | 7.41 | .63 |
| Do. | Wyco (run-of-mine, tippie sample). | do. | S. B. | 76920. | 1 | | | 2,910 | 8.39 | .70 |
| WYOMING. | | | | | | | | | | |
| CAMPBELL COUNTY. | | | | | | | | | | |
| Gillette. | Country bank | Unnamed | Sb. B. | 22972. | 1 | | | 2,080 | 14.66 | 2.06 |
| Do. | Local | "B" | Sb. B. | 22886. | 1 | | | 2,350 | 11.03 | 1.96 |
| Do. | Prospect | "A" (?) | Sb. B. | 22904. | 1 | | | 2,250 | 9.31 | 1.76 |
| CARBON COUNTY. | | | | | | | | | | |
| Hanna. | Local | Unnamed. | Sb. B. | 22972. | 1 | | | 2,110 | 5.18 | 1.20 |
| Medicine Bow. | Johnson. | do. | Sb. B. | 22900. | 1 | | | 2,150 | 7.91 | .98 |
| Rock River. | King. | do. | Sb. B. | 22762. | 1 | | | 2,450 | 9.99 | 1.52 |

TABLE I.—Softening temperatures of coal ash from coals of the United States—Continued.

| Locality. | Mine. | Coal bed. | Rank of coal. | Laboratory No. | Number of samples from mine. | Softening temperature ° F. | | | Average analysis of— | |
|------------------|--------------|-----------------|---------------|-------------------------|------------------------------|----------------------------|----------|----------|----------------------|----------|
| | | | | | | Lowest. | Highest. | Average. | Ash. | Sulphur. |
| 1 | | | | | | | | | | |
| WYOMING—Con. | | | | | | | | | | |
| CONVERSE COUNTY. | | | | | | | | | | |
| Big Muddy | Big Muddy | Lower Big Muddy | Sb. B. | 17723 | 1 | | | 2,100 | 9.15 | 0.77 |
| Do. | do. | Upper Big Muddy | Sb. B. | 17722 | 1 | | | 2,240 | 6.37 | .97 |
| Douglas | Outcrop | Lower Burned | Sb. B. | 11442 | 1 | | | 2,090 | 11.65 | .60 |
| Glenrock | Country bank | Unnamed | Sb. B. | 11048 | 1 | | | 2,360 | 6.42 | .63 |
| Do. | Fairview | do. | Sb. B. | 17667 | 1 | | | 2,210 | 7.15 | .92 |
| Do. | Glenrock | do. | Sb. B. | 17668 | 1 | | | 2,110 | 7.79 | .92 |
| Do. | Prospect | do. | Sb. B. | 17602 | 1 | | | 2,110 | 9.61 | .55 |
| Do. | Diamond | do. | Sb. B. | 14781 | 1 | | | 2,110 | 15.30 | 1.68 |
| Do. | Ines | do. | Sb. B. | 14784 | 1 | | | 2,180 | 11.54 | .98 |
| Do. | Harney Creek | do. | L. | 10911 | 1 | | | 1,990 | 10.49 | 1.13 |
| Do. | Haynes | do. | Sb. B. | 10711 | 1 | | | 2,310 | 7.77 | 1.57 |
| Do. | Onyon | do. | Sb. B. | 10740 | 1 | | | 2,010 | 11.50 | 1.23 |
| Do. | Prospect | "D" | Sb. B. | 11447 | 1 | | | 2,260 | 6.93 | .87 |
| Do. | Rosin | Unnamed | Sb. B. | 10775 | 1 | | | 2,170 | 13.81 | 1.42 |
| Do. | Sunset | do. | Sb. B. | 10686 | 1 | | | 2,220 | 16.05 | .80 |
| CROOK COUNTY. | | | | | | | | | | |
| Sundance | Beishe | Unnamed | B. | 15713-15 | 3 | 2,150 | 2,490 | 2,260 | 14.99 | 6.09 |
| FREMONT COUNTY. | | | | | | | | | | |
| Deeds | Prospect | Unnamed | Sb. B. | 17684 | 1 | | | 2,180 | 15.84 | 4.00 |
| Hudson | Hieley | do. | Sb. B. | 17947 | 1 | | | 2,160 | 5.46 | .90 |
| Do. | Indian | do. | Sb. B. | 10638-41, 43, 13178-79. | 7 | 2,000 | 2,260 | 2,130 | 7.26 | .64 |
| Do. | McKinley | Mass Verde | Sb. B. | 21408-10. | 3 | 2,090 | 2,280 | 2,140 | 9.45 | 1.46 |
| Do. | Mitchell | Unnamed | Sb. B. | 9773, 17248. | 2 | 2,280 | 2,310 | 2,300 | 9.55 | .98 |
| Do. | Propolis | Lander | L. | 17244-45. | 2 | 2,270 | 2,280 | 2,280 | 5.61 | .70 |
| Elverton | Shipton | Unnamed | Sb. B. | 9772 | 1 | | | 2,220 | 9.42 | .90 |

[illegible]

TABLE I.—Softening temperatures of coal ash from coals of the United States—Continued.

| Locality. | Mine. | Coal bed. | Rank of coal. | Laboratory No. | Number of samples from mine. | Softening temperature °F. | | | Average analysis of— | |
|----------------------------|------------------------------|---------------------|---------------|---|------------------------------|---------------------------|----------|----------|----------------------|----------|
| | | | | | | Lowest. | Highest. | Average. | Ash. | Sulphur. |
| 1 | | | | | | | | | | |
| WYOMING—Coal. | | | | | | | | | | |
| SHERIDAN COUNTY—continued. | | | | | | | | | | |
| Kool..... | Hughey prospect..... | Monarch (?)..... | Sb. B. | 12642..... | 1 | | | 2 150 | 6.48 | 0.41 |
| Do..... | Kool..... | Monarch..... | Sb. B. | 10818-21, 10060-66, 20263-78..... | 23 | 1,970 | 2,240 | 2,080 | 6.36 | .83 |
| Monarch..... | Monarch..... | do..... | Sb. B. | 12005-06, 08, 19006-08, 54, 26190-91, 20244-50, 62, 63, 20311-13..... | 21 | 1,970 | 2,260 | 2,190 | 4.85 | .66 |
| Do..... | New Monarch..... | do..... | Sb. B. | 12685, 16830-36..... | 8 | 2,000 | 2,460 | 2,180 | 4.92 | .69 |
| New Acme..... | New Acme..... | do..... | Sb. B. | 16056-58..... | 4 | 2,150 | 2,450 | 2,320 | 4.33 | .67 |
| SWEETWATER COUNTY. | | | | | | | | | | |
| Gunn..... | Gunn-Quealy B. Prospect..... | No. 11..... | Sb. B. | 11807-09..... | 3 | 1,900 | 2,080 | 1,970 | 2.44 | 1.06 |
| Rock Springs..... | Typical..... | Typical..... | Sb. B. | 22832..... | 1 | | | 1,960 | 16.33 | 7.23 |
| Superior..... | No. 1 upper..... | No. 1 upper..... | B. | 10087..... | 1 | | | 2,040 | 2.91 | 1.03 |
| Do..... | B..... | No. 7..... | Sb. B. | 11461-64..... | 4 | 2,190 | 2,410 | 2,330 | 3.92 | 1.05 |
| Do..... | C..... | No. 1..... | Sb. B. | 11466-69..... | 4 | 2,100 | 2,390 | 2,280 | 3.76 | 1.12 |
| UNTA COUNTY. | | | | | | | | | | |
| Elkol..... | Elkol..... | No. 1..... | B. | 11730-31..... | 2 | 2,380 | 2,400 | 2,360 | 4.61 | .75 |
| Frontier..... | Kemmerer No. 1..... | do..... | B. | 11569-68..... | 4 | 2,060 | 2,190 | 2,120 | 7.13 | 1.35 |
| Kemmerer..... | Prospect..... | Unnamed..... | Sb. B. | 11740..... | 1 | | | 1,920 | 3.08 | 2.38 |
| Suda..... | Kemmerer No. 4..... | Kemmerer No. 1..... | B. | 13517-18..... | 2 | 2,040 | 2,270 | 2,160 | 6.82 | 1.42 |
| WESTON COUNTY. | | | | | | | | | | |
| Cambria..... | Antelope No. 3..... | Unnamed..... | Sb. B. | 16428-30..... | 3 | 2,600 | 2,980 | 2,700 | 18.38 | 6.40 |
| Do..... | Antelope No. 4..... | do..... | Sb. B. | 16412-14..... | 3 | 2,780 | 2,870 | 2,810 | 16.65 | 5.55 |
| Monarch..... | Prospect..... | do..... | Sb. B. | 12446..... | 1 | | | 2,400 | 8.13 | .76 |
| Do..... | do..... | do..... | Sb. B. | 12392..... | 1 | | | 2,310 | 9.62 | 1.31 |

TABLE II.—Comparison of softening temperatures of coal ash from mine and car samples.

| State. | County. | Town. | Mine. | Coal bed. | Type of coal. | Laboratory No. | Number of samples. | Average softening temperature, F. | Average analysis of dry coal, percent— | |
|----------------|------------|----------------|--|---------------------------|---------------|--------------------------|--------------------|-----------------------------------|--|----------|
| | | | | | | | | | Ash. | Sulphur. |
| 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 |
| Illno. | Franklin. | Orient. | Orient (mine samples)..... | No. 6 (Herrin)..... | B. | 23442-43..... | 3 | 2,410 | 8.99 | 0.93 |
| Do. | do. | do. | Orient (egg and lump, 32 cars)..... | do. | B. | 30260-67..... | 3 | 2,190 | 9.79 | 1.29 |
| Do. | do. | Sesser. | Sesser No. 1 (mine samples)..... | do. | B. | 25492-96..... | 5 | 2,210 | 9.30 | 1.21 |
| Do. | do. | do. | Sesser No. 1 (egg and nut, 14 cars)..... | do. | B. | 30206, 08, 30485-86..... | 4 | 2,360 | 9.84 | 1.23 |
| Do. | Macoupin. | Gillespie. | Superior No. 1 (mine samples)..... | do. | B. | 18545-46..... | 2 | 2,150 | 11.36 | 4.59 |
| Do. | do. | do. | Superior No. 1 (run-of-mine) (car samples)..... | do. | B. | 18910..... | 1 | 2,040 | 15.11 | 5.75 |
| Maryland. | Alleghany. | Midland. | Consolidation No. 8 (mine samples)..... | Pittsburgh (Big Bed)..... | S. B. | 68192-95..... | 2 | + 3,010 | 7.35 | .72 |
| Do. | do. | do. | Consolidation No. 8 (tippie samples, run-of-mine)..... | do. | S. B. | 68303-09-11-13..... | 4 | + 3,010 | 7.33 | .88 |
| Pennsylvania. | Blair. | Glen White. | Glen White No. 2 (mine samples)..... | Upper Freeport..... | B. | 30830-32..... | 3 | 2,240 | 6.87 | 1.06 |
| Do. | do. | do. | Glen White No. 2 (tippie sample)..... | do. | B. | 30834..... | 1 | 2,120 | 9.61 | 1.06 |
| Do. | Cambria. | Gallitzin. | Gallitzin (mine samples)..... | do. | B. | 30824-27..... | 4 | 2,330 | 6.27 | 1.43 |
| Do. | do. | do. | Gallitzin (car sample)..... | do. | B. | 30829..... | 1 | 2,490 | 8.79 | 1.47 |
| Do. | do. | Nanty Glow. | Lincoln No. 1 (mine sample)..... | Lower Kittanning..... | S. B. | 12180-84, 22514-19..... | 11 | 2,370 | 6.47 | 1.84 |
| Do. | do. | do. | Lincoln No. 1 (car sample)..... | do. | S. B. | 22521..... | 1 | 2,150 | 7.75 | 2.54 |
| Do. | Indiana. | Scott Glen. | Brush Valley (mine samples)..... | do. | S. B. | 22501-05..... | 5 | 2,260 | 8.09 | 2.99 |
| Do. | do. | do. | Brush Valley (car sample)..... | do. | S. B. | 22507..... | 1 | 2,150 | 8.16 | 3.00 |
| Do. | Somerset. | Holsepple. | Haws No. 3 (mine sample)..... | do. | S. B. | 22541-45..... | 5 | 2,370 | 8.00 | 2.48 |
| Do. | do. | do. | Haws No. 3 (run-of-mine, car sample)..... | do. | S. B. | 22547..... | 1 | 2,370 | 9.07 | 2.71 |
| Do. | do. | Windber. | Lechrie Arrow (mine samples)..... | do. | S. B. | 20292-95, 98..... | 5 | + 2,520 | 6.79 | 1.44 |
| Do. | do. | do. | Lechrie Arrow (run-of-mine, car sample)..... | do. | S. B. | 20334..... | 1 | 2,460 | 10.25 | 2.16 |
| Virginia. | Tazewell. | Pocahontas. | West Pocahontas (mine samples)..... | Pocahontas No. 3..... | S. B. | 67910, 53-54, 55..... | 4 | 2,320 | 4.52 | .46 |
| Do. | do. | do. | West Pocahontas (run-of-mine, 4 cars)..... | do. | S. B. | 30284..... | 1 | 2,190 | 5.60 | .67 |
| West Virginia. | Fayette. | Glen Jean. | Nichol (mine samples)..... | Sewell..... | S. B. | 76951-64..... | 4 | 2,550 | 5.30 | .78 |
| Do. | do. | do. | Nichol (run-of-mine, 5 cars)..... | do. | S. B. | 76980..... | 1 | 2,500 | 5.25 | .75 |
| Do. | do. | Ingram Branch. | Ingram Branch No. 2 (mine samples)..... | Big Eagle..... | B. | 77228-36..... | 4 | 2,660 | 5.05 | 1.08 |
| Do. | do. | do. | Ingram Branch No. 2 (run-of-mine, 14 cars)..... | do. | B. | 76504..... | 1 | 2,780 | 8.36 | .99 |
| Do. | do. | Lochelly. | Lochelly (mine samples)..... | Sewell..... | B. | 77229-32..... | 4 | 2,490 | 2.95 | .51 |
| Do. | do. | do. | Lochelly (run-of-mine, 6 cars)..... | do. | B. | 77000..... | 1 | 2,600 | 4.95 | .56 |
| Do. | do. | Long Branch. | Long Branch No. 1 (mine samples)..... | Eagle..... | B. | 77215-17..... | 3 | 2,780 | 3.56 | .69 |

TABLE II.—Comparison of softening temperatures of coal ash from mine and car samples—Continued.

| State. | County. | Town. | Mine. | Coal bed. | Type of coal. | Laboratory No. | Number of samples. | Average softening temperature, °F. | Average analysis of dry coal, percentage of— | |
|----------------|-----------|---------------------|--|---|---------------|---|--------------------|------------------------------------|--|----------|
| | | | | | | | | | Ash. | Sulphur. |
| 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 |
| West Virginia. | Payette | Long Branch | Long Branch No. 2 (mine samples). | No. 2 Gas. | B. | 7718-22. | 5 | 2,810 | 4.63 | 0.81 |
| Do. | do. | do. | Long Branch Nos. 1 and 2 (run-of-mine, 16 cars). | Eagle, 23 per cent; and No. 2 Gas, 77 per cent. | B. | 7648. | 1 | 2,890 | 8.46 | .86 |
| Do. | do. | Mount Hope. | Catherine (mine samples). | Sewell. | S. B. | 77169-70. | 2 | 2,490 | 4.38 | 1.12 |
| Do. | do. | do. | Catherine (run-of-mine, 25 tons). | do. | S. B. | 75393. | 1 | 2,450 | 7.13 | 1.08 |
| Do. | do. | do. | Minnie Bell (mine samples). | do. | S. B. | 77173-74. | 2 | 2,560 | 3.96 | .91 |
| Do. | do. | do. | Minnie Bell (run-of-mine, 1 car). | do. | S. B. | 76377. | 1 | 2,450 | 4.74 | 1.23 |
| Do. | do. | do. | Warner (mine sample). | do. | S. B. | 77171-72. | 2 | 2,520 | 4.60 | 1.62 |
| Do. | do. | do. | Warner (run-of-mine, 30 tons). | do. | S. B. | 76392. | 1 | 2,270 | 9.96 | 1.86 |
| Do. | do. | Summerlee. | Summerlee (mine sample). | do. | B. | 77179-82. | 4 | 2,370 | 2.60 | .81 |
| Do. | do. | do. | Summerlee (run-of-mine, 7 cars). | do. | B. | 77091. | 1 | 2,310 | 4.22 | .70 |
| Do. | do. | Sun. | Sunset (mine sample). | do. | S. B. | 77175-76. | 2 | 2,340 | 3.97 | .92 |
| Do. | do. | do. | Sunset (run-of-mine, 1 car). | do. | S. B. | 76807. | 1 | 2,300 | 5.91 | .74 |
| Do. | do. | Willis Branch. | Willis Branch (mine sample). | Alma. | B. | 76931-33. | 3 | 2,900 | 5.55 | .76 |
| Do. | do. | do. | Willis Branch (run-of-mine, 1 car). | do. | B. | 77165. | 1 | 2,850 | 10.60 | .71 |
| Do. | McDowell. | Gilliam. | Gilliam (mine sample). | Pocahontas No. 3. | S. B. | 69000, 022-23. | 3 | 2,250 | 5.40 | .45 |
| Do. | do. | do. | Gilliam (run-of-mine, 4 cars). | do. | S. B. | 30238. | 1 | 2,260 | 8.05 | .43 |
| Do. | do. | Hartwell. | Berwind No. 5 (mine sample). | do. | S. B. | 69066, 72, 76. | 3 | 2,200 | 4.17 | .68 |
| Do. | do. | do. | Berwind No. 5 (run-of-mine, 4 cars). | do. | S. B. | 30283. | 1 | 2,310 | 9.55 | .96 |
| Do. | do. | Havaco. | Havaco (mine sample). | do. | S. B. | 10034, 13775, 67996, 69019-20. | 5 | 2,350 | 5.66 | .74 |
| Do. | do. | do. | Havaco (run-of-mine, 4 cars). | do. | S. B. | 30237. | 1 | 2,240 | 8.19 | .81 |
| Do. | do. | Jenkin Jones No. 6. | Pocahontas No. 3 (mine sample). | do. | S. B. | 14185 - 87, 89 - 90, 20518-23, 69063, 90, 96. | 14 | 2,260 | 5.03 | .51 |
| Do. | do. | do. | Pocahontas No. 3 (car sample, 8 cars). | do. | S. B. | 30256. | 1 | 2,280 | 5.9 | .50 |
| Do. | do. | Jenkin Jones No. 8. | Pocahontas No. 3 (mine samples). | do. | S. B. | 20531-35, 69038-39, 93, 95. | 9 | 2,320 | 5.16 | .59 |
| Do. | do. | do. | Pocahontas No. 3 (run-of-mine, 24 cars). | do. | S. B. | 30239. | 1 | 2,280 | 6.88 | .54 |
| Do. | do. | Lick Branch. | Delta (mine sample). | do. | S. B. | 67891, 94, 967. | 3 | 2,210 | 4.30 | .45 |

| | | | | | | | | | | |
|-----|----------|-----|--|-----------------|-------|-----------------------------------|---|--------|-------|------|
| Do. | do. | do. | Delta (carsample, 11 cars) | do. | S. B. | 30260 | 1 | 2,370 | 7.71 | .50 |
| Do. | Newhall | do. | Berwind No. 6 (mine sample) | do. | S. B. | 68066-75 | 2 | 2,810 | 4.90 | .63 |
| Do. | do. | do. | Berwind No. 6 (run-of-mine, 9 cars) | do. | S. B. | 30265 | 1 | 2,660 | 9.53 | .80 |
| Do. | do. | do. | Berwind No. 8 (mine sample) | do. | S. B. | 68071, 75-79 | 3 | 2,660 | 4.43 | .65 |
| Do. | do. | do. | Berwind No. 8 (run-of-mine, 8 cars) | do. | S. B. | 30262 | 1 | 2,700 | 7.59 | .70 |
| Do. | do. | do. | Mahor or No. 4 (carsamples, run-of-mine) | Sewell | S. B. | 22480-85, 89 | 2 | 2,660 | 4.46 | .71 |
| Do. | do. | do. | do. | do. | S. B. | 22487-88 | 8 | 2,680 | 4.31 | .71 |
| Do. | do. | do. | King (mine samples) | Poahontas No. 3 | S. B. | 68024, 25, 27, 30 | 4 | +2,420 | 4.85 | .44 |
| Do. | Vivian | do. | King (car samples, 11 cars) | do. | S. B. | 30261 | 1 | 2,280 | 8.05 | .56 |
| Do. | do. | do. | Tidewater (mine samples) | do. | S. B. | 68035, 8-9 | 3 | 2,270 | 5.63 | .55 |
| Do. | do. | do. | Tidewater (run-of-mine, 15 cars) | do. | S. B. | 30266 | 1 | 2,330 | 8.55 | .55 |
| Do. | do. | do. | Weyanoke No. 1 (mine sample) | do. | S. B. | 14224-26, 28, 67336, 945-66 | 7 | 2,660 | 4.00 | .58 |
| Do. | Weyanoke | do. | do. | do. | S. B. | 30233 | 1 | 2,600 | 8.33 | .58 |
| Do. | do. | do. | Weyanoke No. 1 (car sample, 9 cars) | do. | S. B. | 24891-93, 95, 68115, 122, 123 | 7 | +3,000 | 4.19 | .56 |
| Do. | Raleigh | do. | Affinity (mine samples) | Beckley | S. B. | 79004 | 1 | 2,900 | 7.45 | .75 |
| Do. | do. | do. | Affinity (run-of-mine, 5 cars) | do. | S. B. | 78922 | 1 | 2,510 | 5.25 | .75 |
| Do. | do. | do. | Battleship (mine sample) | Fire Creek | S. B. | 78197 | 1 | 2,510 | 6.71 | .88 |
| Do. | Beaumont | do. | Battleship (run-of-mine, 3 cars) | do. | S. B. | 77185-86 | 2 | 2,450 | 2.23 | .70 |
| Do. | do. | do. | City No. 1 (mine samples) | do. | S. B. | 78399 | 1 | 2,430 | 5.70 | .83 |
| Do. | Beckley | do. | Clyde Poahontas (mine samples) | Poahontas No. 3 | S. B. | 78265-66 | 2 | 2,620 | 4.10 | .66 |
| Do. | do. | do. | Clyde Poahontas (run-of-mine, 24 cars) | do. | S. B. | 78322 | 1 | 2,850 | 10.21 | .66 |
| Do. | do. | do. | Big Stick (mine samples) | Beckley | S. B. | 28684 - 86, 88 - 89, 68118, 94-95 | 8 | +2,950 | 4.20 | .55 |
| Do. | do. | do. | Big Stick (run-of-mine, 84 cars) | do. | S. B. | 78989 | 1 | 2,870 | 7.61 | .57 |
| Do. | do. | do. | Eccles No. 5 (mine samples) | do. | S. B. | 19774-76 | 5 | +2,620 | 6.44 | .78 |
| Do. | do. | do. | Eccles No. 5 (run-of-mine, 8 cars) | do. | S. B. | 78922 | 1 | 2,900 | 6.44 | .79 |
| Do. | do. | do. | Douglas No. 2 (mine sample) | Fire Creek | S. B. | 78857-60 | 4 | 2,660 | 4.08 | .94 |
| Do. | do. | do. | Douglas No. 2 (run-of-mine, 10 cars) | do. | S. B. | 77728 | 1 | 2,750 | 5.88 | .96 |
| Do. | do. | do. | Leckie No. 2 (mine samples) | do. | S. B. | 78907-09 | 3 | 2,760 | 4.60 | .80 |
| Do. | do. | do. | Leckie No. 2 (run-of-mine, 6 cars) | do. | S. B. | 77495 | 1 | 2,570 | 7.15 | .82 |
| Do. | do. | do. | Lillybrook No. 2 (mine samples) | do. | S. B. | 76481-83 | 3 | 2,720 | 4.11 | .91 |
| Do. | do. | do. | Lillybrook No. 2 (run-of-mine, 6 cars) | do. | S. B. | 77722 | 1 | 2,600 | 5.67 | 1.09 |
| Do. | do. | do. | Gulf (Hotcoal) (mine samples) | Beckley | S. B. | 78924-27, 68 | 5 | +2,810 | 4.05 | .72 |
| Do. | do. | do. | Gulf (Hotcoal) (run-of-mine, 10 cars) | do. | S. B. | 78568 | 1 | 2,960 | 6.50 | .62 |
| Do. | do. | do. | Laurel Smokeless (mine samples) | Poahontas No. 3 | S. B. | 78542-43 | 2 | 2,780 | 4.49 | .82 |
| Do. | do. | do. | Laurel Smokeless (run-of-mine, 2 cars) | do. | S. B. | 78685 | 1 | 2,830 | 8.94 | .74 |
| Do. | do. | do. | Neal (mine samples) | Sewell | S. B. | 78951-52 | 2 | 2,570 | 4.88 | 1.00 |
| Do. | do. | do. | Neal (run-of-mine, 2 cars) | do. | S. B. | 78660 | 1 | 2,810 | 12.43 | 1.37 |
| Do. | do. | do. | Lillybrook No. 1 (mine samples) | Fire Creek | S. B. | 78348-401 | 4 | 2,660 | 4.79 | .97 |
| Do. | do. | do. | Lillybrook No. 1 (run-of-mine, 84 cars) | do. | S. B. | 76241 | 1 | 2,770 | 6.22 | .86 |

TABLE II.—(Comparison of softening temperatures of coal ash from mine and car samples—Continued.)

| State. | County. | Town. | Mine. | Coal bed. | Type of coal. | Laboratory No. | Number of samples. | Average softening temperature, °F. | Average analysis of dry coal, percentage of— | |
|---------------|---------|--------------|---|------------------|---------------|---------------------------------|--------------------|------------------------------------|--|----------|
| | | | | | | | | | Ash. | Sulphur. |
| 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 |
| West Virginia | Raleigh | Lillybrook | Lilly and Hornbrook No. 3 (mine samples) | Poconantas No. 6 | S. B. | 76581-83 | 3 | 2,760 | 5.13 | 0.88 |
| Do. | do. | do. | Lilly and Hornbrook No. 3 (run-of-mine, 3 cars) | do. | S. B. | 76182 | 1 | 2,670 | 9.98 | .90 |
| Do. | do. | McAlpin | McAlpin (mine samples) | Beckley | S. B. | 69389-90 | 2 | +2,470 | 2.05 | .65 |
| Do. | do. | do. | McAlpin (run-of-mine, 21 cars) | do. | S. B. | 77720 | 1 | +2,780 | 7.03 | .64 |
| Do. | do. | Pemberton | Pemberton (mine samples) | do. | S. B. | 76582-84 | 3 | +2,920 | 5.48 | .67 |
| Do. | do. | do. | Pemberton, (run-of-mine, 44 cars) | do. | S. B. | 76587 | 1 | 2,120 | 7.34 | .76 |
| Do. | do. | do. | Reiland (mine samples) | do. | S. B. | 77223-26 | 3 | +2,930 | 6.20 | .69 |
| Do. | do. | do. | Reiland (run-of-mine, 8 cars) | do. | S. B. | 77231 | 1 | 2,400 | 7.46 | .66 |
| Do. | do. | Raleigh | Raleigh No. 3 (mine samples) | do. | S. B. | 10234-37 | 3 | 2,670 | 2.97 | .73 |
| Do. | do. | do. | Raleigh No. 3 (run-of-mine, 3 cars) | do. | S. B. | 77683 | 1 | 2,780 | 4.13 | .71 |
| Do. | do. | Shelton | Granberry No. 3 (mine samples) | Sewell | S. B. | 76672-76 | 4 | 2,660 | 2.99 | .73 |
| Do. | do. | do. | Granberry No. 3 (run-of-mine, 9 cars) | do. | S. B. | 76674 | 1 | 2,740 | 5.48 | .81 |
| Do. | do. | Slab Fork | Slab Fork No. 1 (mine samples) | Beckley | S. B. | 14387 | 1 | 2,480 | 2.87 | .61 |
| Do. | do. | do. | Slab Fork No. 1 (run-of-mine, 16 cars) | do. | S. B. | 77328 | 1 | 2,400 | 5.18 | .71 |
| Do. | do. | do. | Slab Fork No. 3 (mine samples) | do. | S. B. | 14329-26, 28-30, 31, 34, 35, 61 | 8 | +2,760 | 2.99 | .60 |
| Do. | do. | do. | Slab Fork No. 3 (run-of-mine, 34 cars) | do. | S. B. | 77330 | 1 | 2,400 | 5.17 | .69 |
| Do. | do. | Tamroy | Tamroy (mine samples) | Sewell | S. B. | 68199-98 | 9 | +2,580 | 1.97 | .60 |
| Do. | do. | do. | Tamroy (run-of-mine, 4 cars) | do. | S. B. | 77123-24 | 2 | 2,440 | 4.08 | .64 |
| Do. | do. | Viscova | Viscova (mine samples) | do. | S. B. | 77128-34 | 7 | 2,910 | 12.08 | .64 |
| Do. | do. | do. | Viscova (run-of-mine, 3 cars) | do. | S. B. | 76380 | 1 | 2,830 | 16.09 | .63 |
| Do. | do. | Winding Gulf | Winding Gulf No. 1 (mine samples) | Beckley | S. B. | 14388-70, 76585-87 | 6 | +2,760 | 2.14 | .64 |
| Do. | do. | do. | Winding Gulf No. 1 (run-of-mine, 11 cars) | do. | S. B. | 77337 | 1 | 2,640 | 5.92 | .60 |
| Do. | Wyoming | Bud | Thomso Poconantas (mine samples) | do. | S. B. | 76080-82 | 3 | 2,800 | 5.09 | 1.28 |
| Do. | do. | do. | Thomso Poconantas (run-of-mine, 8 cars) | do. | S. B. | 75901 | 1 | 2,440 | 8.23 | 1.67 |
| Do. | do. | Caloite | Smith Poconantas (mine samples) | Poconantas No. 3 | S. B. | 76888-91 | 3 | 2,400 | 7.08 | .98 |

| | | | | | | | | | | |
|---------|---------|-------------------|---|-----------------------|-------|---------------|---|-------|-------|-----|
| Do..... | do..... | do..... | Smith Pocahontas (run-of-mine,
7 cars). | do..... | S. B. | 78486..... | 1 | 2,910 | 13.43 | .75 |
| Do..... | do..... | Corrine..... | Virginia Smokeless (mine
samples). | do..... | S. B. | 78151-54..... | 4 | 2,890 | 6.21 | .72 |
| Do..... | do..... | do..... | Virginia Smokeless (run-of-mine,
tippie sample). | do..... | S. B. | 78917..... | 1 | 2,960 | 11.85 | .65 |
| Do..... | do..... | Devil's Fork..... | Devil's Fork (mine samples)..... | do..... | S. B. | 78289-90..... | 3 | 2,910 | 6.41 | .65 |
| Do..... | do..... | do..... | Devil's Fork (run-of-mine, 4 cars) | do..... | S. B. | 78012..... | 1 | 2,910 | 6.33 | .70 |
| Do..... | do..... | Iroquois..... | Iroquois (mine samples)..... | do..... | S. B. | 78224-25..... | 3 | 2,890 | 6.43 | .69 |
| Do..... | do..... | do..... | Iroquois (run-of-mine, 5 cars)..... | do..... | S. B. | 78984..... | 1 | 2,900 | 8.12 | .66 |
| Do..... | do..... | O'wego..... | Sabine (mine samples)..... | Beckley..... | S. B. | 78900-02..... | 3 | 2,890 | 4.41 | .88 |
| Do..... | do..... | do..... | Sabine (run-of-mine, tippie
sample). | do..... | S. B. | 78918..... | 1 | 2,890 | 10.17 | .72 |
| Do..... | do..... | Traceol..... | Trace Fork (mine samples)..... | Pocahontas No. 3..... | S. B. | 78861-63..... | 3 | 2,940 | 6.67 | .72 |
| Do..... | do..... | do..... | Trace Fork (run-of-mine, 6 cars). | do..... | S. B. | 78838..... | 1 | 2,940 | 12.27 | .94 |
| Do..... | do..... | Tracee..... | Barker's Creek No. 2 (mine
samples). | Pocahontas No. 6..... | S. B. | 78014-15..... | 3 | 2,480 | 5.63 | .53 |
| Do..... | do..... | do..... | Barker's Creek No. 2 (run-of-
mine, 1 car). | do..... | S. B. | 78878..... | 1 | 2,510 | 6.38 | .51 |
| Do..... | do..... | do..... | Tracee No. 2 (mine samples)..... | Pocahontas No. 3..... | S. B. | 78017-19..... | 3 | 2,800 | 5.53 | .64 |
| Do..... | do..... | do..... | Tracee No. 2 (run-of-mine, 9 cars). | do..... | S. B. | 78960..... | 1 | 2,690 | 7.93 | .67 |
| Do..... | do..... | Wyco..... | Wyco (mine samples)..... | do..... | S. B. | 78289-97..... | 5 | 2,940 | 7.41 | .63 |
| Do..... | do..... | do..... | Wyco (run-of-mine, tippie
sample). | do..... | S. B. | 78920..... | 1 | 2,910 | 8.89 | .70 |

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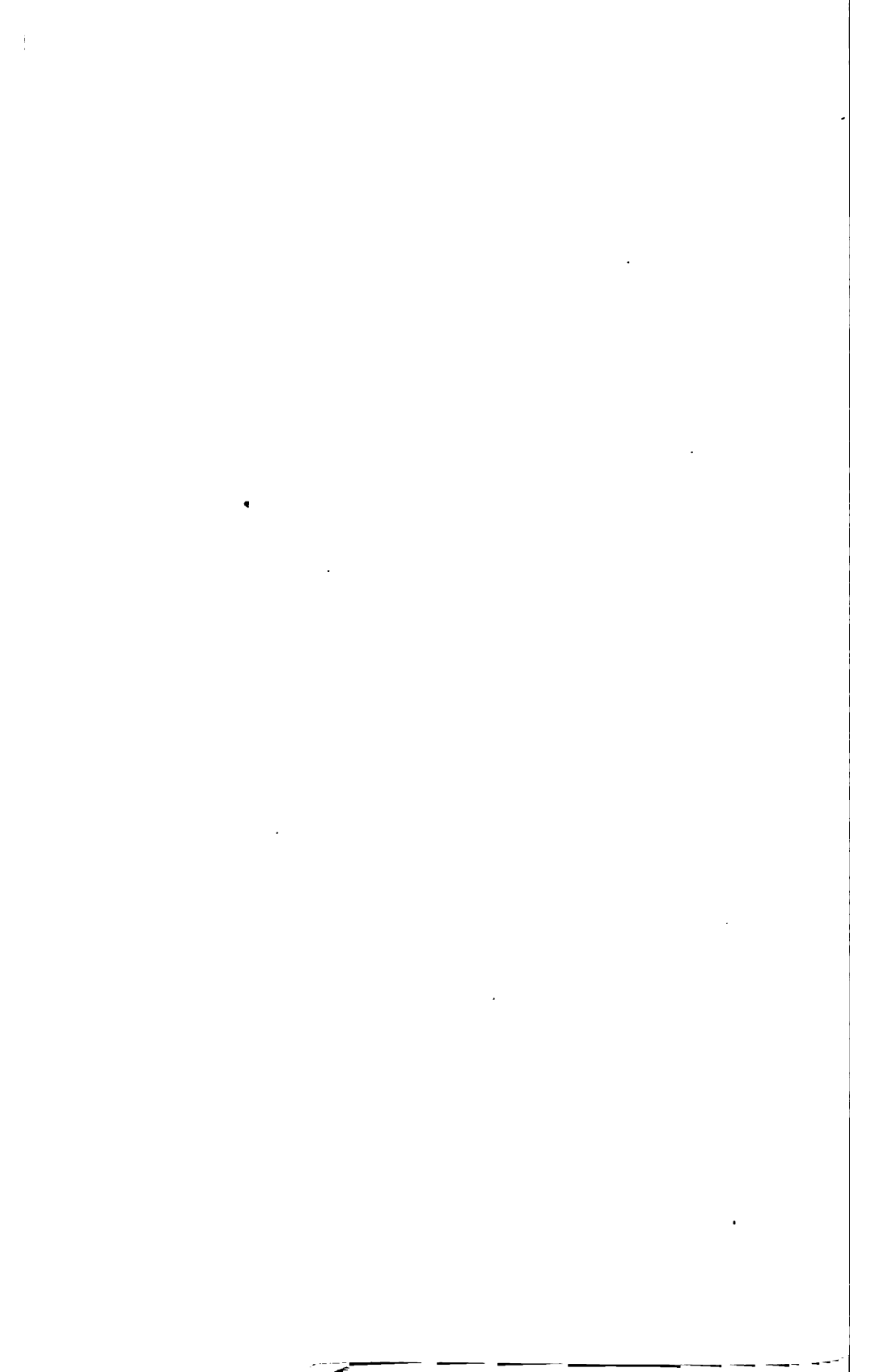
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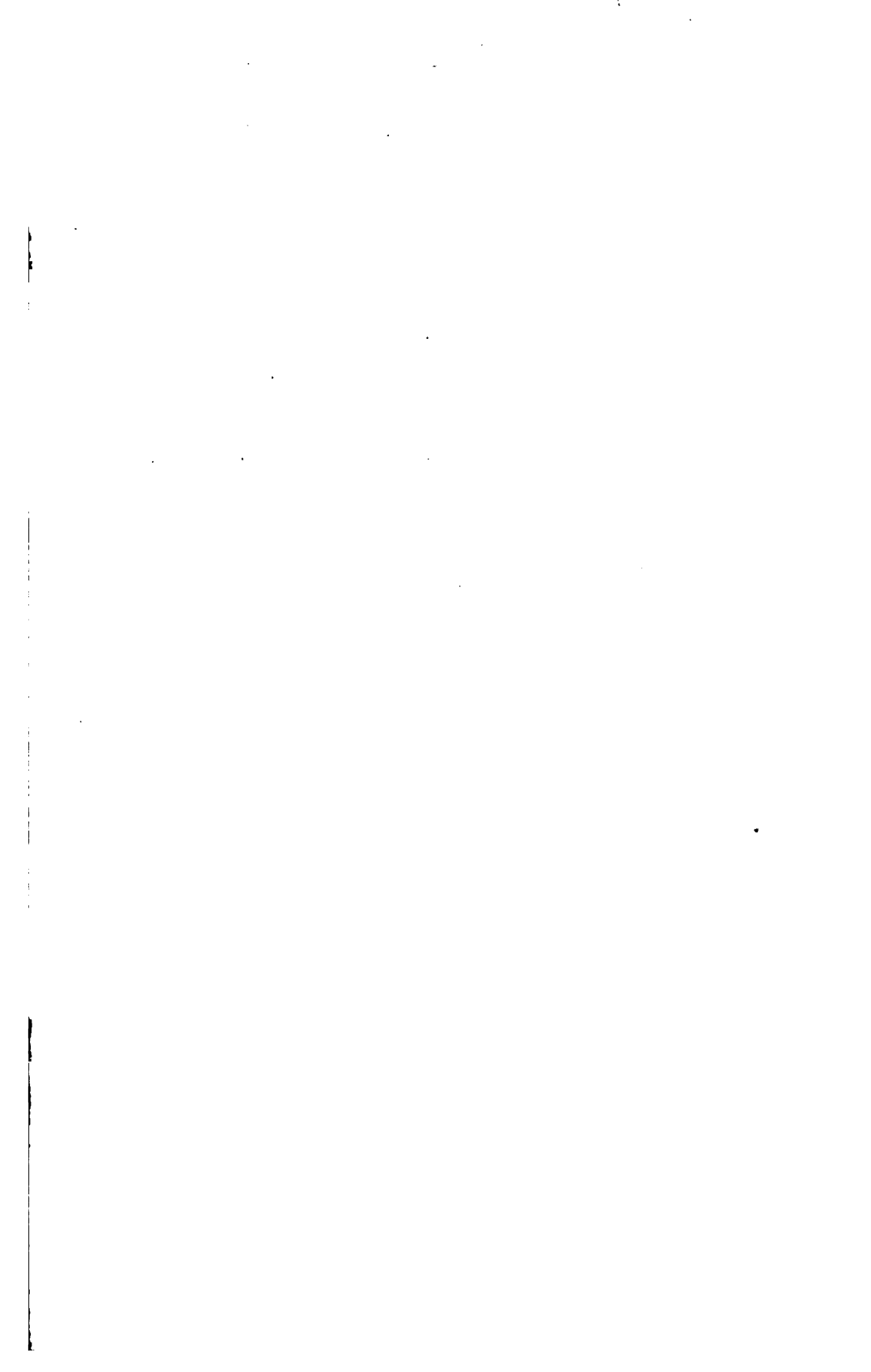
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TYPICAL OIL-SHALE FORMATION NEAR DE BEQUE, COLO.

Darker and more resistant beds are richer in shales.

DEPARTMENT OF THE INTERIOR

HUBERT WORK, SECRETARY

BUREAU OF MINES

H. FOSTER BAIN, DIRECTOR

OIL SHALE

AN HISTORICAL, TECHNICAL, AND ECONOMIC STUDY

BY

MARTIN J. GAVIN,

**THIS PAPER REPRESENTS WORK DONE UNDER A COOPERATIVE
AGREEMENT BETWEEN THE BUREAU OF MINES, DEPARTMENT OF THE INTERIOR
AND THE STATE OF COLORADO**



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FOREWORD TO FIRST EDITION.

The State of Colorado, among its resources of potential importance, considers its vast oil-shale deposits of particular value and the basis of a future great industry that ultimately will place Colorado in the front rank of the oil-producing States of the Union. It affords me much gratification that the United States Bureau of Mines has so generously cooperated with the State of Colorado in working out the solution of some fundamental problems of the coming industry and in presenting the results of its studies to the public.

OLIVER H. SHOUP,
Governor.

PREFACE.

The oil-shale investigations of the Bureau of Mines, begun in 1916, were interrupted by the entrance of the United States into the World War. Early in 1919, however, the work was taken up again in cooperation with the department of metallurgical research of the University of Utah, Salt Lake City, Utah. This cooperative work has continued to date.

In January, 1920, the bureau entered into a cooperative agreement with the State of Colorado for the conduct of laboratory investigations of the oil shales of Colorado. Under this agreement a laboratory has been installed and equipped at the University of Colorado, Boulder, Colo., and a research staff organized.

In 1921 an informal arrangement was made with the Department of Conservation of the State of Indiana, under which the bureau has acted in an advisory capacity in an investigation of the oil resources of the State of Indiana. This investigation is being conducted at the University of Indiana, Bloomington, Ind.

It is the primary purpose of the investigational work of the Bureau of Mines to determine the most favorable conditions for retorting oil shale to obtain the largest yield of the best products. Numerous papers dealing with the program of the investigations and with the work accomplished to date have been published in pamphlet and graphed form by the Bureau of Mines and in bulletin form by the State of Colorado. It is the purpose of the bureau and its cooperating agencies to continue publishing short reports as frequently as material becomes available.

This bulletin brings together pertinent facts regarding oil shale which hitherto have not been assembled in one volume, and presents them the results of the research to date. It is published to present in well-rounded form the present state of knowledge of the subject.

I take pleasure in acknowledging not only the cooperation of the States of Utah, Colorado, and Indiana, as mentioned above, but also the publication of the first edition of this bulletin by the State of Colorado, this action permitting issue at a much earlier date than would otherwise have been possible.

H. FOSTER BAIN.
Director

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OIL SHALE—AN HISTORICAL, TECHNICAL, AND ECONOMIC STUDY.

By MARTIN J. GAVIN.

INTRODUCTION.

The results of investigations of the oil-shale resources of the United States were first published by the United States Geological Survey in 1915.¹ Other reports² have followed.

These reports, investigations by private individuals and corporations, and the beginning of the development of an oil-shale industry in this country have aroused considerable interest in the possibilities of these shales as a commercial source of hydrocarbon oils. The Bureau of Mines has appreciated these possibilities, and since 1916 one or more of its engineers have given all or part of their time to investigations relating to the treatment of oil shales. A short pamphlet³ containing general information on the possibilities of oil shale in this country, particularly the shales of the Rocky Mountain district, has already been published. This pamphlet gives methods for assaying the shales of oil and ammonia and contains a bibliography for the use of those desiring to search the literature.

Further investigations of oil shales and the oil-shale industry, both in this country and in Scotland, where shales are being treated commercially, have made possible the presenting of the subject in greater detail. Since most of the published information on oil shale is scat-

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tered through a great many technical journals and treatises on petroleum and mineral oils, this bulletin summarizes pertinent facts of the literature, and in addition contains information that has been obtained by the Bureau of Mines in its investigations.

The appendix presents conclusions based on laboratory experiments of the Bureau of Mines on the retorting of oil shales. Details of the experimental methods and data are to be presented in a separate publication, soon to be issued.

Since the first edition of this bulletin was printed the material contained in it has been revised and brought up to date. The writer's attention has been called to certain errors in statements appearing in the first edition, and these have been corrected.

Throughout the bulletin, unless stated otherwise, a ton is 2,000 pounds and a gallon 231 cubic inches (U. S. gallon). Likewise, a barrel is 42 gallons of 231 cubic inches each.

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To those who by constructive criticism of the first edition of this bulletin have assisted in its revision the author is particularly grateful.

PRESENT STATUS OF THE PETROLEUM INDUSTRY IN THE UNITED STATES.

The petroleum in the United States is a natural resource that is rapidly being consumed. The fact that the supply is not inexhaustible and the gravity of the situation that will result from the supply becoming so depleted as to be inadequate to meet demands have been generally appreciated only within the past few years. As a result, these years have seen widespread efforts to conserve petroleum and its products and to seek possible substitutes for it.

The past history of the petroleum industry in this country reveals an astounding disregard for the conservation and efficient utilization of a valuable and irreplaceable fuel resource. Millions of barrels of petroleum and billions of cubic feet of natural gas have been wasted. Many productive oil fields have not been cared for properly and have thus been permanently damaged. Improper drilling and production methods have caused wastes that are beyond accurate estimate in dollars and cents. There has been much waste in refining, and the utilization of petroleum and its products, has been notably inefficient. With the growth of the industry, methods of producing, refining, and utilizing have improved, but the fact remains that no real and widespread effort to conserve petroleum and its products has been made until comparatively recently. Even now the discovery of a new oil field is only too often followed by overdrilling in order to gain the supposed benefits of "flush" production, and wells of large production are "brought in" before storage and transportation facilities are available. A large part of the oil produced in this country is burned as fuel for steam-raising and like purposes. Such utilization is inefficient and may be termed wasteful, for oil fuel should be burned in internal-combustion engines; steam boilers should not be fired with petroleum which represents only 3 per cent of the total fuel supply in this country, but they should be fired with coal which represents 97 per cent of the total fuel supply.

TABLE 1.—*Production and consumption of crude petroleum in the United States during the years 1914–1922, inclusive.**

| Year. | Domestic production, millions of barrels. ^b | Estimated domestic consumption, millions of barrels. ^c | Imports, millions of barrels. ^d | Proportion of consumption produced in United States, per cent. | Excess of consumption over domestic production, millions of barrels. | Yearly increase of imports, per cent. |
|-------------------------|--|---|--|--|--|---------------------------------------|
| 1914..... | 266 | 261 | 17 | 102 | -5 | |
| 1915..... | 281 | 273 | 18 | 103 | -8 | 5 |
| 1916..... | 301 | 319 | 21 | 94 | 18 | 13 |
| 1917..... | 335 | 378 | 36 | 89 | 43 | 47 |
| 1918..... | 356 | 412 | 56 | 86 | 56 | 25 |
| 1919..... | 378 | 420 | 53 | 90 | 42 | 40 |
| 1920..... | 443 | 531 | 108 | 83 | 88 | 101 |
| 1921..... | 472 | 526 | 125 | 90 | 64 | 18 |
| 1922 ^e | 551 | 586 | 124 | 94 | 35 | -1 |

* In rounded millions of barrels. Data from statistics issued by the United States Geological Survey.

^b Marketed production.

^c Sum of domestic production and net imports, plus decrease of stocks or minus increase of stocks.

^d Chiefly from Mexico.

^e Preliminary figures.

Both production and consumption of crude petroleum have increased greatly in the United States during the past 10 years, as is shown by Table 1 and Figure 1 which give the domestic production of crude petroleum, the imports, and the domestic consumption for the years 1914 to 1922, inclusive. It is to be noted that these data apply to crude petroleum only. The table shows that the United States has consumed more crude petroleum than it produced during the past 7 years, but when imports and exports of refined products are considered, ultimate domestic consumption of crude and refined oils has been less than total domestic production of crude. Actual indicated ultimate consumption of crude and refined products within the territorial borders of the United States for the past 3 years has been as follows:⁴

| | Indicated ultimate domestic consumption, barrels. |
|-----------|---|
| 1920..... | 419,693,294 |
| 1921..... | 419,012,943 |
| 1922..... | 481,743,190 |

The difference between the estimated domestic consumption of crude petroleum, as shown in Table 1, and the indicated ultimate domestic consumption, given above, is made up of exports of products to foreign countries and territorial possessions of the United States and fuel or bunker oil taken by vessels engaged in foreign trade.

The progress of the petroleum industry is marked by periods of overproduction or decreased demand and low prices, and underproduction or increased demand and high prices. From time to time new, highly productive oil fields are discovered, or there may be a

⁴ Data from Bull. Amer. Petrol. Inst., vol. 4, Apr. 16, 1923.

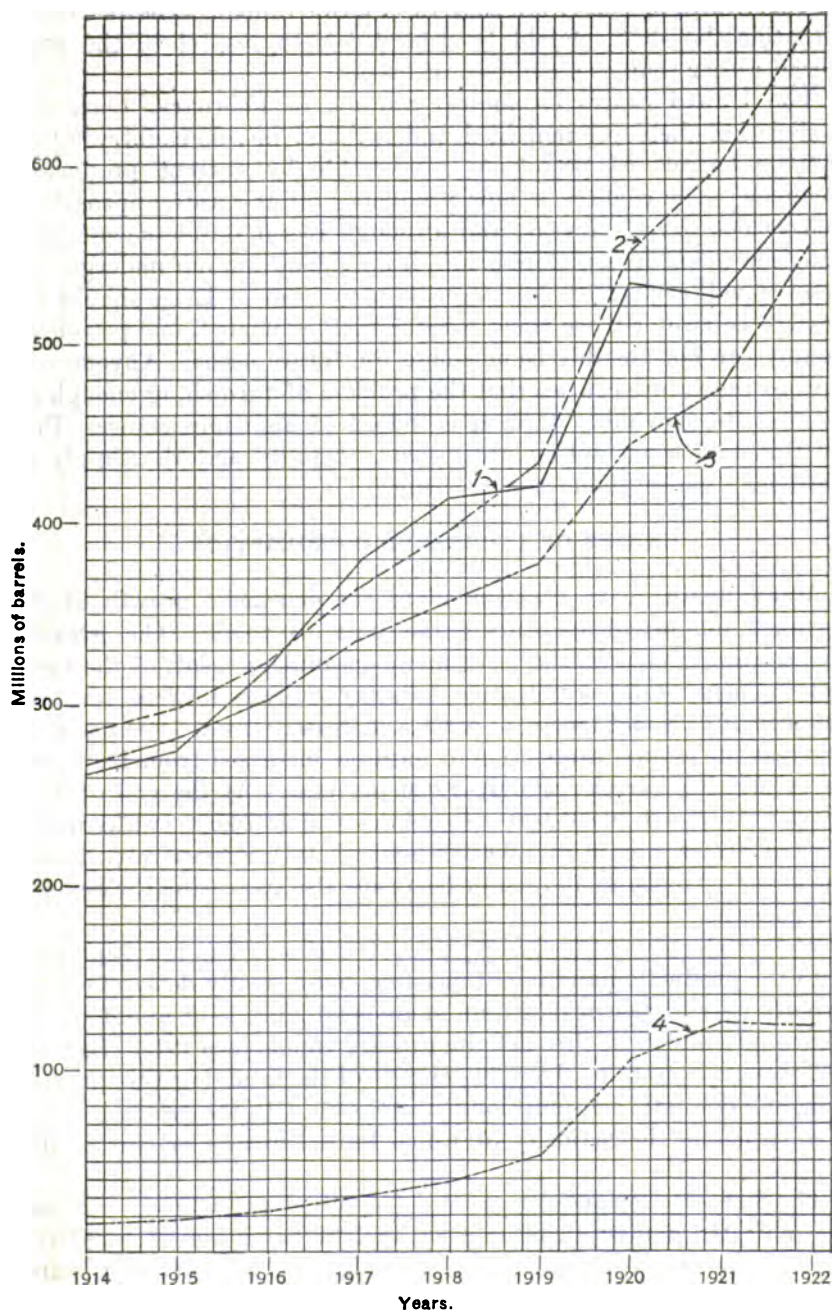


FIGURE 1.—Curves showing production, consumption, and imports of petroleum for the United States, 1914–1922. (1) Estimated domestic consumption; (2) estimated domestic production plus imports; (3) estimated domestic production; (4) imports.

temporary surplus of imports, and it may then seem that discussion of substitutes or substitute sources is unwarranted. The petroleum situation, however, should be regarded with respect to the years ahead rather than as a question of to-day.

The United States Geological Survey, in February, 1922, estimated that there yet remained underground in this country 9,150,000,000 barrels of petroleum recoverable by current production methods. The country is now producing oil at a rate approaching 800,000,000 barrels a year, and although recent developments indicate that the above estimate is low, it is evident that our underground reserve will not suffice for a great many years. Even should the reserve be two or three times greater than estimated, our petroleum supply can not possibly be adequate for future needs. Anyone who visions the future can see that the problem of furnishing enough oil to meet the inevitable demand is indeed of high importance. Production and consumption are growing rapidly and the supply of petroleum underground is decreasing.

CAUSES OF INCREASED CONSUMPTION.

Many factors have contributed to the enormous growth of the demand for petroleum during the past 10 years. The greatest increase in demand has been for motor gasoline, a result of the rapid development of the automotive industry. From 1909 to 1922 the number of cars and trucks in service increased about 3,900 per cent, whereas the annual production of gasoline increased from 13,000,000 to 147,672,000 barrels, or only 1,036 per cent. At the end of 1922 there were about 10,865,000 automobiles and 1,376,000 trucks in the United States, a gain of 1,520,000 and 257,000, respectively; during the year.* Airplanes and motor boats are also increasing the demand for gasoline.

The production and consumption of kerosene has not kept pace with the production and use of gasoline, but a steady demand exists; exports were higher in 1922 than in any year since 1919, and domestic consumption took its first definite upward trend since 1918. Exports of kerosene amount to about 39 per cent of the total domestic production, and domestic consumption is increasing with the growing use of farm tractors, the majority of which burn kerosene or similar distillates.

The domestic demand for gas and fuel oils shows a steady increase, the indicated domestic consumption in 1922 being almost 42,000,000 barrels more than in 1921. The use of gas oil for enriching water gas is growing rapidly and this demand is expected to continue, if

* Figures obtained from Automotive Division, Bureau of Foreign and Domestic Commerce, Department of Commerce.

not greatly increase. Gas oil is also extensively used as raw material for the production of gasoline by cracking processes. Distillate fuel is used in certain metallurgical processes and this use will probably become larger. There is also a growing demand for this fuel for heating dwellings. The demand for residual fuel oil increases year by year. Many coal consumers, including railroads and power plants, have turned to oil as fuel because of its convenience. This tendency is particularly evident among ship operators. Our Navy uses large quantities of fuel oil, and commercial steamers, particularly passenger liners, are increasingly burning clean, cheaply and conveniently handled liquid fuel. The use of fuel oil is not confined to shipping by any means; it is burned extensively in locomotives, in power plants, and in industrial heating plants.

The production, domestic consumption, and exports of lubricating oil, which fell off after reaching a high peak in 1920, are again increasing. The growth of industrial activity and the larger use of automobiles, tractors, and trucks is increasing the demand for lubricants. The export demand is certain to be much greater as internal affairs in Europe become stabilized and industrial expansion begins.

It is clearly evident that the demand for petroleum products is steadily increasing, and even though domestic production may be adequate for the time being, our reserves are limited and sooner or later will fail. Before the time of critical shortage is reached new sources of petroleum products, or substitutes for them, must be found.

POSSIBLE NEW SUPPLIES OF PETROLEUM.

IMPORTS.

The chief possibility of new supplies of natural petroleum seems to lie in larger imports, chiefly from Mexico, the countries bordering the Caribbean Sea, and South America. Mexico has produced large amounts of oil, but production of the grades most suitable for refining decreased in 1922, and there is some doubt as to the future production. Much of the Mexican crude is heavy and asphaltic, and is best suited for making fuel oil, a little gasoline, some kerosene, and gas oil containing so much sulphur as to be unsuitable for the enrichment of water gas.

Other countries bordering the Gulf of Mexico and the Caribbean Sea contain oil deposits, but these are comparatively undeveloped and little is known as to the amount and the quality of the oil they will yield.

The United States, however, would be at a great disadvantage if wholly or even partly dependent on imports of a commodity so vital as petroleum. Mineral oils are essential to the Nation's safety, as

was demonstrated during the World War, and to its commercial and industrial life, and every effort must be made to insure domestic sources of supply.

INCREASED PRODUCTION.

As noted on page 6, the United States has an estimated underground supply of 9,150,000,000 barrels of petroleum recoverable by present methods of production. Lewis* has estimated that ordinary production methods often do not recover more than 10 to 20 per cent of the oil underground. This waste suggests that new methods may be developed for lengthening the life and increasing the production of known oil fields. There is also the possibility that deeper drilling, at present restricted by high costs and inadequate equipment, may add to the supply by finding new productive sands in present fields. Moreover, it is probable that new fields of importance will be developed; but these expedients can hardly be expected to do more than postpone for a comparatively short period the time of critical deficiency. In brief, the demand for mineral oils in the United States in the near future will evidently be supplied partly from domestic production (probably soon to decrease), partly from imports, and partly from substitutes. More efficient methods of production and utilization will undoubtedly help to prolong the life of our oil supplies.

INCREASED PRODUCTION AND MORE EFFICIENT UTILIZATION OF GASOLINE AND FUEL OIL.

Much of the gasoline marketed in this country is "straight run"; that is, distilled from the crude oil without being cracked and without being blended with natural-gas gasoline. However, increasing quantities of cracked gasoline and gasoline recovered from natural gas are being satisfactorily blended with straight-run gasoline. Gasoline produced by cracking, or breaking down the molecules of heavier distillates, when properly produced and refined, is a satisfactory motor fuel. Natural-gas or "natural" gasoline is obtained by recovering the gasoline vapors in natural gas by compression or absorption. As it is usually too volatile or "wild" to use alone, it is ordinarily mixed or blended with heavier petroleum distillates before being marketed as motor fuel.

It does not seem likely that the production of gasoline can be materially increased by using more of the crude petroleum and making the product less volatile, as has been done in the past, until automotive engines that will satisfactorily burn less volatile fuel are

* Lewis, J. O., Methods of increasing the recovery from oil sands: Bureau of Mines Bull. 148, 1917, pp. 25-28.

designed. Also the demand for the next fraction below gasoline—kerosene—is steady and probably will increase. To include any considerable quantity of the lighter fractions of kerosene in the gasoline will cause a shortage of kerosene and a consequent increase in price. The extraction of gasoline from natural gas tends to increase with the production of petroleum and will decrease as petroleum production fails. The changing of fuel oil into gasoline by cracking processes involves considerable expense and some waste. Therefore unless there is a decided difference between the price of fuel oil and of gasoline cracking is not profitable. The demand for fuel oil is increasing and it may not always be profitable to crack more fuel oil into gasoline unless the price of gasoline goes higher. However, the increasing efficiency of cracking processes and the fact that the price of fuel oil is limited by the price of its chief competitor—coal—indicates that the production of gasoline by cracking will continue to increase.

More efficient utilization of gasoline is promised by recent developments of Midgley.⁷ The efficiency of the modern gasoline engine can be increased by increasing the pressure attained by the gaseous explosion mixture, but when this is done with the ordinary petroleum products a so-called “detonation” or “knocking” results, which is destructive to the engine and which more than offsets the theoretically increased efficiency. Midgley has discovered that the addition to the gasoline of small amounts of organic compounds of certain metals suppresses this tendency to detonation. The full effect of this discovery will come only with the general manufacture and use of engines designed for higher compression pressures.

It is altogether probable also that the future will see the development of automobile and truck engines that will burn fuel oil efficiently. Such a development will not only relieve the danger of an impending fuel shortage but will make possible even greater extension of the use of the automobile and truck. An automotive engine burning fuel oil can compete against any other user of fuel oil, because this would be the most efficient use to which the oil could be put. Consequently the supply of motor fuel would be limited only by the total supply of fuel oil. In the same manner the more extensive use of the Diesel-type engine to replace engines driven by steam from oil-fired boilers will make possible a much more efficient utilization of our petroleum.

Any consideration of the increased use of the heavy-oil engine must take into account the fact that in future much of the oil now used as fuel will be needed for the manufacture of lubricants. As

⁷ Midgley, Thomas, jr., and Boyd, T. A., Chemical control of gaseous detonation with particular reference to the internal-combustion engine: *Jour. Ind. and Eng. Chem.*, vol. 14, 1922, pp. 894-898.

yet the demand for lubricants is not sufficient to consume all the lubricants that might be made from the crude oils of the country, but the demand is certain to increase. Lubricants are the most necessary of all the products of petroleum. The industrial development of to-day would be impossible without lubricating oils, and these can be obtained in sufficient quantity, so far as we know, only from petroleum.

SUBSTITUTES FOR PETROLEUM.

ELECTRIC POWER.

On the sea the only source of power that can be substituted for oil is coal; on land, however, electricity can and undoubtedly will replace fuel oil in many places. Some railroads—the Chicago, Milwaukee & St. Paul, for example, with nearly 800 miles of main line electrified—have electrified parts of their main lines; others, including one of the largest users of fuel oil in the country, are said to be planning extensive electrification. Various industries will undoubtedly make increasing use of electricity as the water-power resources of the country are developed. The proposed super-power system for the North Atlantic States includes both hydroelectric plants and huge steam-driven generating stations, some of which will be close to coal mines.

BY-PRODUCTS FROM COAL.

When bituminous coal, cannel coal, lignite, and peat are destructively distilled, tars of various kinds are produced, from which different products can be obtained. At present, gasoline blended with commercial benzol (recovered at by-product coke ovens) and blends of gasoline, alcohol, and benzol are being marketed as satisfactory motor fuels. The heavier coal tar is being used extensively as fuel oil in some parts of the eastern United States. As by-product coke ovens replace the wasteful beehive ovens, more and more products from coal can be expected as substitutes for those obtained from petroleum, but coal by-products can be only of minor aid in supplying these substitutes. It has been estimated that if all the coal mined in the country for a year were coked in by-product ovens, the quantity of motor fuel that might be made available thereby would amount to less than 20 per cent of the annual domestic consumption of gasoline. The development of low-temperature processes for carbonizing coal may, however, greatly increase this potential yield.

ALCOHOL.

Unblended alcohol can be used as fuel in engines designed for compression higher than those ordinarily used with gasoline. Com-

mercial alcohol can not be blended directly with gasoline, but mixtures of gasoline, alcohol, and benzol that will not separate on standing are being used in a limited way at present, and this use of alcohol will probably grow if the price of motor gasoline increases or the cost of making alcohol decreases. The calorific value of alcohol is lower than that of gasoline, but this is offset by its high efficiency. Alcohol can be produced from a large number of vegetable waste products, and probably can also be obtained in quantity from wood wastes, such as sawdust, although this has not yet been demonstrated on a large scale. At present the production of alcohol is limited by the availability of raw material suitable for its manufacture. It is to be noted that land on which plants for the production of alcohol are grown is no longer usable for the production of food.

SUBSTITUTE SOURCES OF LUBRICANTS.

It is difficult to see where satisfactory substitutes for petroleum lubricants can be found except in shale oil, a source of somewhat doubtful utility (p. 120). The production of lubricating oils from petroleum has in no way reached its limit (p. 9), but the time is coming when substitute sources will be really needed. Good lubricating oils, such as castor oil, which is an excellent lubricant, can be made from vegetable sources, but the cost at present is very high.

OIL SANDS.

Important potential sources of petroleum products in the United States and other countries are the large deposits of sand saturated or partly saturated with oil. These deposits are now yielding oil in the productive fields but ultimately, through partial exhaustion of oil and decline of gas pressure, they will not yield oil through wells. Also there are enormous deposits of oil sands that never have produced oil and, because they are not saturated with oil or because gas is absent, are not capable of producing oil through wells. Many of these deposits outcrop in such manner that their mining would be relatively inexpensive and sometime, it is to be expected, they will be mined and their oil extracted by suitable methods. Certain of these deposits, satisfactorily situated with respect to transportation, supplies, and markets, seem to offer a better opportunity for development in the near future than do our oil-shale reserves.

SHALE OIL.

As sources of substitutes for petroleum oils, the reserves of oil shale in the United States stand out as most important. Oil shales in Scotland, France, and Australia have been the source of products

similar to those obtained from petroleum, and this fact makes it highly probable that in the near future oil shales which underlie great areas of this country will be extensively worked.

The pages following treat of the history, technique, and economics of the oil-shale industry and emphasize the value of our deposits. The technical and business problems that must be solved before these deposits are developed on a large scale are not beyond our technical skill or financial ability. A broad view of the outlook justifies the statement that our oil-shale reserves insure a domestic supply of mineral-oil products similar to those obtained from petroleum for a long time to come.

FOREIGN DEPOSITS OF OIL SHALES.

SCOTLAND.

Oil shale occurs in many parts of the world, but the shale deposits of Scotland, because of their having been the basis of a commercial industry for many years, are the most widely known. These deposits lie mostly to the west of Edinburgh and south of the Firth of Forth.* The present operations of importance are all in one field in West and Mid-Lothian, the center of the field being about 12 miles west of Edinburgh. The field is well defined, stretching from the Firth of Forth, west of the Forth Bridge, southward for about 16 miles between the Almond River and the Bathgate hills to the region of Cobbinshaw and Tarbrax. The oil shales form part of the calciferous sandstone series at the base of the Carboniferous system.

The shales now being worked are black or brownish black, break with a conchoidal fracture, and give a dark-brown streak. They are tough, flexible, free from grittiness, and curl to the cut of a knife. The rich shale is very resistant to weathering, thus differing from the barren shales, called "blaes," because of their bluish gray color, by the miners. The specific gravity of the shales worked for oil ranges from 1.71 to 2.30.

The shale is finely laminated, a property best noted in the spent shale, although the laminæ in the raw shale can be seen distinctly under the microscope. Layers of rich and lean shale alternate, probably representing alternate periods of rich vegetable growth followed by water overflow and sedimentation.

Good oil shale resembles hard, dark wood, or even leather, and an experienced worker can determine its quality by the readiness with which it curls when cut with a sharp knife. This property and its

* Carruthers, R. G., *The geology of the oil-shale fields: Memoir Geol. Survey Scotland, The oil-shale fields of the Lothians*, 2d ed., 1912, Part I, pp. 1-95.

color, weight, and streak enable it to be distinguished from the "blaes," although the latter often passes or grades into an oil shale yielding enough oil and ammonia to be worked profitably.

Scottish shale operators distinguish plain and "curly" shale, the former being flat and smooth and the latter contorted and polished or slickensided on the squeezed faces; the same shale may be partly plain and partly curly; the curly shales often, but by no means always, yield more oil than the plain shales. However, curly shales always yield well, as only a reasonably rich shale curls under lateral pressure.

The shales being worked in 1921 yielded about 25 United States gallons of oil and 36 pounds of ammonium sulphate to the short ton. Formerly shales yielding much more oil were mined, but they were worked out years ago. The shale-oil industry in Scotland started on a mineral yielding very little ammonia, but up to 130 gallons of oil to the ton. Some geologists do not consider this material, known as torbanite, from its occurrence at Torbane Hill, to be a true oil shale. (See p. 35.) Since the exhaustion of the torbanite, the oil yield of the shales worked has grown smaller and smaller. As a rule the Scottish shales that yield the most oil yield comparatively little ammonia, and vice versa. Naturally the shales rich in oil were worked first, the ones now mined yield less oil but more ammonia. (See p. 92.)

In the Scottish deposits the oil shales are interstratified with ordinary carbonaceous shales (the "blaes" of the miners) which contain little or none of the characteristic material of the oil shales. These beds, however, in the main differ only in the amount of organic matter, and portions are retorted under favorable economic conditions. Beds of sandstone and limestone also occur in the shale series.

At different times some 20 different seams of oil shale have been mined, but at present only about 10 are being worked. These range in thickness from $3\frac{1}{2}$ feet to 8 or 10 feet, averaging about 5 feet. The seams are generally persistent but of varying thickness and grade into "blaes" laterally. Generally both roof and floor also consist of "blaes" but may be fire clay, sandstone, or limestone. The shale beds are folded, usually in small basins, and in many places are badly faulted. At Pentland, however, the seams extend for nearly two miles without serious faulting and dip uniformly. The seams are now well mapped and correlated.

ENGLAND.

Oil shale has been known for a long time in the Kimmeridge district of Dorset and has lately been found in and around Norfolk. Kimmeridge shale is known locally as Kimmeridge coal or black-

stone. It has been worked in a small way in Dorset from time to time for centuries. There are probably very large quantities of oil shale in the district, but thus far successful commercial development has been impossible, primarily because the crude oils from the shale contain 6 to 9 per cent of sulphur and, on account of a low proportion of paraffin oils, are extremely difficult to refine. Proposals to use the oil for fuel have amounted to little, as the present British Admiralty specification for fuel oil allows a maximum of not more than 3 per cent of sulphur, and efforts to reduce the sulphur content of Kimmeridge shale oil to that amount have not been successful. Within the last three years it has been reported that sulphur-free shales underlie those previously discovered in Norfolk, and a company has been formed to exploit them. During 1923, operations in Norfolk are reported to be at a standstill, and the company organized to exploit the shales of that district is said to be in financial difficulties.

The yield of oil from core-drill samples of Kimmeridge shale ranges from 15 to possibly over 42 gallons per ton and sulphate of ammonia up to 30 pounds per ton.⁹ The seams of shale that might be of economic importance are apparently rather thin.

WALES.

Oil shales of rather inferior quality were discovered in Wales a long time ago, and considerable quantities were treated many years since, but the competition of cheap foreign petroleum caused the abandonment of the industry.

CANADA.

Oil shales occur in New Brunswick, Nova Scotia, Quebec, and Newfoundland. There are other deposits in other parts of the Dominion, but they have not been more than casually examined. The shales of the first two Provinces mentioned are considered of importance, and efforts are now being made to develop them. A deposit of possible value has also been reported on the west coast of Newfoundland.

In 1907 a large quantity of shale from New Brunswick was sent to the Pumpherson Oil Co. works in Scotland for testing. The results indicated that 24 to 48 gallons of oil and from some samples over 70 pounds of ammonium sulphate could be obtained per ton of shale.¹⁰

⁹ Strahan, A., Special reports on the mineral resources of Great Britain, vol. 7; Lignite, jet, Kimmeridge oil shale, mineral oil, cannel coals, natural gas. Part I, England and Wales. Memoir Geol. Survey, London, 1918, pp. 29-30.

¹⁰ Ellis, R. W., Joint report on the bituminous or oil shales of New Brunswick and Nova Scotia, also on the oil-shale industry of Scotland: Canada Dept. Mines, Bull. 55, 1909, 57 pp.

In the Scottish retorts the throughput of the richer of these shales was much slower than that of Scottish shale.

AUSTRALIA.

Oil shales occur in New South Wales and Queensland. In addition a peculiar spore deposit known as tasmanite, which yields considerable oil, occurs in Tasmania. According to reports in 1921, a large company had obtained an option to these tasmanite deposits and was to start the construction of retorts at once. Official reports of the Tasmanian Government estimate that the possible quantity of shale available is 12,000,000 long tons. The oil produced from tasmanite has been considered unrefinable, but under some conditions it could be used crude as fuel oil.

Two grades of shale have been mined and retorted in New South Wales for many years: One termed "kerosene shale," a rich variety partly used locally and exported for gas making; and the other, "seconds," a lower grade retorted locally. The two grades are considered identical in nature and origin—differing only in richness—and are generally considered torbanites.

The following description of the Australian oil shales and their occurrence is abstracted from articles describing the industry of New South Wales in detail.¹¹

OCCURRENCE IN NEW SOUTH WALES.

Oil shale occurs in New South Wales mostly in lenticular patches, few more than a mile long, but some several miles wide, in the upper coal measures of Permo-Carboniferous age. The seams range in thickness from 1 or 2 inches to 4½ feet or more.

The most important earlier workings were at Hartley Vale, Joadja, and Torbane, in the Capertee Valley. At Hartley Vale the shale was rich and 3½ to 4 feet thick, with about 10 inches of lower-grade shale below that had to be removed with an ax. At Capertee there was about 5½ feet of good shale, with a few inches of splint above and below. The important works, in 1922, were mining the Capertee-Wolgan bed at Newnes, in the Wolgan Valley. This seam, the most extensive yet discovered in New South Wales, is 14 to 50 inches thick. On the Capertee side one tunnel is said to have exposed for a distance of over 4,000 feet a seam averaging 3 to 4 feet thick.

Other shale deposits of lesser importance occur in the Blue Mountains and in the Hunter River valley.

¹¹ Stewart, D. R., *The oil shales of the Lothians, Part III, The chemistry of the oil shales: Memoir Geol. Survey, Scotland, 2d ed., 1912, p. 163.*

Carne, J. E., *The kerosene shale deposits of New South Wales: Memoir Geol. Survey, New South Wales, 1903, 338 pp.*

Handbook of the mineral products of New South Wales: Dept. Mines, New South Wales, 1920, pp. 13-16.

NEW ZEALAND.

To work a so-called oil shale, which is probably a lignite of high sulphur content, a plant of the Scottish type was erected in 1900 at Orepuki. The work has since been discontinued on account of the low grade of the shale, the poor quality of the oil produced, and unexpectedly high operating costs. In 1901, 12,048 long tons of shale, valued at £6,024, were mined; in 1902 only 2,338 tons; and only a negligible quantity from that time to date. Besides the deposit at Orepuki other oil-shale deposits have been discovered in different parts of New Zealand, but that at Orepuki appears of most potential importance.

AFRICA.

Oil shale has been discovered in southern Rhodesia, the Transvaal, and Natal; but as far as is known no active development has been undertaken, although proposals have been made recently. Other deposits are reported in Morocco, Angola, and—perhaps the largest of all—in the Belgian Congo. No reports giving accurate data as to the size of the deposits are available. As a rule the shale seams are relatively thin and the oil yield low. The yields from the Transvaal shales are about 30 gallons of crude oil and 30 pounds of ammonium sulphate per long ton.

FRANCE.

The most important oil-shale deposits are at Buxière and Saint Hilaire (Allier), which have been worked since 1858, and at Autun (Saône-et-Loire), where operations started in 1862. It is interesting to note here that the first oil shales ever worked were those of Autun, products made from them by Selligie having been exhibited at the Paris Exposition in 1839. Selligie can really be considered the father of the oil-shale industry, although operations in France were soon outstripped by those in Scotland. According to Redwood¹² the oil-shale industry of France dates from the year 1830.

Abraham states that the shales at Autun and Buxière are not true oil shales, but consist of semiliquid asphalt associated with shales.¹³

Besides the two fields mentioned, shales have been worked in the Departments of the Var, Basses Alpes, and Puy du Dôme. Little work has been done in these fields, but recent reports (1921) indicate that development of the shales of the Boson field in the Var is about to begin on a large scale.

In the Buxière region are three oil-shale works that obtain yields of oil varying from 13 to 26 gallons per ton. The specific gravity of

¹² Redwood, Boverton, *Treatise on petroleum*. Vol. 2, 3d ed., London, 1913, p. 87.

¹³ Abraham, Herbert, *Asphalts and allied substances*. 1920, p. 158.

the crude oil ranges from 0.850 to 0.900 and yields refined products as follows:¹⁴

Products obtained from crude shale oil at Buzière.

| | Specific gravity. | Per cent. |
|------------------------|-------------------|-----------|
| Lamp oils ----- | 0. 810-0. 820 | 28-30 |
| Heavy oils ----- | . 870-0. 925 | 30-40 |
| Pitch ----- | | 29 |
| Loss in refining ----- | | 12-20 |

The heavy oils are said to make rather good lubricants, and from them is obtained a good grade of paraffin wax. The pitch finds a ready market for use in the manufacture of asphalt preparations. The average normal annual production of shale in the Allier district is about 67,000 tons.

The Autun field, the most important of the French shale districts, has an area of about 250 square kilometers. Several beds of oil-yielding shale are interstratified with coal, gravel, clay, limestone, and conglomerates. The deepest bed mined yields about 12 gallons of oil per ton. A middle seam, called the "big seam" and the most important of those worked, yields 12 to 25 gallons per ton. Upper seams have been worked to a small extent. About 110,000 tons of shale are mined annually in the Autun field.

When refined the oils from Autun give products approximately as follows:¹⁵

Products obtained from crude shale oil at Autun.

| | Specific gravity. | Per cent. |
|------------------------|-------------------|-----------|
| Lamp oil ----- | 0. 820 | 35-40 |
| Heavy oils ----- | 0. 860-0. 868 | 4 |
| Green oil ----- | . 895 | 25 |
| Pitch ----- | . 980 | 20 |
| Loss in refining ----- | | 14 |

According to Redwood¹⁶ the shale-oil industry in France is not remunerative, chiefly because of the poor quality of the shale. He also states that the oils are very difficult to refine. However, the present heavy import duty on oil enables several deposits to be worked successfully.

YUGOSLAVIA.

Large deposits of oil shale have been known to exist near Alexinatz for many years.¹⁷ Some of the shale was tested years ago in one of the Scottish works, and is said to have yielded over 45 gallons of oil

¹⁴ Petroleum World, Oil Industry in France and Alsace: Vol. 16, September, 1919, p. 372.

¹⁵ Petroleum World, reference cited.

¹⁶ Redwood, Boverton, Treatise on petroleum. Vol. 2, 3d ed., London, 1913, p. 90.

¹⁷ Redwood, Boverton, work cited. Vol. 2, p. 91.

to the ton. The main bed is said to be more than 90 feet thick; above and below it are beds of lower grade.

Recently a prospecting right covering an area of about 120 square kilometers has been granted to a newly discovered area of oil shale¹⁸ at Radovagne, near Belgrade. The deposit contains two kinds of shale, a rich brownish-yellow shale and a black bituminous shale. The former covers an area of 30 square kilometers and is 1 meter thick at the outcrop. In laboratory tests the shale yields about 15.5 per cent of oil by weight.

SPAIN.

Extensive deposits of oil-yielding shale are known in the Pedro Martinez Basin, Province of Granada; at Rubielos de Mora (Teruel); and at Ribes Albes (Castellon) and other places. There are no accurate data on the supply of shale available. A deposit near Puertollano (Ciudad Real) of Carboniferous age is now being mined and the shales retorted in a plant of the Scottish type.¹⁹ It is said that the crude-oil production amounts to 3,500 gallons a day. The shale is mined at a depth of 300 feet.

SWEDEN.

Large oil-shale deposits occur in Sweden, the principal ones being at Kinnekulle, Närke, and Östergötland.²⁰ The shales are known locally as "alum" shales. Such information as has been obtained indicates that the oil yield is relatively poor. One company is said to have spent over \$800,000 in an effort to create a profitable industry out of the production of shale oil, with discouraging results. Late in 1921 the plant was offered to the Swedish Government for \$134,000. The Government has appropriated funds for an investigation of extraction methods, but, as far as is known, has not taken over the plant.

BULGARIA.

There are said to be large deposits of oil shales near Breznik, Radomir, Popovtzi, Kazanlik, and Serbinovo.²¹ In the deposit near Breznik alone the surface shale is estimated at about 30,000,000 tons. The beds seem to be thick enough to be considered suitable for commercial working. Their yield of oil, as tested by various chemists, ranges from 7 to 21 per cent, 13 per cent being about an average.

¹⁸ From unpublished report of H. I. Smith, U. S. Bureau of Mines.

¹⁹ Ritter, Etienne A., Distillation of oil shales at Puertollano, Spain: Eng. and Min. Jour.-Press, vol. 115, Feb. 17, 1923, pp. 326-327.

²⁰ Murphy, D. I., Manufacture of mineral oils from alum shales: Commerce Reports, Bureau of Foreign and Domestic Commerce, Oct. 2, 1919, p. 27.

²¹ Kemper, G. H., Deposits of oil shale in Bulgaria: Commerce Reports, Bureau of Foreign and Domestic Commerce, Apr. 17, 1920, pp. 540-541; also, Commerce Reports, May 19, 1921, pp. 1020-1021.

The oil is said to yield less gasoline and kerosene, but more lubricating oils and wax, than the Scottish oil. Also, the nitrogen content of the Bulgarian shales is less than that of the Scottish shales. Three concessions to work these shales are said to have been granted.

GERMANY.

Oil shales occur in the Rhine Provinces and in the States of Hesse, Wurtemberg, Baden, and Bavaria. The Bavarian deposits, which yield about 16 gallons of oil per ton, are to be worked in the near future, according to recent reports. During the war the deposits at Messel, Hesse, were worked to some extent.

In Saxony an important industry has been established around the distillation of brown coal. The technique of the industry is much like that of the oil-shale industry, but the products, except paraffin wax, are dissimilar.²²

ITALY.

A concession to work a bituminous shale deposit in the Province of Messina, Sicily, was granted in 1915. The shale is said to be readily accessible and to yield 7 to 9 per cent of a rather heavy oil containing 3.4 per cent sulphur. Several other deposits of low-grade shales have been discovered recently.

SWITZERLAND.

In 1915 a concession of about 8,000 acres was granted to work deposits of oil sands and oil-yielding shales in the Canton of Geneva. The shale is said to yield about 15 gallons of oil to the ton and the oil seems to be of good quality. The commercial possibilities of this deposit are not yet known.

ESTHONIA.

Much interest has been manifested recently in the oil-shale deposits of Esthonia, which extend from the west coast near Port Baltic to the north shore of Peipus Lake. The shales occur in several thin beds, interbedded with limestone, attain a total maximum thickness of 8 feet, and are covered with only a few feet of overburden; so mining will be inexpensive. The deposit now being developed in the northeastern part of the country, near the towns of Iewe and Wesenberg, is said to cover an area ²³ of 800 square kilometers, and is believed to contain over 1,500,000,000 tons of shale. The average oil yield is stated to be well over 50 gallons per ton.

²² Scheithauer, W., *Shale oils and tars and their products*, London, 1913, 183 pp.

²³ Petroleum Times, *The chemical properties of the Esthonian shales*: June 3, 1922, pp. 777-778.

Three mines are now working this deposit, which in 1921 produced 85,000 long tons of the shale. Distillation of the shale is still in the experimental stage, but large quantities have been used as fuel in cement mills and private dwellings and for gas manufacture. Attempts to use it as boiler fuel have apparently been unsuccessful. The Esthonian Government in its efforts to prevent the formation of a monopoly is not disposed to make terms attractive to the many companies which have requested concessions, but in 1922 a concession of a large area was granted to an English company. Other concessions have been granted to Esthonian and Belgian interests.

BRAZIL.²²

Oil shales occur in Alagôas, Bahia, Maranhão, São Paulo, and in south Brazil; the tertiary shales of São Paulo are by far the most important.

Alagôas.—Tertiary oil shales are exposed at low tide along the beach near Maceio, but seem to be of local interest only. Development in a very crude way was unsuccessfully attempted several years ago.

Bahia.—The "turfa" at Marahú is of recent formation, local in extent, and resembles peat rather than oil shale. It yields on testing more than a barrel of oil per ton. Development attempted 30 years ago met failure and the plant then built is now in ruins.

Maranhão.—Thin, unimportant beds of Permian oil shale are known at several places.

São Paulo.—Thick beds of soft, easily cut shale yielding 25 to 30 gallons of oil per ton are within a few feet of the surface over a large area along the Rio Parahyba. A 20-ton retorting plant recently installed at Taubaté was idle during 1921. Considerable shale mined during the war was used for gas production in the cities of Rio de Janeiro, São Paulo, and Santos.

South Brazil.—Low-grade Permian shale may be of considerable importance because of its great thickness (100 feet in places) and probable great extent. A retorting plant is said to have been erected recently at São Gabriel.

OTHER COUNTRIES.

Oil-shale deposits of possible commercial importance are known in Argentina, Chile, and Uruguay. Because of lack of fuel resources in many parts of South America, some of these deposits may present excellent opportunities for successful development.

Discoveries of large deposits of oil shale of good quality at Fushun, Manchuria, have been reported recently and plans are now being

²² The notes on the shale deposits of Brazil were prepared for this bulletin by Mr. D. E. Winchester, consulting geologist, Denver, Colo.

made to develop them. Oil shale has also been discovered in Mongolia. Deposits are likewise reported in interior Arabia and in the Machada Plain district of Syria. Little is known as to the exact location of the deposits or the amount and quality of the shale. Oil shale and bitumen deposits on the shores of the Dead Sea are said to have been worked by the Germans during the war.

OIL SHALES IN THE UNITED STATES.

Oil shales occur in many parts of the United States. Those that are now receiving particular attention and are, perhaps, of most economic importance occur in the States of Colorado, Nevada, Utah, and Wyoming. The richest of these are in the Rocky Mountain region and belong to the Green River formation of Eocene age. The shales of Nevada possibly are slightly younger than Green River, and most of those of Montana are certainly older (Upper Paleozoic). The black shales of the eastern States—Illinois, Missouri, Indiana, New York, Kentucky, Ohio, Pennsylvania, and Tennessee—occur mainly at one general horizon in the Upper Devonian, although extensive deposits occur in the Lower Devonian and Ordovician; those of California, which, strictly speaking, are not true oil shales, possibly with minor exceptions, are of Miocene age.

OIL SHALES OF THE ROCKY MOUNTAIN REGION.

The oil shales of the Green River formation occur in Garfield, Mesa, and Rio Blanco Counties, northwestern Colorado; in Uinta, Sweetwater, and Lincoln Counties, southwestern Wyoming; and in Uinta, Duchesne, Carbon, and Wasatch Counties, northeastern Utah. Of these areas, the largest now known is in Utah.

The Green River formation consists mostly, but not wholly, of shale. (See frontispiece, Pl. I.) Some of the shale does not yield oil, and much of it is too low grade or is not in thick enough beds to be of present economic importance. According to Winchester²⁵ the Green River formation contains persistent beds up to 49 feet thick that will yield at least 35 gallons of crude oil to the ton. More thorough surveying and sampling may discover thicker seams of equal or greater richness. A recently discovered 20-foot seam, sampled across the face at one point, yielded over 60 gallons of oil to the ton, by laboratory test.

In Colorado the Green River formation attains a maximum thickness of about 2,600 feet. The middle part of the formation contains most of the oil shale, the upper and lower parts being practically barren. The oil shales lie practically horizontal, except for occasional local displacements and foldings, and appear to be relatively

²⁵ Winchester, D. E., personal communication, 1922.

uniform in thickness and yield. At least two definite workable beds, yielding over 25 gallons of oil to the ton, have been found practically wherever the formation has been carefully examined. In some places, above these two, there is a third bed of greater richness and usually of workable thickness. The rich oil-shale strata are from a few inches to over 30 feet thick, and are interstratified with lean shales, barren shales, sandstone, limestones, conglomerates, and thin-bedded oolites.²⁰

The oil-yielding shales usually outcrop high on the sides of steep-walled canyons (Pls. I and III, *A*) and are darker colored and more resistant to weathering than the barren beds. Their altitude above sea level ranges from 6,000 to 8,000 feet. The regions in which the shales lie are mostly arid or semiarid, sparsely populated, and in general not well supplied with transportation facilities.

Some beds, most of them too thin to be worked commercially, have yielded up to 90 gallons of oil to the ton in laboratory tests, and it is safe to assume that the Green River formation contains enough shale of workable thickness to supply a large amount of material that for many years will yield over 35 gallons to the ton. These shales will also yield upward of 20 pounds of ammonium sulphate to the ton.

The oil shales of the Green River formation occur in two principal forms, paper and massive (Pls. II, *A*, *B*, and III, *B*), with many gradations. It is believed by many that the paper shale is merely a weathered form of the massive shale. Some workings seem to indicate this, as one tunnel driven in paper shale for more than 20 feet from the surface found that this shale gradually became massive farther in. The massive shale has plain or curly form; the latter is more or less contorted and folded and rarely shows slickensided surfaces. The paper shale is obviously foliaceous, but in much of the massive shale the foliated structure can be seen only in the spent shale or in thin sections under the microscope.

The massive shale, which is black or dark brown, is extremely tough and withstands weathering remarkably; it weathers to a bluish-white color on the surface, but weathering probably does not affect the oil yield of the shale to a depth of more than a few inches. The massive shale gives a brown or brownish-yellow streak and has a conchoidal fracture. Fresh surfaces of many samples of the shale have a strong odor of petroleum, which, however, quickly disappears. If the shale is fairly rich, it can be ignited and burned.

The paper shale is gray to brown or even black in color, and its thin laminæ are remarkably flexible, even when badly weathered,

²⁰ Winchester, D. E., Oil shale in northwestern Colorado and adjacent areas: U. S. Geol. Survey Bull. 641, 1917, pp. 189-198.



4. OIL SHALE NEAR WATSON, UTAH.



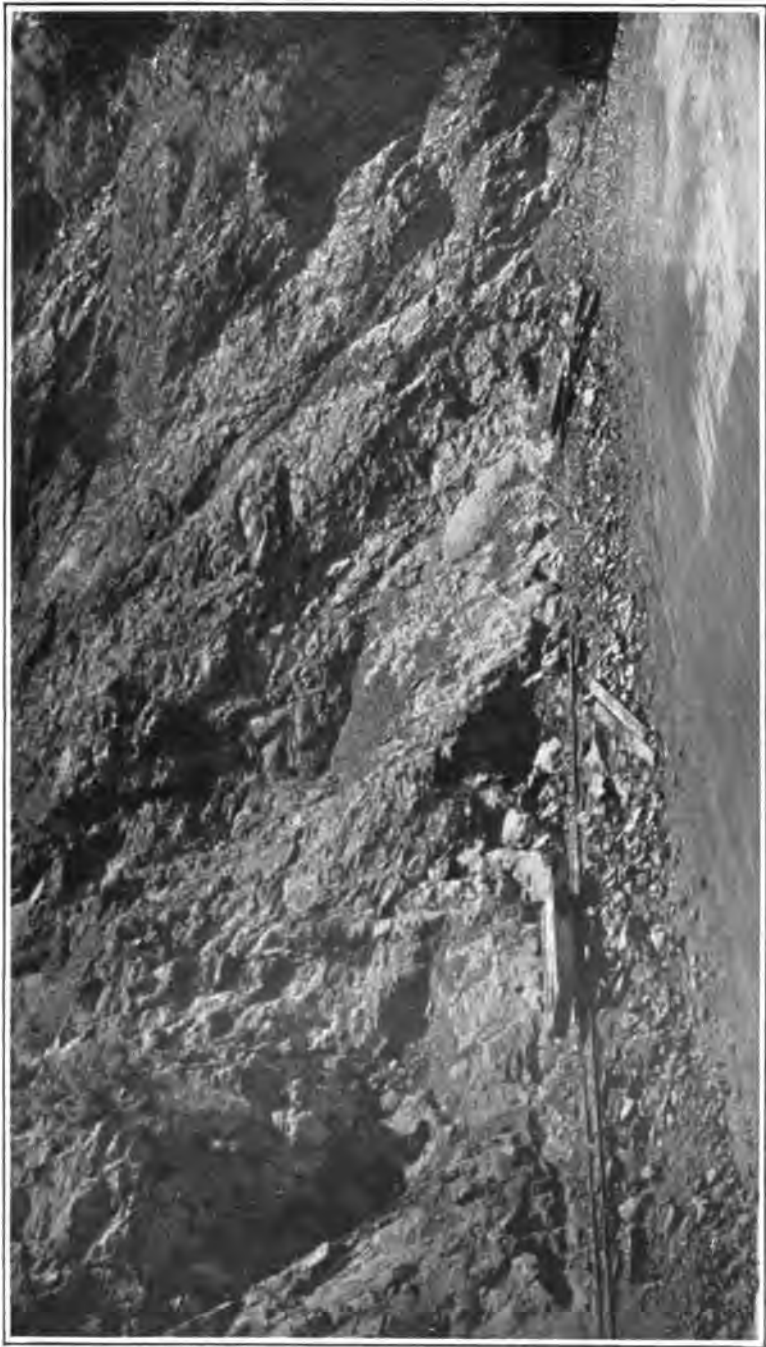
8. OIL SHALE NEAR GILLULY, UTAH.



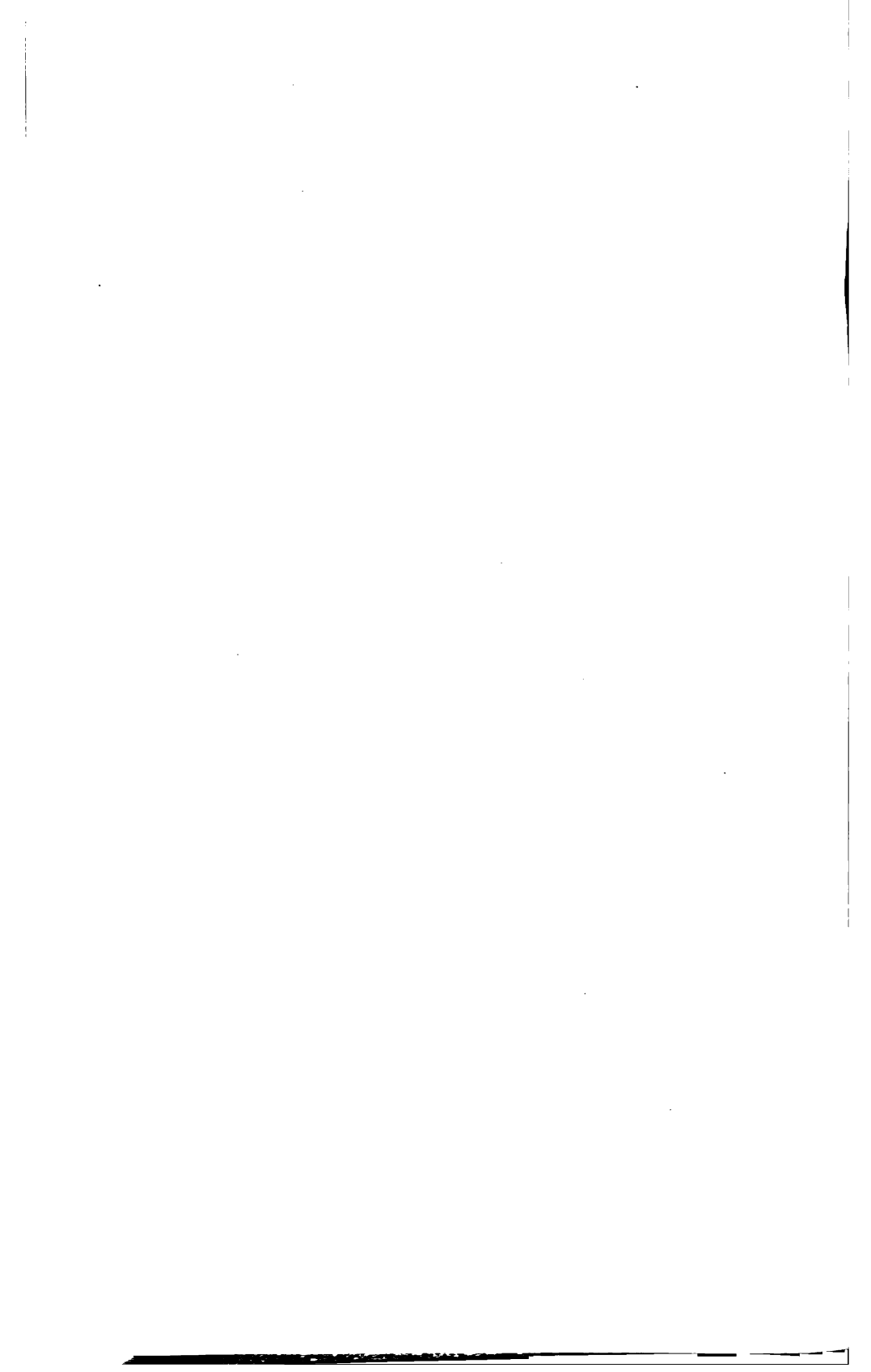
A. THIN-BEDDED OIL SHALE NEAR WATSON, UTAH.



B. MASSIVE CONTORTED OIL SHALE NEAR DE BEQUE, COLO.



MONTEREY OR DIATOMACEOUS OIL SHALE NEAR CASMALIA, CALIF.



thus distinguishing it from ordinary carbonaceous shale. Weathering, as noted above, affects the paper shale much more than the massive.

The average specific gravity of the massive shale of this formation is about 2 (see p. 152), the richer layers having the lower specific gravity. Oil and ammonia yield seem to be directly proportional, the leaner beds containing less nitrogen. (See p. 30.)

OIL SHALES OF NEVADA.

The oil shales of Nevada, probably of Green River age, are mostly in the eastern district near Elko and Carlin. They differ physically and chemically from the shales of the Green River formation and are usually somewhat lighter colored for the same yield of oil.

The Nevada shales generally lie in somewhat shallow basins of considerable extent. As a rule the beds are rather thin, dip steeply, and are extensively folded and faulted. Mining conditions are not as favorable as in most parts of the Green River formation.

The richest seam of oil shale in Nevada known to the writer is about 4 feet thick; one sample taken across the face yielded 32 gallons of oil per ton and contained 0.43 per cent of nitrogen, equivalent to 40.5 pounds of ammonium sulphate to the ton. The richest 2 feet of this seam yielded over 60 gallons of oil per ton. This bed dips at an angle of about 23°. Above and below it are much thicker beds of leaner shales, probably of little economic importance.

OIL SHALES OF CALIFORNIA.

The California deposits for the most part are hardly true oil shales, as the greater part of the oil obtained from them occurs as such and can be extracted by suitable solvents. The most extensive deposits in this State are part of the Monterey formation of Tertiary age, and physically and chemically are much different from the oil shales of Scotland and from other oil shales in the United States. The mineral matter of the shale is diatomaceous, and the beds that yield oil occur in massive formation. (Pl. IV.) They are much softer than the other oil shales mentioned, not at all flexible, brown or brownish-yellow in color, and when freshly broken smell strongly of petroleum. Their oil yield differs greatly from place to place, but the average is not high; even the best do not yield much over 20 gallons to the ton, and their nitrogen content is low. In many places the deposits are thick and accessible; so notwithstanding their low yield of oil, they have commercial possibilities. Most geologists consider the Monterey shales to have been the origin of the oil in some of the oil fields of California.

OIL SHALES OF MONTANA.

The oil shales of Montana lie at two distinct geologic horizons, one in the Phosphoria formation, probably of Permian age, and the other in Tertiary beds.²⁷ The Phosphoria shales of western Montana are characterized by a rather high percentage of phosphates.

Near Dillon there are beds 3 feet or more thick that will probably yield up to 30 gallons of oil to the ton. The phosphate beds, associated with the shales, are possibly of future importance as a commercial source of phosphates.

The Phosphoria shales of this region are dark brown or black, give a brownish streak, and the richer portions are frequently oolitic (in rounded form or pebbles). On weathered surfaces the shales exhibit a great variety of colors, a bluish-white usually predominating. On fractured surfaces the shales are often slickensided. When freshly broken or rubbed they give off an odor of petroleum and will burn rather freely when placed in a fire. They carry a little pyrite disseminated in very small grains.

The richest of the shales in this locality will probably not yield more than 30 gallons of oil to the ton, and apparently no beds of workable thickness are as rich as this. However, seams of workable thickness will yield up to 20 gallons and contain as high as 0.77 per cent nitrogen (equivalent to 71 pounds of ammonium sulphate per ton) and about 2.0 per cent of phosphorus calculated as phosphoric pentoxide (P_2O_5). Beds associated with the oil shale contain as high as 24 per cent of P_2O_5 . There is no known method of treating the shales for both their oil and phosphate content at the same time.

The shales of the Phosphoria formation outcrop along the slopes of the chief mountain ranges. In contrast with the shales of the Green River formation, the Phosphoria shales in general are in steeply dipping beds that are extensively faulted.

The Tertiary oil shales of this district lie in long, narrow basins and are of small extent; they are interstratified with sandstones, sandy shales, and lignite. The shales occur at about the middle of the Tertiary beds, are light brown, on weathered surfaces nearly white, and break on weathering into thin, more or less flexible laminae. In contrast with the Phosphoria shales, they do not smell of petroleum when freshly broken, but will burn when exposed to a hot flame. Samples from some beds, up to 5 feet thick, yielded as much as 24 gallons of oil per ton on laboratory distillation and contained from 0.1 to 0.9 per cent nitrogen.

²⁷ Bowen, C. F., Phosphatic oil shales near Dell and Dillon, Beaverhead County, Montana: U. S. Geol. Survey, Bull. No. 661, 1918, pp. 315-320.

Condit, D. D., Oil shale in western Montana, southeastern Idaho, and adjacent parts of Wyoming and Utah: U. S. Geol. Survey, Bull. No. 711B, 1919, 26 pp.

The Phosphoria formation also outcrops in southeastern Idaho and adjacent parts of Wyoming and Utah, where it carries black and brown shales that are practically barren of oil.

OIL SHALES OF THE EASTERN UNITED STATES.

Black shales of Devonian age, capable of yielding possibly economic quantities of oil, occur in the States of Indiana, Illinois, Kentucky, Ohio, New York, Pennsylvania, West Virginia, and Tennessee.²² Shales of the same age yielding notable quantities of oil have been reported in Missouri, Kansas, and Arkansas.

Most of the economically important black shales of the States mentioned belong to the upper Devonian or possibly in part to the lower Carboniferous. Other extensive deposits occur in the lower Devonian and in the Ordovician. In places oil shales of Carboniferous age overlie coal beds worked by open pits, and the shale has to be removed when the coal is mined.

In most places the Devonian black shales yield oil, although the average yield is low in comparison with that of the shales of the Rocky Mountain region. However, the thickness and the extent of the eastern deposits and the fact that they may be mined by easy and inexpensive methods make their development in many places seem almost as feasible as that of many of the western deposits. The Bureau of Mines has tested samples of Kentucky shale, said to be of workable thickness, that yielded 16 gallons of oil to the ton and contained 0.62 per cent of nitrogen, equivalent to a theoretical production of 58.6 pounds of ammonium sulphate per ton. Since 1921 the Bureau of Mines, cooperating with the State of Indiana, has been studying the shale resources of that State in an effort to determine the possible economic value of the resource. Some of the detailed results of the study have already been published as Reports of Investigations,²³ and others are to follow in the near future.

The Devonian shales are black or brown, extremely tough, and usually contain a relatively high percentage of sulphur—most of it present as pyrite. On long weathering the shale breaks into thin, fissile, ash-colored fragments which crumble and finally form a stiff clay. The oil produced from these shales seems of higher quality than that produced from many western shales, especially as to the amount and quality of the motor-fuel fraction. Table 25 (p. 173) contains a distillation analysis of oil made by the Bureau of Mines

²² Ashley, G. H., Oil resources of black shales of the Eastern United States: U. S. Geol. Survey, Bull. 641, 1917, pp. 311-324.

²³ Beeves, John E., The New Albany shale of Indiana: Bureau of Mines Reports of Investigations, Serial No. 2390, August, 1922, 8 pp.; A section through the New Albany shale: Serial No. 2425, December, 1922, 5 pp.

from unweathered Indiana shale. This oil is characteristic of that generally yielded by the black Devonian shales of the Eastern States.

NATURE AND ORIGIN OF OIL SHALE

DEFINITION OF OIL SHALE.

The writer defines oil shale as follows: "Oil shale is a compact, laminated rock of sedimentary origin, yielding over 33 per cent of ash and containing organic matter that yields oil when distilled but not appreciably when extracted with the ordinary solvents for petroleum."

According to Conacher,²⁰ "oil shales and torbanites form a group of materials which have in common the characteristic that on distillation they yield a product consisting typically of paraffin and olefins, and this feature is the source of their industrial importance."

Ashley²¹ says that the line between a coal and a shale has never been sharply drawn, but he makes the suggestion "that material which, when burned, breaks down and yields an ash that goes through the grate bars and shows no tendency to maintain its original shape is a coal, and that material which on burning yields an ash that tends to maintain its original shape is a shale. The exact percentage of ash that should distinguish a coal from a shale can not yet be given, but until more exact figures are available, it is suggested that material that yields less than 33 per cent of ash be considered a coal."

Thiessen states²² that "a shale is generally defined as a rock formed by the consolidation of clay, mud, or silt, having a finely stratified, laminated, or fissile structure. When such a rock contains organic matter it is termed carbonaceous or black shale; when the matter is of a bituminous nature it is called bituminous shale, and when rich in bituminous substances, yielding oil and gas on distillation, it is called oil shale."

The definition given at the head of this section may be generally applied to all true oil shales, but it excludes much material which has been considered of importance as a source of oil by distillation, such as the diatomaceous shales of the Monterey formation in California. A large part of the organic matter (probably oil as such) in these shales can be extracted with certain solvents for petroleum. Practically all shales will yield small amounts of bitumens or hydro-

²⁰ Conacher, H. R. J., A study of oil shales and torbanites: Trans. Geol. Soc. Glasgow, vol. 16, part 2, 1917, p. 164.

²¹ Ashley, G. H., Cannel coal in the United States: U. S. Geol. Survey, Bull. 659, 1918, p. 11.

²² Thiessen, Reinhardt, Origin and composition of certain oil shales: Econ. Geol., vol. 16, Nos. 4 and 5, June, July, August, 1921, p. 289.

carbons when extracted with various solvents. Table 2, below, indicates the solubilities of different shales.

In Table 2 the second column shows the amount of oil produced from the shale by distillation in the Bureau of Mines type of assay retort; the third column shows the amount, expressed as a percentage by weight of the raw shale; the columns under the heading "Extraction with solvent" show, first, the percentage (by weight) of the shale extracted by the particular solvent, and, second, the relative amount of extract produced, expressed as a percentage of the oil yield by destructive distillation.

Similar results have been obtained with Scottish shales by Steuart.²² This investigator found that Broxburn shale is soluble in various solvents as follows: Ether, 1.66 per cent; carbon bisulphide, 2.04 per cent; shale gasoline, 1.79 per cent; and a mixture of equal parts of shale gasoline and ether, 1.95 per cent.

The amount soluble, based on the weight of the shale, is small in all cases, but it is to be noted that a considerable percentage of the organic matter in the shale is soluble in most of the solvents used. A solubility of 2 per cent means an extraction of 40 pounds, or, roughly, 5 gallons to the ton.

The extracts in no manner resemble the products of distillation, nor can they be considered oils as the word is generally understood; but, without exception, the organic matter of all shales thus far examined is appreciably soluble in petroleum solvents. In other words, all these shales contain appreciable amounts of bitumen, though it is probably correct to say that they contain little or no oil as such. Oil shales contain a substance, or substances, usually classed as a pyro-bitumen, that by destructive distillation, or pyrolysis,²⁴ yields oils somewhat similar to petroleum. These substances have been termed "kerogen" from two Greek words meaning producer of wax.

According to Steuart,²⁵ Prof. Crum Brown suggested "the term 'kerogen' to express the carbonaceous matter in shale that gives rise to crude oil on distillation."

COMPOSITION OF OIL SHALE.

What kerogen is and how it formed are subjects of much controversy. The various theories as to its origin are presented later (pp. 33 to 39). When by proper treatment it is separated from the mineral part of the shale, it resembles a very old, dry leaf mold. Under the microscope kerogen is seen scattered through the ground mass

²² Steuart, D. R., *Oil shales of the Lothians, Part III, The chemistry of the oil shales: Memoir Geol. Survey Scotland*, 2d ed., 1912, p. 159.

²⁴ The word "pyrolysis" has been suggested by Mr. W. A. Hamor.

²⁵ Steuart, D. R., work cited, p. 143.

of the shale as small globules or irregular streaks, ranging in color from yellow or reddish yellow to dark brown or nearly black; its abundance in a given bed is a measure of oil-yielding capacity. Kerogen is undoubtedly the source of the oils obtained by retorting the shale.

TABLE 2.—*Solubilities of oil shales in various solvents for petroleum.**

| Source of shale. | Oil yield by distillation. | | Extraction with solvent— | | | | | |
|---------------------------------|----------------------------|---------------------------|--------------------------|---------------------------------|-----------------------|---------------------------------|-------------------|---------------------------------|
| | Gallons per ton. | Per centage of raw shale. | By carbon tetrachloride. | | By carbon bisulphide. | | By acetone. | |
| | | | Per cent soluble. | Per cent of distillation yield. | Per cent soluble. | Per cent of distillation yield. | Per cent soluble. | Per cent of distillation yield. |
| Kentucky..... | 18.22 | 7.28 | 0.037 | 0.51 | 0.015 | 1.44 | | |
| Soldier Summit, Utah..... | 44.60 | 16.64 | .74 | 4.45 | .76 | 4.57 | 0.53 | 3.16 |
| De Beque, Colo..... | 35.75 | 13.24 | 2.04 | 15.42 | 1.85 | 12.97 | 1.33 | 10.04 |
| Green River, Wyo..... | 58.65 | 22.67 | 1.195 | 5.27 | 1.27 | 5.53 | 1.22 | 5.35 |
| Ione, Calif. ^b | 52.00 | 19.65 | 7.555 | 33.46 | 5.63 | 29.68 | 10.96 | 55.80 |

| Source of shale. | Oil yield by distillation. | | Extraction with solvent— | | | | | |
|---------------------------------|----------------------------|---------------------------|--------------------------|---------------------------------|-------------------|---------------------------------|-------------------|---------------------------------|
| | Gallons per ton. | Per centage of raw shale. | By ether. | | By benzol. | | By chloroform. | |
| | | | Per cent soluble. | Per cent of distillation yield. | Per cent soluble. | Per cent of distillation yield. | Per cent soluble. | Per cent of distillation yield. |
| Kentucky..... | 18.22 | 7.28 | | | 0.06 | 0.82 | 0.14 | 1.99 |
| Soldier Summit, Utah..... | 44.60 | 16.64 | 9.745 | 4.45 | .91 | 5.47 | 1.06 | 6.32 |
| De Beque, Colo..... | 35.75 | 13.24 | | | 2.23 | 16.84 | 2.41 | 18.22 |
| Green River, Wyo..... | 58.65 | 22.67 | | | 1.37 | 6.02 | 1.75 | 7.72 |
| Ione, Calif. ^b | 52.00 | 19.65 | | | | | 10.16 | 51.73 |

* Gavin, Martin J., and Aydelotte, John T., Solubility of oil shales in solvents for petroleum: Reports of investigations, Bureau of Mines, Serial No. 2313, January, 1922.

^b Considered a lignite by most investigators.

Naturally the mineral matter of oil shales is largely clay, or aluminum silicate, and Scottish shales are composed of it almost entirely. Shales from different localities differ in the composition of their mineral matter, but the examples given in Table 3 may be taken as fairly representative. Attention is directed to the difference between the Scottish and the Green River shales, particularly as to the alkaline constituents.

TABLE 3.—Analyses of ash of various representative oil shales.

| Shale. | SiO ₂ , per cent. | Al ₂ O ₃ and Fe ₂ O ₃ , per cent. | CaO, per cent. | MgO, per cent. | Total, per cent. | Remarks. |
|------------------------------------|------------------------------|---|----------------|----------------|------------------|----------------------------|
| d, good average s..... | 55.6 | 34.77 | 1.55 | Trace. | 100.73 | Sulphur, 0.94. |
| d-Broxburn, seam s..... | 49.72 | 35.6 | 2.4 | 2.20 | 89.92 | |
| s, England s..... | 49.5 | 30.5 | 11.7 | 1.2 | 92.9 | |
| lia, kerosene shale s..... | 29.6 | 67.4 | 1.4 | .3 | 100.63 | |
| lia, Calif., Monterey shale s..... | 75.8 | 19.1 | 1.4 | .9 | 97.2 | |
| Utah s..... | 50.4 | 20.9 | 14.8 | 6.8 | 92.9 | Bureau of Mines No. P1006. |
| Nev. s..... | 65.5 | 25.5 | .6 | .8 | 92.4 | |
| ity, Ky. s..... | 52.0 | 19.1 | 12.5 | 8.2 | 91.8 | |
| que, Colo. s..... | 48.8 | 17.9 | 16.6 | 8.6 | 91.9 | |
| s..... | 47.3 | 18.2 | 19.3 | 8.4 | 93.2 | |
| n, Utah s..... | 45.8 | 16.4 | 23.9 | 7.9 | 94.0 | Bureau of Mines No. P1010. |
| s..... | 46.8 | 17.5 | 23.9 | 8.0 | 96.2 | Bureau of Mines No. P1011. |
| River, Wyo. s..... | 38.9 | 12.4 | 38.3 | 4.9 | 94.5 | Bureau of Mines No. P1012. |
| s..... | 41.9 | 18.8 | 17.6 | 10.9 | 89.2 | Bureau of Mines No. P1013. |
| Calif. s..... | 43.0 | 38.1 | 8.7 | 2.5 | 92.3 | Bureau of Mines No. P1014. |
| r Summit, Utah s..... | 44.8 | 17.1 | 23.9 | 5.4 | 91.2 | Bureau of Mines No. P1015. |

Donacher, H. R. J., Personal communication.
 Includes 8.27 per cent soluble in water containing 0.92 per cent SO₂.
 Stuart, D. R., The oil shales of the Lothians. Part III, The chemistry of the oil shales: *Memor Geol. Survey Scotland*, 2d ed., 1912, p. 159.
 Forbes-Leslie, William, The Norfolk oil shales: *Jour. Inst. Petrol. Tech.*, vol. 3, December, 1916, p. 22.
 Carne, J. E., Kerosene shale deposits of New South Wales: *Memor Geol. Survey New South Wales*, p. 272.
 Includes K₂O, 0.748 per cent, Na₂O, 0.322 per cent; phosphoric acid, 0.744 per cent. Total ash in raw shale, 16.12 per cent.
 B. Brighton, Bureau of Mines, analyst.

Table 4 gives the proximate analysis of various American and foreign shales. The reader should note that in many American shales the carbonate percentages are so high that the determinations of volatile matter and fixed carbon are of doubtful accuracy.

TABLE 4.—Proximate analyses of representative American and foreign oil shales.

| Shale. | Moisture. | Volatile. | Fixed carbon. | Ash. |
|---|-----------|-----------|---------------|-------|
| Broxburn seam, Scotland s..... | 7.50 | 16.5 | 55.2 | 70.8 |
| Broxburn seam, Scotland s..... | 6.47 | 19.25 | 58.37 | 65.91 |
| Broxburn seam, Scotland s..... | 1.92 | 22.41 | 1.81 | 73.86 |
| Broxburn No. 5 seam, Scotland s..... | 1.70 | 21.35 | 5.03 | 71.02 |
| Broxburn, Australia, best export s..... | 1.02 | 75.20 | 15.30 | 8.48 |
| Broxburn, Australia, retorting shale s..... | .85 | 45.32 | 31.27 | 22.55 |
| Broxburn, Nev. s..... | 2.40 | 29.3 | 14.2 | 64.4 |
| Broxburn, Albany, Ind. s..... | .85 | 15.86 | 5.54 | 77.78 |
| Broxburn, City, Ky. s..... | .70 | 28.6 | 1.7 | 70.0 |
| Broxburn Summit, Utah s..... | .50 | 38.10 | | 61.1 |
| Broxburn, Colo. s..... | .10 | 45.70 | | 54.2 |
| Broxburn, Do. s..... | .10 | 42.30 | | 57.6 |
| Broxburn, Utah s..... | 1.10 | 42.20 | | 56.7 |
| Broxburn, Do. s..... | 1.05 | 34.35 | | 64.6 |
| Broxburn River, Wyo. s..... | .25 | 34.95 | | 64.8 |
| Broxburn, Do. s..... | .15 | 48.05 | | 51.8 |
| Broxburn, Calif. s..... | 8.50 | 62.90 | | 28.6 |

Stuart, D. R., The oil shales of the Lothians. Part III, The chemistry of the oil shales: *Memor Geol. Survey Scotland*, 2d ed., 1912, p. 158.
 Includes sulphur.
 Carne, J. E., Kerosene shale deposits of New South Wales: *Memor Geol. Survey New South Wales*, p. 264.
 Stuart, D. R., Work cited, p. 163.
 T. B. Brighton, Bureau of Mines, analyst.
 Ash and fixed carbon determinations on raw shale. Percentages of carbonates in raw shale.

Nitrogen content is a measure of recoverable ammonia, and Table 5 and Figure 2 show the nitrogen content of several foreign and American shales.

TABLE 5.—*Nitrogen content of various representative oil shales.**

| No. on curve, Fig. 2. | Shale. | Nitrogen, per cent. | Theoretical available (NH ₄) ₂ SO ₄ , pounds per ton. | Oil yield, gallons per ton. ^b |
|-----------------------|---------------------------|---------------------|---|--|
| 1..... | Elko, Nev..... | 0.43 | 40.5 | 32.52 |
| 2..... | do..... | .15 | 14.1 | 7.28 |
| 3..... | Clay City, Ky..... | .62 | 58.5 | 18.22 |
| 4..... | Soldier Summit, Utah..... | .58 | 54.7 | 44.60 |
| 5..... | De Beque, Colo..... | .96 | 90.5 | 60.15 |
| 6..... | do..... | .76 | 71.7 | 48.77 |
| 7..... | Dragon, Utah..... | .74 | 69.8 | 41.66 |
| 8..... | do..... | .40 | 37.7 | 21.70 |
| 9..... | Green River, Wyo..... | .35 | 33.0 | 23.45 |
| 10..... | do..... | .83 | 78.2 | 58.65 |
| 11..... | Icone, Calif..... | .39 | 36.8 | 52.00 |

* Determined by Kjeldahl-Gunning method, T. B. Brighton, analyst.

^b Laboratory report.

The nitrogen content of oil shales ranges from a trace to over 1 per cent. In Scotland the nitrogen is the source of the most important by-product obtained from the shales—ammonium sulphate. The value of the nitrogen in the American shales is discussed on page 120. Apparently the nitrogen exists in combination with the kero-gen or oil-yielding matter. Although in Scottish oil shale the nitro-gen content is said to vary inversely as the oil yield, in shales in the United States the reverse seems to hold. The almost linear relation-ship between the oil yield and nitrogen content of shales of the Green River formation is shown to best advantage in Figure 2. Samples represented by points 1 and 2 on that curve, although from Elko, Nev., and therefore not certainly of Green River age, seem to fit in well with the general relationship. Samples represented by points 3 and 11, Kentucky oil shale and California lignite, do not follow the straight-line relationship of the other shales represented. However, an examination of analyses of oil shales from different parts of the United States indicates that there is always a direct relationship between oil yield and nitrogen content of the shale, although the factor that might be used to show the relationship differs for shales of different districts. For example, the Devonian shales of the eastern United States contain more nitrogen for a given oil yield than do the Green River shales, but there is a direct and apparently linear relationship between nitrogen content and oil yield for the Devonian shales. As probably almost all of the nitro-gen in a shale is present in organic form, it is to be expected that the amount present would be in proportion to the amount of organic matter in the shale and hence to the oil yield.

It has been stated that the nitrogen content of the Scottish shales shows an inverse relationship to the oil yield, but it is difficult to explain why the Scottish shales should differ in this respect from

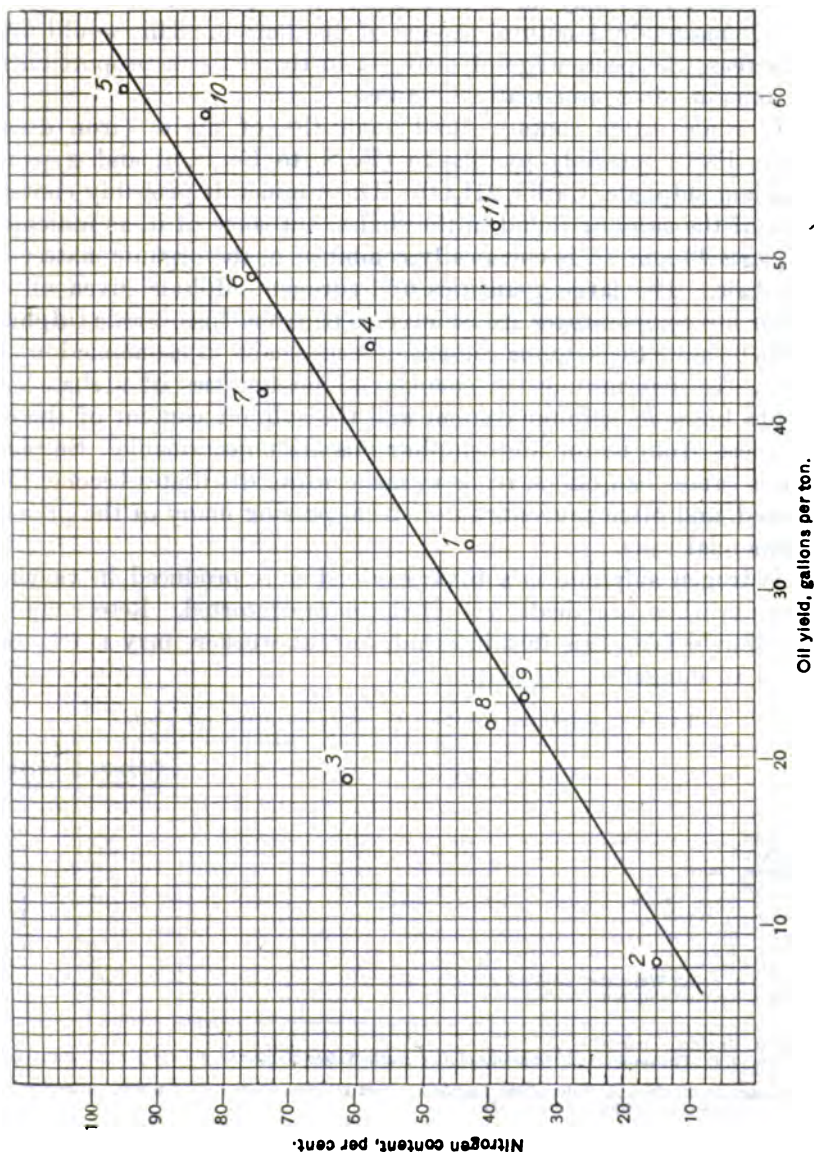


FIGURE 2.—Curves showing relationship between nitrogen content and oil yield of shales of Green River formation.

those of this country. However, most of the rich Scottish shales were worked out before ammonia recovery was undertaken by the shale producers in Scotland. The belief in the inverse relationship may be based on the fact that no ammonia was recovered from the

richer shales, whereas, as the shales mined yielded less and less oil improved methods and greater necessity of recovering by-product have steadily increased the yield of ammonia. On the other hand the leaner shales may have been deprived of part of their original organic matter by a natural process of distillation. This would tend to increase the percentage of nitrogen in the shale in proportion to the amount of organic matter distilled.

Oil shales often contain small quantities of pyrite (iron disulphide, FeS_2), possibly pyrrhotite (Fe_7S_{10} to $\text{Fe}_{11}\text{S}_{12}$), and gypsum (calcium sulphate, $\text{CaSO}_4 \cdot 2\text{H}_2\text{O}$). These minerals probably contain most of the sulphur found in the shales, but some of it, as indicated on pages 43 and 47, is chemically combined in the organic matter of the shale. The large quantities of hydrogen sulphide given off at rather low temperatures are believed to be from this organic sulphur. Table 6 shows the sulphur content of some shales of economic importance. In determining the commercial possibilities of a shale one should know its sulphur content and the sulphur content of the oil produced from it, for high-sulphur oils are objectionable. Sulphur in oils causes bad odors, tends to make white distillates become discolored, and often prevents an oil from passing many of the present commercial tests.

Hydrogen sulphide is a toxic gas and it is produced in varying amounts when practically all oil shales are retorted. Retort operators should keep this fact in mind, so that workers may not be exposed to dangerous concentrations.

TABLE 6.—*Sulphur content of representative oil shales.*

| Source of shale. | Sulphur, per cent. | Sample No. |
|---|--------------------|------------|
| Scotland ^a | 2.66 | P1000 |
| Elko, Nev. ^a | 4.10 | P1002 |
| De Beque, Colo. ^a | 1.16 | P1011 |
| Ione, Calif. ^a | 2.13 | P1047 |
| Soldier Summit, Utah ^a | 2.16 | P1048 |
| Green River, Wyo. ^a | 3.84 | P1055 |
| Dillon, Mont. ^a | 2.98 | P1068 |
| Clay City, Ky. ^a | 3.16 | P1081 |
| Watson, Utah ^a | .48 | P1086 |
| Joadja, New South Wales (best grade) ^b | .31 | |
| Scotland, Roman Camp mine Jubilee seam ^c | .35 | |

^a T. B. Brighton, analyst.

^b Carne, J. E., The kerosene shale deposits of New South Wales: Memoir Geol. Survey New South Wales, 1908, p. 228.

^c Stuart, D. R., The oil shales of the Lothians. Part III, The chemistry of the oil shales: Memoir Geol. Survey Scotland, 2d ed., 1912, p. 154.

POTASH AND RARE METALS FROM OIL SHALES.

Many oil shales contain small quantities of potash, but even after complete retorting—that is, after removal of all volatile and fixed carbon—the amount of water-soluble potash obtained is too small to make recovery feasible. It is not believed that oil shales can be worked commercially for their potash.

—All oil shales may not have been formed in the same manner or the same kind of organic matter, and that different conclusions have been reached by different investigators because they examined different shales. Likewise many of the theories may be correct in part and many of the suggested methods of origin may have had nothing to do with the formation of oil-shale deposits.

—Microscopic examination of oil shales from all parts of the world indicates that they are composed of—

-) Minute carbonized or partly carbonized fragments of plants—showing cellular structure—small spores, pollen, and the like.
-) Yellow or reddish-yellow, more or less spherical bodies, regarded as algæ, spores, spore cases, grains of resin, or globules of oil.
-) Irregular streaks of reddish-yellow, dark-brown, or opaque material.

—) Shells of small crustaceans and parts of the skeletons and scales of fish.

—) Mineral matter, such as sand grains and pyrite crystals.

There is little evidence to indicate that animal life had much to do with the formation of the oil-yielding material of shales, and much evidence to the contrary. Possibly the animal remains are the source of much of the nitrogen and sulphur.

In general the following opinion prevails: That the yellowish globules and yellowish or brown, often nearly black, streaks in the shales are the sources of the oil obtained by distilling the shales and are probably of vegetable origin, or are petroleum as such or in some altered form. The fact that little real oil can be extracted from shales by ordinary solvents of petroleum seems to preclude the idea that the globules are petroleum as such.

Under the microscope some shales show practically no organic matter, except the irregular globular yellowish bodies; others contain few globular bodies, the organic matter being in irregular dark-colored streaks. The first shale worked in Scotland, torbanite, is an example of the former; many of the shales of the Green River formation in the United States are examples of the latter. All degrees of gradations exist between the two extremes. Some investigators differentiate the origin of torbanite from that of oil shale, but the consensus of opinion is that the origin is the same.

It is pretty generally agreed that the mineral matter of the shales was laid down in fresh-water swamps or lagoons, but opinions on the origin of the oil-yielding substance differ. Most investigators have examined shales such as those of Scotland, in which the yellowish bodies predominate, and less information has been published on the origin of the irregular dark-colored streaks and globules present in American shales.

richer shales, whereas, as the shales mined yielded less and less oil, improved methods and greater necessity of recovering by-products have steadily increased the yield of ammonia. On the other hand, the leaner shales may have been deprived of part of their original organic matter by a natural process of distillation. This would tend to increase the percentage of nitrogen in the shale in proportion to the amount of organic matter distilled.

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| De Beque, Colo. ^a | 1.16 | P1011 |
| Idaho, Calif. ^a | 2.13 | P1047 |
| Soldier Summit, Utah ^a | 2.16 | P1048 |
| Green River, Wyo. ^a | 3.84 | P1055 |
| Dillon, Mont. ^a | 2.98 | P1058 |
| Clay City, Ky. ^a | 3.16 | P1061 |
| Watson, Utah ^a | .48 | P1066 |
| Joadja, New South Wales (best grade) ^b | .31 | |
| Scotland, Roman Camp mine Jubilee seam ^c | .35 | |

^a T. B. Brighton, analyst.

^b Carne, J. E., The karosene shale deposits of New South Wales: Memoir Geol. Survey New South Wales, 1903, p. 225.

^c Stewart, D. R., The oil shales of the Lothians. Part III, The chemistry of the oil shales: Memoir Geol. Survey Scotland, 2d ed., 1912, p. 154.

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Much discussion has been aroused by reports that various investigators have obtained gold, silver, and platinum from oil shales, usually by secret processes. Assays, made by the Bureau of Mines, of samples of oil shale said to carry 1 ounce and more of gold per ton, with corresponding silver and platinum, have indicated that although gold was present, it existed only as traces of the order of 0.01 ounce (about 20 cents) per ton of shale. No silver or platinum has been detected.

As many of those who claim to have found gold and other precious metals in paying quantities state that the metals can not be detected or recovered unless they are "unlocked" by preliminary treatment, the bureau made a series of tests and assays to determine the validity of these claims. Several of the tests were made in the presence of and according to the methods said to be necessary by one of the private investigators who has received much publicity from his reputed discovery of methods of recovering precious metals from oil shales. A shale from which this investigator claimed he had recovered \$7.50 in gold per ton was used in the tests. Assays were made of the raw shale, spent shale, decarbonized shale, decarbonized shale treated with chlorine at temperatures most favorable for the formation of gold chloride, and of shale decomposed by chromic acid. All the assay returns checked closely and indicated that the shale carried less than 65 cents in gold. The fire assay return on the raw shale was the same as on the shale subjected to the various preliminary treatments.

This work and similar assays by the bureau, as well as by other investigators, indicate little chance of gold and similar precious metals being commercially recoverable from oil shale. There is no particular reason why gold should occur in oil shale any more than it does in sea water, and although it is possible that small partings or fissures in the shales—not the shale itself—might carry notable quantities of these metals, it is not believed that oil shale will ever be a commercial source of gold, silver, platinum, or allied metals.

KEROGEN, THE OIL-YIELDING MATERIAL OF OIL SHALES.

Little information is available as to the chemical composition of the organic oil-yielding material of oil shales, the so-called kerogen. From a careful study of Scottish shales, Mills²² gives the composition of an average sample as:

Ultimate analysis of Scottish shale.

| | Per cent. |
|---------------|-----------|
| Carbon..... | 25.27 |
| Hydrogen..... | 3.67 |
| Oxygen..... | 5.65 |

²² Mills, E. J., *Destructive distillation*. 4th ed., London, 1892, p. 50.

| | |
|----------------|---------|
| Nitrogen | 1. 14 |
| Sulphur | . 49 |
| Ash | 63. 78 |
| | 100. 00 |

Undoubtedly part of the nitrogen and sulphur of the shale are in combination with the kerogen, but excluding these and the ash, Mills finds the chemical composition of kerogen to be as follows :

Composition of kerogen of Scottish shale.

| | |
|----------------|-----------|
| | Per cent. |
| Carbon | 73. 05 |
| Hydrogen | 10. 62 |
| Oxygen | 16. 33 |
| | 100. 00 |

This corresponds to the empirical formula $C_6H_{10}O$. Mills therefore believes that the kerogen molecule of Scottish shale can be represented by the formula $n(C_6H_{10}O)$, n being undoubtedly a large number.

Certainly the kerogen molecule, if kerogen is a definite chemical substance, is large and complicated. Although the kerogen in different shales shows certain resemblances under the microscope, it is not probable that the kerogen of all shales has the same composition, nor that the kerogen of any one shale is a definite chemical compound. The properties of the oil from different shales differ widely, even when the shales are retorted under identical conditions, as is shown by Table 7.

TABLE 7.—Yield and nature of oil from different American oil shales.

[Retorted under identical conditions in Bureau of Mines laboratory retort.]

| No. | Source of shale. | Yield of oil, gallons per ton. | Specific gravity of oil. | Setting point of oil, °C. | Sample No. |
|-----|---|--------------------------------|--------------------------|---------------------------|------------|
| 1 | Elko, Nev. ^a | 32. 50 | 0. 883 | 28. 5 | P1002 |
| 2 |do. ^a | 7. 28 | . 882 | 25. 0 | P1004 |
| 3 | Clay City, Ky. ^a | 18. 22 | . 924 | 15. 0 | P1006 |
| 4 | De Beque, Colo. ^a | 60. 15 | . 899 | 25. 7 | P1010 |
| 5 |do. ^a | 48. 77 | . 906 | 25. 0 | P1011 |
| 6 | Dragon, Utah ^a | 41. 65 | . 904 | 25. 7 | P1012 |
| 7 |do. ^a | 21. 70 | . 897 | 27. 0 | P1013 |
| 8 | Green River, Wyo. ^a | 23. 45 | . 898 | 33. 0 | P1014 |
| 9 |do. ^a | 58. 65 | . 899 | 32. 0 | P1015 |
| 10 | Iona, Calif. ^a | 52. 00 | . 907 | 44. 0 | P1047 |
| 11 | Soldier Summit, Utah ^b | 44. 60 | . 881 | 31. 0 | P1049 |
| 12 | Scotland ^b | 17. 64 | . 886 | 34. 5 | P1000 |

^a Average of duplicate closely agreeing assays.

^b Average of several closely agreeing assays.

There is much conflict of evidence as to the origin of oil shales, and evidently much research work must be done before the question can be definitely settled. In the pages following, which present various theories of origin, it is well to keep in mind the probability

that all oil shales may not have been formed in the same manner or from the same kind of organic matter, and that different conclusions have been reached by different investigators because they examined different shales. Likewise many of the theories may be correct in part, and many of the suggested methods of origin may have had something to do with the formation of oil-shale deposits.

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(3) Irregular streaks of reddish-yellow, dark-brown, or opaque material.

(4) Shells of small crustaceans and parts of the skeletons and scales of fish.

(5) Mineral matter, such as sand grains and pyrite crystals.

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Bertrand and Renault³⁷ believe that the torbanites were composed of accumulations of microscopic gelatinous algæ, which were preserved from complete decomposition by an antiseptic solution, possibly derived from their own decay, and that the antiseptic solution is now represented by the opaque organic matter of the shales.

Jeffery³⁸ attempts to disprove the algal theory, and suggests that the supposed algæ are deeply sculptured coats of the spores of vascular cryptogams.

Seward³⁹ believes that the so-called organic bodies might after all be found to be of inorganic origin.

David White⁴⁰ originally accepted the algal theory of Bertrand and Renault, but later transferred his support to the spore theory of Jeffery.⁴¹

Cadell⁴² suggests that the animal matter (entomostraca and fish remains) of the shales may be the source of much of the nitrogenous matter, and that shales now devoid of fossils, but rich in oil and nitrogen, may have derived their oil-yielding material from organisms with soft bodies and no shell. Later⁴³ he suggested that oil shales are derived from an extremely macerated peat.

Steuart⁴⁴ concludes, from his own work and that of previous investigations, that "oil shales may therefore be composed of (1) vegetable matter which has been made into a pulp by maceration in water and preserved by combining with salts in solution * * *; (2) richer materials of many kinds, such as spores, which nature has provided with means for protection against decay; and (3) a proportion of animal matter." He also concludes that oil shale "may be considered a torbanite containing a larger proportion of earthy matter from its original deposition, or is simply a torbanite which has deteriorated with age." He gives this reason for the variations in the organic matter:

In accounting for the differences between different shales and between shale, torbanite, etc., we must remember that during the deposition of the sediments, as the lapse of time was very great, the climate must have varied to some extent, and the algæ and plankton generally of the lagoon must have changed

³⁷ Bertrand, C. Eg., and Renault, B., *Pila bibractensis* et le boghead d'Autun: Bull. Soc. Hist. Nat. Autun, vol. 5, 1892, p. 159.

³⁸ Jeffery, E. C., On the composition and qualities of coal: Econ. Geol., vol. 9, 1914, p. 735. The mode of origin of coal: Jour. Geol., vol. 23, 1915, p. 220.

³⁹ Seward, A. C., Fossil plants. Vol. 1, 1898, p. 178.

⁴⁰ White, David, Zinc and lead deposits of the Upper Mississippi Valley: U. S. Geol. Survey, Bull. 294, 1906, p. 26. The effect of oxygen on coal: U. S. Geol. Survey, Bull. 382, 1909, p. 50.

⁴¹ White, David, and Thiessen, R.; The origin of coal: Bull. 38, Bureau of Mines, 1913, p. 3.

⁴² Cadell, H. M., The oil shale fields of the Lothians: Trans. Inst. Min. Eng., vol. 22, 1901, pp. 314-371.

⁴³ Cadell, H. M., The story of the Forth. Glasgow, 1913, p. 8.

⁴⁴ Steuart, D. R., The oil shales of the Lothians. Part III, The chemistry of the oil shales: Memoir Geol. Survey Scotland, 2d ed., 1912, pp. 164-165.

also, together with the shore vegetation producing pollen, spores, and seeds. The differences and varieties might have arisen from changes in the proportion of, or nature of, the humus, the spores, and the animal matter. And there are also the modifications produced by age, and the different conditions to which they have been subjected during their long history in the earth, such as warmth from depth or from proximity to intruded igneous rock.

Scheithauer⁴⁵ believes that a "large number of marine animals perished on certain occasions," possibly as the result of volcanic eruptions, and states as evidence that the Scottish shale deposits have been shown to belong to the volcanic region.

Robertson⁴⁶ gives as his conclusions that "there is little resinous matter in oil shales, and that the organic matter is a decomposition product of vegetable matter, of a nature like that found in peat and cannel coal, * * * and produced by a definite combination of external conditions."

Davis⁴⁷ writes as follows regarding the microscopic studies he was making at the time of his death:

It is clear that the structureless material of the Green River shales probably originated in a collection of plant debris which has, by decomposition and the activities of bacteria and other microscopic organisms, passed into a jellylike phase, which is to be found in certain kinds of peat deposits. The plant remains that have been found characterizing the shales from every locality from which they have been examined (Green River formation) are those of microscopic algae mixed in small percentage with pollen and similar parts of higher plants. Animal remains have been very rare in the material studied, and those noted were chiefly those of insects in a very fragmental state.

It seems apparent, therefore, that the study of the microscopic structure of these shales, as seen in vertical and horizontal sections, leads to the conclusion that the material was laid down originally in water, and that it passed through a series of stages of decomposition before consolidation and lithification had taken place. The remarkably well-preserved state of the delicate plant structures which have been examined indicates very slight disturbances of the original material and an almost entire lack of changes produced by the action of metamorphosing agencies since lithification.

Cunningham-Craig⁴⁸ advances the theory that an "oil-shale field may be considered as the relics of a former oil field." That is, oil has been formed in argillaceous deposits from vegetal debris, has migrated into porous beds, and "if any argillaceous beds of sufficient absorptive capacity exist in the vicinity of oil rocks, they will gradually absorb and adsorb the heavier and unsaturated hydrocarbons, and by their smaller porosity, especially if capped by impermeous beds, will protect the adsorbed liquid from displacement by

⁴⁵ Scheithauer, W., *Shale oils and tars and their products*. London, 1918, p. 12.

⁴⁶ Robertson, J. B., *A chemical examination of the organic matter in oil shales*: Proc. Roy. Soc. Edinburgh, vol. 34, 1914, pp. 190-201.

⁴⁷ Winchester, D. E., *Oil shale in northwestern Colorado and adjacent areas*: U. S. Geol. Survey, Bull. 641, 1917, pp. 163-165.

⁴⁸ Cunningham-Craig, E. H., *Kerogen and berogen shales*: Jour. Inst. Pet. Tech., vol. 2, 1916, pp. 238-269; *A treatise on British mineral oil*. London, 1919, pp. 3-63.

lixiviation, while inspissation is gradually modifying the petroleum in the direction of kerogen." Craig also believes that torbanite, etc., differs from oil shale in origin, and says that the former may be considered "an incipient form of petroleum developed, adsorbed, and inspissated in situ."

Conacher⁴⁹ believes, after exhaustive microscopic investigations, that the "oil-yielding material consists of resin fragments (yellowish in color and in shape usually like flattened spheres) which owe their external form either to their site of origin in the plant or to attrition in transport." He believes⁵⁰ that petroleum has played no part in the origin of Scottish oil shales or torbanites, and that the opaque organic matter of the shales contains the nitrogen and represents the liquid putrefaction products of vegetable matter; further, that this matter yields the tarry products of distillation, with possibly some of the paraffins. He states that on distillation those shales containing the higher percentages of what he terms resin (the yellowish bodies) yield a larger quantity of lighter oil than those in which opaque matter predominates. "It seems that this opaque matter is of the nature of the amorphous groundmass of ordinary coal, and that from it is derived by distillation such products as are characteristic of coal tars (even low-temperature tars) while the yellow bodies are the source of typical shale products,"⁵¹ Further, he states, "evidently it is this material (the opaque matter) that yields * * * possibly the bulk of the nitrogenous compounds (including ammonia and the nitrogen combined with hydrocarbons)."

With reference to the oil shales of the Green River formation in the United States, Mr. Conacher⁵² informs the writer that though they contain yellow particles, these bodies are not like those in the Scottish oil shales and torbanites.

Mr. A. C. Thomson, chief refinery engineer of the Pumphreston Oil Co., of Scotland, has kindly given the writer some notes on a swamp in Brazil which are of interest as describing the actual formation of an oil-shale deposit. The notes are summarized below:

At Taubaté, about 50 miles from São Paulo, a shale was worked for some time. About 5 miles from this place a flat, very marshy stretch extends for two-thirds of a mile on each side of the river. When Mr. Thomson examined this marsh it was well covered with vegetation (lichens), and in muddy pools on it fish were alive. In pools nearly dry fish were dead in the mud. In exposures behind this level place soft, damp, but well-laminated shales had been

⁴⁹ Conacher, H. R. J., *A study of oil shales and torbanites*: Trans. Geol. Soc. Glasgow, vol. 16, part 2, 1916-17, pp. 164-192.

⁵⁰ Conacher, H. R. J., *Same work*, p. 185.

⁵¹ Conacher, H. R. J., *Same work*, p. 174.

⁵² Conacher, H. R. J., *Personal communication*, 1920.

noted, and to get information as to their possible formation a pit was sunk in the marsh to a depth of 20 feet, passing through alternate beds of black and brown mud. At 16 feet distinct lamination began, and between the laminations a few well-defined fish fossils were found, showing exactly the same structure as those found in the river to-day. The black mud, considered shales in formation, was deposited when the vegetable growth was of long duration. The brown mud would form the brown earthy ribs always found in shale beds and was deposited during a period of water overflow. When the water receded, vegetable growth would again begin.

Thiessen,¹⁸ after a preliminary study of a Scottish shale and of Devonian shales of Illinois, Indiana, and Kentucky, concluded—

* * * that the oil shales, as far as examined, do not contain oil as such, but that the oils derived from them are derived from organic matter contained in them. All the identifiable matter consists of plant matter or plant degradation matter. No animal matter has been recognized. The larger part of the organic matter consists of spores, the proportion of spores varying in different shales and in different strata of the same shale. A considerable amount of cuticular matter and some woody degradation matter is also present, but little or no resinous matters have been recognized.

He also says¹⁹ that the Scottish shale is very similar to the Devonian shales of the eastern part of this country.

While the same kind of constituents comprise this shale they are present in somewhat different proportions * * *. The shale also contains a slightly larger number of larger resinous-appearing dark-brown bodies of a roughly rounded shape. The larger spores, of which there are several kinds, are of different form and sculpturing, and hence are of different species than those of the Illinois shale.

THE CHEMISTRY OF OIL-SHALE DISTILLATION.

To all intents and purposes, as has been shown, oil shale contains no oil as such, the source of the oil being an organic substance termed kerogen. Although a tentative empirical formula for the kerogen of Scotch shale has been determined (see p. 34), it is more than likely that kerogen is not a definite chemical compound but is made up of molecules of various types and sizes, and that the oil-yielding materials or "kerogens" of different shales are unlike chemically.

Whatever its chemical composition, kerogen is undoubtedly of organic origin and is made up of carbon, hydrogen, oxygen, nitrogen, and sulphur; also its molecules are undoubtedly large and complex. Kerogen, as such, is of no practical value, but by heating in a retort its complex and large molecules are "cracked" to form the more simple molecules of gaseous, liquid, and solid hydrocarbons and

¹⁸ Thiessen, Reinhardt, *Origin and composition of certain oil shales: Econ. Geol.*, vol. 16, 1921, p. 298-300.

¹⁹ Thiessen, Reinhardt, *work cited*, p. 296.

their derivatives, that are more or less similar to those derived from oil-well petroleums.

HYDROCARBONS.

A hydrocarbon is an organic compound of the elements hydrogen, H, and carbon, C. Generally speaking, the simplest and lightest hydrocarbon molecules form gases, the heavier and more complex molecules form liquids, and molecules still heavier and more complex form solids. As a rule the specific gravities and boiling points of the hydrocarbons increase with the size and complexity of the component molecules.

For example, natural gas is chiefly the hydrocarbon methane, CH_4 , sometimes with small amounts of ethane, C_2H_6 , and higher hydrocarbons. Commercial gasoline, when made by ordinary distillation from a paraffin-base crude, is a mixture of several liquid hydrocarbons of the paraffin series, such as hexane, C_6H_{14} , heptane, C_7H_{16} , octane, C_8H_{18} , and others. The hydrocarbons of kerosene and the heavier fluid petroleum oils are composed of molecules still more complex and with more carbon and hydrogen atoms, such as tridecane, $\text{C}_{13}\text{H}_{28}$. The solid paraffins or waxes are generally considered to be composed of still heavier and more complex hydrocarbons, such as trikosane, $\text{C}_{23}\text{H}_{48}$, etc.

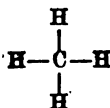
SATURATED AND UNSATURATED HYDROCARBONS.

PARAFFIN SERIES, SATURATED.

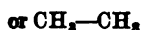
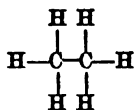
Hydrocarbons of the paraffin series are said to be saturated, because the molecules are saturated with hydrogen; that is, they are unable to take up more hydrogen atoms. All the paraffin hydrocarbons are members of the paraffin series. Their type formula is $\text{C}_n\text{H}_{2n+2}$, where n theoretically may be any number. (The hydrocarbon n . hexacontane $\text{C}_{60}\text{H}_{122}$ is the longest normal paraffin chain known.) The carbon atom is said to be quadrivalent or to have the power of combining with four other atoms, or groups of atoms, each atom or group having a combining power or valence of one. Graphically the valencies of the carbon atom may be represented.



and thus the saturated methane molecule, CH_4 , in which all these valencies are completely utilized, is represented:



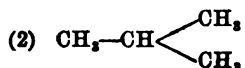
A carbon atom may be combined with hydrogen atoms, the atoms of other elements and groups of atoms, and with other carbon atoms. Thus the ethane molecule, C_2H_6 , has the graphic formula:



In the graphic formula all the valencies of the two carbon atoms are shown to be completely utilized.

As the number of atoms in a paraffin hydrocarbon molecule increases, the structure of the molecule becomes more complex, and several molecules can have the same constitutional formula but different structural relations. For example, the hydrocarbon butane, C_4H_{10} , exists in two forms:

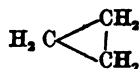
(1) $CH_3-CH_2-CH_2-CH_3$, the normal form (n butane), or



the "isomeric" form (isobutane). Butane and isobutane have quite different properties. Moreover, as hydrocarbon molecules become more complex, or larger, more isomeric forms can and do exist.

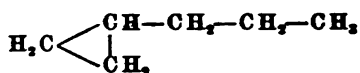
NAPHTHENE SERIES, SATURATED.

Saturated hydrocarbons of another series may be important constituents of shale oils. This is the saturated monocyclic or naphthene series, having the type formula C_nH_{2n} , but the structure of the molecules of this series is represented as a closed chain, or ring; for example, the first member of the series, the hydrocarbon trimethylene or cyclopropane, C_3H_6 , is graphically indicated:



This hydrocarbon behaves much less like a saturated hydrocarbon than the cyclic compounds with five and six carbon atoms in the ring, probably because the valencies or bonds connecting the carbon atoms of cyclopropane are bent or strained from their normal position. The properties of cyclopentane, C_5H_{10} , and cyclohexane, C_6H_{12} , are very much like those of normal pentane, C_5H_{12} , and normal hexane, C_6H_{14} , respectively.

Naphthene hydrocarbons may be complex; the graphic formula above, by addition of a side chain, may become:



propyl trimethylene.

SATURATED POLYCYCLIC HYDROCARBONS.

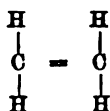
Saturated polycyclic hydrocarbons having the empirical formula $\text{C}_n\text{H}_{2n-2}$, $\text{C}_n\text{H}_{2n-4}$, $\text{C}_n\text{H}_{2n-6}$, etc., may occur in shale oils. As the class name for this group indicates, the molecular structures are built up of more than one ring. Many lubricating oils are believed to be composed of polycyclic hydrocarbons.

UNSATURATED HYDROCARBONS.

A hydrocarbon molecule containing less hydrogen than the maximum possible is said to be unsaturated; if it is an "open-chain" molecule, it contains a smaller percentage of hydrogen than a saturated "open-chain" molecule of the same number of carbon atoms. Under proper conditions, unsaturated hydrocarbons can be saturated by direct addition of hydrogen atoms, whereas direct addition of such atoms to a saturated hydrocarbon is not possible. The three unsaturated series of hydrocarbons of possible importance in shale oils are the olefin, diolefin, and acetylene series.

OLEFIN SERIES.

The type formula of the olefin series is C_nH_{2n} . The first member is the gas, ethylene, C_2H_4 , sometimes graphically represented



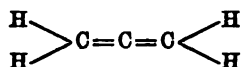
but usually $\text{CH}_2=\text{CH}_2$

In ethylene each carbon atom has an unsatisfied, strained, or uncombined valence or combining power, indicated by the double bond between the carbon atoms. The double bond does not represent a reinforcement of a single bond, but indicates a point of molecular weakness or strain. Ethylene contains 14.3 per cent of hydrogen, as compared with 20 per cent in the corresponding paraffin hydrocarbon ethane, C_2H_6 . The second member of the olefin series is propylene, C_3H_6 , or $\text{CH}_2=\text{CH}-\text{CH}_3$. Among the higher members there are isomeric types. The olefins are probably the predominating unsaturated hydrocarbons in shale oils.

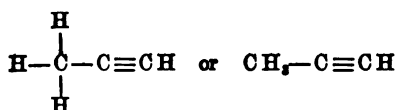
DIOLEFIN SERIES.

Another unsaturated series of hydrocarbons undoubtedly occurring in shale oil is the diolefin; the first member of this series is an un-

stable gas, propadiene, or allene, C_3H_4 , which contains 10 per cent of hydrogen and is represented by the graphic formula $CH_2=C=CH_2$, or



There is a series of compounds based on this hydrocarbon, with its more complex isomeric types, as in the other series mentioned. It may be noted that propadiene is isomeric with allylene, C_3H_4 , the second member of the acetylene series, but the latter has the following structural or graphic formula:



ACETYLENE SERIES.

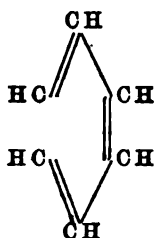
Another series of unsaturated hydrocarbons that may be present in shale oils is the acetylene, though there is much experimental evidence that acetylenes are not present in oils produced by destructive distillation. The first member of the series is the gas acetylene, C_2H_2 , which contains 7.7 per cent of hydrogen and is graphically represented:



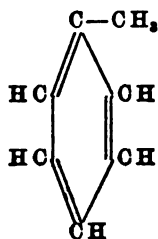
Each carbon atom thus has two unsatisfied combining powers or valencies, indicated by the triple bond. The type formula for this series is C_nH_{2n-2} , which also serves as the type formula for the diolefin series.

BENZENE SERIES.

Members of still another series of hydrocarbons—the aromatic or benzene series—probably are present in small quantities in shale oil. The aromatic hydrocarbons are really unsaturated, but often are not so considered because many of their properties are like those of the saturated compounds. The type formula is C_nH_{2n-6} . The first member of the series is the liquid benzene, C_6H_6 , graphically represented:



The next member is the liquid toluene, C_7H_8 , or,



and so on.

As this brief discussion has indicated, hydrocarbon molecules can be indefinitely complex, and unsaturated and saturated groupings may be present in the same molecule. In addition, the oxygen, nitrogen, and sulphur derivatives of hydrocarbons of all these series have to be considered in the study of shale oil.

CHANGES OF OIL-SHALE HYDROCARBONS BY DESTRUCTIVE DISTILLATION.

The oil-producing substance of oil shales, though its composition differs in different shales, has been given the name kerogen, as already noted. Kerogen is practically insoluble in the common organic solvents of bitumens, such as carbon disulphide, and is therefore classed as a pyrobitumen. Certain pyrobitumens—those classed as asphaltic pyrobitumens—on being heated change into bitumens; that is, they become largely soluble in organic solvents, particularly in carbon disulphide and carbon tetrachloride.⁵⁵ Apparently the oil-producing materials of most oil shales are correctly considered as asphaltic pyrobitumens, and thus, when an oil shale is heated, the first step in the conversion of kerogen into oil and gas is the change of the insoluble pyrobitumen—kerogen—into a bitumen largely soluble in carbon disulphide, and which, when extracted from the shale and freed from its solvent, is at ordinary temperatures a solid or semisolid, asphaltic-like substance.

How pyrobitumen changes to bitumen is not known, but it has been suggested that the change may be of the nature of a complete or partial depolymerization. It has been believed that the conversion begins at a relative high temperature (approximately 400° C.), but recent work of the Bureau of Mines shows that, for certain shales at least, the conversion is apparently progressing, although very slowly, at temperatures as low as 200° C. The rate of conversion

⁵⁵ McKee, Ralph H., and Lyder, E. E., Thermal decomposition of oil shales: Engineering and Scientific Paper No. 6, Columbia University, New York, August, 1921, pp. 6-10.

Abraham, Herbert, Asphalts and allied substances, 1920, pp. 19-27, 56-58, 149-164.

Engler, C., Die Chemie und Physik des Erdöls. Bd. 1. Leipzig, 1913, pp. 35-37.

increases with the temperature; pressure seemingly has little direct influence on it. At temperatures as high as 400° C. the bitumen forms rapidly, and decomposes almost as fast as it forms, unless the distillation system is under considerable pressure. When the bitumen decomposes it forms gas, oil vapors, and fixed carbon, the relative quantities of each apparently depending on the temperature and pressure under which decomposition takes place. If the bitumen is caused to form at temperatures not too high, and its decomposition products are removed rapidly, particularly with steam, some of the unaltered or only partly altered bitumen evidently may be carried out of the retort in the vapor stream.

Production of oil from oil shale is thus broadly divisible into two stages, (1) the conversion of kerogen into a bitumen, and (2) the decomposition of the bitumen into gas, oil vapors, and carbon. Some water may also be produced in the decomposition. Often the two steps of the process probably occur almost simultaneously. Pressure greatly influences the amount and nature of the products of the decomposition of the bitumen.

Kerogen is undoubtedly composed of large and complex molecules, and the bitumen formed from it in the first step of the conversion must likewise have large and complex molecules. In general, organic molecules are unstable under the action of heat in direct proportion to their complexity and size. A large and complex organic molecule, when heated to a temperature high enough, decomposes, breaks down, or "cracks" into smaller, simpler molecules. Thus the large and complex molecules of the first-formed bitumen, when heated in the absence of air, decompose into simpler forms. Heating an organic substance to a temperature sufficient to decompose its complex molecules into simpler molecules constitutes the process of destructive distillation, often termed pyrolysis. Essentially, the retorting of oil shale is the pyrolysis of a heavy and chemically complex bitumen.

The degree of decomposition or cracking of a molecule subjected to heat and pressure depends on the composition of the molecule, the degree and duration of the heat and pressure, the material with which the molecule is in contact, and on still other factors. The physical and chemical properties of the mineral matter of oil shales have a considerable effect on the products obtained. By prolonged heating at high temperatures it is possible to decompose a hydrocarbon molecule into its constituent elements—hydrogen and carbon—a result not desired in oil-shale distillation. At different temperatures and pressures, and with different substances in contact with the molecules undergoing distillation, products of widely differing characteristics can be obtained, and the number of such products of different nature that can be obtained from a given molecule increases with the size

and complexity of the molecule. As there are in the kerogen of oil shales substances that probably contain extremely complex molecules and as the molecular composition of the bitumen formed from the kerogen probably differs in different shales and with different conditions of formation, it is evident that many kinds of hydrocarbons and hydrocarbon derivatives can be produced from an oil shale, and that products having different properties may be made from different shales, or from the same shale when treated under different conditions.

The oils produced when oil shales are destructively distilled seem to be chiefly mixtures of members of the paraffin, olefin, diolefin, and naphthene series and their derivatives, with some of the aromatic and possibly some of the acetylene series. These oils have boiling points ranging from less than 50°C . to more than 400°C ., and specific gravities from less than 0.7 to over 1. From this mixture of organic compounds comprising shale oil, commercial products may be obtained by refining. In addition to the oils, permanent or fixed gases are produced when the shales are retorted. These gases are chiefly carbon monoxide, carbon dioxide, hydrogen, hydrogen sulphide, and the lower members of the various hydrocarbon series—those of the paraffin and olefin series predominating.

As organic molecules decompose in so many ways and yield hydrocarbons and hydrocarbon derivatives of different series under different conditions, the commercial distillation of oil shale can not be a haphazard process, but must be carefully controlled in accordance with the conditions most suitable for obtaining the desired quality of crude oil. It is possible to distill under conditions that yield no oil but only gas and carbon as final products, but it has not yet been possible to produce oil alone without gas. Because the complex molecules break down in so many different ways under the same or only slightly different conditions, and because in commercial practice mechanical and economic limitations render practically impossible the exact control of retorting conditions, it is not possible to produce from an oil shale only those hydrocarbons desired. Distillation can not be stopped just when the required decomposition has taken place. Some of the hydrocarbons in the oils produced will be those desired, but others will be objectionable and must be removed by refining. A fact to be emphasized is that the properties of the crude shale oil, and therefore of its products, are greatly influenced by the conditions under which the oil is produced. It is also evident that when an oil shale is heated light distillates are not produced at low temperatures and heavier oils at the temperature increases. A heavy bitumen is first produced, and this decomposes into a mixture of vapors of light and heavy products, from which fractions may be separated by fractional condensation, or perhaps better by bulk con-

densation and fractional distillation of the bulk condensate—the crude oil. (See p. 174.)

It is possible that no conversion of pyrobitumen into bitumen, as previously described, takes place when certain shales containing oil-producing materials classed as nonasphaltic pyrobitumens are retorted. This does not invalidate the foregoing discussion. If bitumen is not first produced, the pyrobitumen, of large and complex molecular structure, decomposes directly into gas, oil vapors, and carbon; that is, there is one less step in the process of conversion. The point emphasized here is that regardless of the exact mechanism of the conversion of kerogen into oil, this conversion is fundamentally the decomposition of cracking of heavy and complex organic molecules.

In the commercial distillation of oil shale the main object is to obtain profitably the highest possible yield of the best oil. The oils having a so-called "paraffin base," those chiefly composed of members of the paraffin series of hydrocarbons, are usually considered most suitable for the commercial production of refined oils, such as gasoline, kerosene, and the like. During destructive distillation organic molecules tend to lose hydrogen atoms. Shale oils frequently contain more unsaturated than saturated hydrocarbons and are said to be highly unsaturated. In commercial petroleum products some of the olefin and practically all of the diolefin and acetylene hydrocarbons are objectionable; some give a bad odor to the oils; also, by oxidizing or slowly combining with themselves—a process known as polymerization—they form gums and resins. In particular, the more highly unsaturated compounds must be removed by refining in order to make satisfactory commercial products. The diolefins especially seem to be responsible for gum formation. For a detailed description of the various oil and gas products from oil shale, the reader is referred to pages 76 and 85 to 89.

As the kerogen of oil shales contains oxygen, nitrogen, and sulphur in addition to carbon and hydrogen, shale oils usually contain fairly large amounts of nitrogenous compounds, such as pyrrol (C_4H_5N), pyridine (C_5H_5N), quinoline (C_8H_7N), and their derivatives; traces of compounds of the phenol type, such as phenol (C_6H_5OH); undoubtedly sulphur-containing compounds, probably of the type C_nH_mS , such as $C_{10}H_{20}S$; and probably naphthenic acids—oxygen derivatives of the naphthene hydrocarbons. The color and odor of shale-oil fractions are probably due to the presence of sulphur and nitrogen compounds and naphthenic acids. Some of these compounds, if abundant enough in the oils, might be of value as by-products, but most of them, with the possible exception of the nitrogenous compounds, seem to be present in such small proportions that they merely constitute objectionable impurities that must be removed

in refining. Even the nitrogen compounds, which may possibly be present in amount sufficient to make their recovery worth while, must be removed from the refined products, as they are particularly evil-smelling and discolor the oils badly.

In retorting oil shale, therefore, the effort should be made to produce an oil containing a minimum of objectionable hydrocarbons and other organic compounds, and a maximum of those that are satisfactory. The shale and the vapors from it must not be excessively heated in the retort or elsewhere until all of the oil has been distilled. In retorting a particular shale a temperature and a period of heating to that temperature will probably be found that will give the best results. Other factors, such as the size of the pieces of shale, the thickness of the charge, the method of heat transfer, the duration of contact of the vapors with the heated zone, and the presence of different gases or vapors in the retort, have a bearing on the quality and quantity of the products. Usually, of course, the theoretically best conditions must be somewhat modified to meet commercial considerations of feasibility and cost, and will vary according to the product chiefly desired. A fact to be emphasized is that different shales probably will require different distillation conditions for best results, and the best conditions may have to be determined for each different type of shale, and changed with changes in the market for finished products. (See also p. 119.)

USE OF STEAM IN DESTRUCTIVE DISTILLATION.

In Scottish practice steam is let into the vertical retort at the bottom, where it comes in contact with the residual or spent shale that has been slowly heating after all the oil has been distilled from it in the upper part of the retort. When the shale reaches the bottom of the retort its temperature is such that the steam reacts with its fixed carbon, probably in accordance with the reaction:



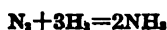
This is probably the main reaction, but the temperature is high enough to cause some production of water gas, a mixture of hydrogen and carbon monoxide, according to the equation:



It has long been known that much of the nitrogen of the shale remains in the spent shale as long as carbon is present and is freed from the shale as the carbon is removed. A large part of the nitrogen seems to be fixed in relatively stable compounds and can be

¹⁰⁰ This reaction may be theoretical; probably the carbon monoxide forms by the action of carbon dioxide on carbon: $CO_2 + C = 2CO$.

recovered only by burning these compounds or subjecting them to chemical action. Thus when the nitrogen-containing compound is broken down with steam at high temperatures, amino (NH_2), or imino (NH) groups, or the like, may be reduced by the hydrogen from the steam to form ammonia (NH_3). When carbon reacts with the steam, nitrogen may possibly be set free to combine at once with free hydrogen, as follows:



but it is not likely that this reaction takes place appreciably, if at all. As soon as the ammonia forms it must be removed to a zone of lower temperature, for it rapidly decomposes if it remains for any considerable time in the highly heated lower part of the retort.

In the Scottish industry the production of ammonium compounds is extremely important, and although some ammonia may be produced from oil shale without the use of steam, the maximum yield of ammonia is obtained by using steam in the retorts and heating the shales to high temperatures after the oil has been driven off. The use of steam also gives a greater yield of combustible gases, makes heat transfer more efficient, cools the spent shale, and in certain retorts may improve the quality of the oil, as pointed out in detail in later sections (pages 176 to 178). In general, American shales contain nitrogen, and the use of steam in retorting will probably depend largely on whether the nitrogen is to be recovered as ammonia and whether in commercial practice American shales will yield satisfactory oils without the use of steam.

A satisfactory method of heating an oil shale is the prime essential in obtaining a satisfactory production of oil, and one important feature of the use of steam in retorting is the manner in which steam distributes heat throughout the charge in the retort and recovers it from the spent shale. If ammonium compounds are desired from American shales, the retorts will have to be steamed, but in the opinion of the writer even though ammonia production be neglected the use of steam in retorting has so many favorable aspects that it deserves careful consideration.

HISTORY OF THE OIL-SHALE INDUSTRY.

BEGINNING OF THE INDUSTRY.

The production of tars by the dry distillation of wood was known at least as early as the latter part of the seventeenth century.⁵⁷ About this time (1681) coal was distilled in England for the production of oil and tar, and later coke. In 1694 Eele, Hancock, and Portlock

⁵⁷ Schettbauer, W., *Shale oils and tars*. London, 1912, p. 1.

distilled "oyle from a kind of stone,"⁵⁸ possibly oil shale, and bituminous or oil shales were distilled as early as 1761,⁵⁹ for the production of oils to be used for medicinal purposes. The possibilities of producing paraffin and petroleumlike oils from oil shales seem not to have been recognized until 1830.

THE OIL-SHALE INDUSTRY IN FRANCE.

In 1830 Laurent⁶⁰ obtained paraffin by distilling oil shale, but the French oil-shale industry really began in 1838 with the perfection of a process developed by Selligie.⁶¹ In 1839, at the Paris Industrial Exposition, Selligie exhibited samples of shale-oil products, light oils, burning oils, heavy oils, and paraffin. The early patents of Selligie and DeBuisson (English patent 10726, 1845) describe distilling apparatus, the use of steam in distilling, and the products and methods of refining.

The French industry grew in importance until 1864, when it was checked and set back by imports of cheap American oils, and it has never fully recovered. It received another blow in 1898, when the French Government cut in half the import duty on petroleum. Until 1903 the retorts used in France were heated to comparatively low temperatures and steam was not used in them. The introduction of the Scottish type of retort increased the yield of oils from the shales more than 10 per cent, and in recent years the industry has given indications of recovering some of its former importance, its development having been stimulated by the high price of imported petroleum oils during the World War and the years 1919 and 1920. In 1920 there were reports that new work on a large scale was about to be undertaken, particularly in the Boson fields in the Department of the Var. The present high import duty on petroleum in France is also encouraging the industry.

It is interesting to note that up to 1893 the Scottish type of oil-shale retort was not used in France.⁶² By 1905, however, practically all the retorts used in that country were of the Scottish type, and recent advice from Scotland affirms that at present the Scottish retorts are used to the practical exclusion of others. In the year 1904 the production of oil shale in France was 184,030 tons, and the average yield of oil slightly over 20 United States gallons per ton, for a total of 3,693,783 gallons.⁶³

⁵⁸ English patent 330, 1694.

⁵⁹ Bacon, R. F., and Hamor, W. A., *The American petroleum industry*. New York, vol. 2, 1916, p. 808.

⁶⁰ Laurent, Aug., *De l'huile des schistes bitumineux et de quelques produits qu'on en obtient*: *Compt. rend.*, t. 4, 1837, pp. 908-912.

⁶¹ Selligie, A. F., French patent 9467, Nov. 14, 1838; addition, Mar. 27, 1839.

⁶² Chesneau, G., *L'industrie des huiles des schistes en France et en Ecosse*: *Ann. mines*, 9th ser., t. 3, 1893, pp. 617-674.

⁶³ Aron, A., *L'industrie française des schistes bitumineux*: *Ann. mines*, 10th ser., t. 9, 1906, pp. 47-75.

HISTORY OF THE OIL-SHALE INDUSTRY IN AUSTRALIA.*

At the beginning of the industry in 1865-66 shale was retorted at Kembla Valley and at Hartley Vale. In 1878 retorting began at the Joadja deposit, near Mittagong. For a number of years the industry was successful at Hartley and Joadja, but gradually the exportation of high-grade shale for gas enrichment took precedence over retorting. Finally the Joadja works closed and the Hartley works restricted operations to refining crude oil from retorts erected near Capertee by the same company. Paraffin was recovered and the oil was used for gas enrichment in the large cities of the State.

In 1908 the Commonwealth Oil Corporation took over the Hartley and Capertee works and built a new plant at Newnes, which started operations in 1911. Operations ceased about a year later, as the original large capital was exhausted before satisfactory results were obtained. The Scottish retorts installed proved unsuitable for the higher-grade shales of the Wolgan-Capertee seam, which yield 64 to 128 gallons of oil to the ton, compared to about 25 gallons from Scottish shales. Intumescence of the shale fractured the retorts and prevented the gradual descent of the spent material. Explosions were frequent. In June, 1914, this company was reorganized as John Fell & Co. New types of retorts were built on the site of the old ones, and these have successfully overcome the difficulties previously encountered.

METHODS OF TREATING THE SHALES AND PRODUCTS RECOVERED.

Retorting and refining methods for oil shales and shale oils in Australia have been fairly similar to those in Scotland. Both vertical and horizontal retorts have been used—the verticals until about 1910 or later being somewhat like the Young and Beilby type. (See p. 66.) A bench of modern Scottish-type retorts erected more recently was not wholly successful, as noted above.

The yield of the richest Hartley shale in horizontal retorts without steam was 130 to 170 gallons of oil to the ton.** The oil was light colored, had a pleasant smell, and could be refined easily. Its specific gravity was 0.855 and its setting point 58° F. The richest Capertee shale, treated in the same manner, yielded 96 to 102 gallons of oil to the ton. This oil was considered somewhat inferior to that from the

* Data for this section were obtained from Mr. Douglas A. Fell, of Sydney, Australia, and the following works:

Steuart, D. R., *The oil shales of the Lothians. Part III, The chemistry of the oil shales: Memoir Geol. Survey Scotland*, 2d ed., 1912, p. 163.

Carne, J. E., *The kerosene shale deposits of New South Wales: Memoir Geol. Survey New South Wales*, 1903, 333 pp.

Handbook of the mineral products of New South Wales: Dept. Mines, New South Wales, 1920, pp. 13-16.

** Steuart, D. R., work cited, p. 163.

Hartley shale; its specific gravity was 0.866 and setting point 58° F. The horizontal retorts, made of boiler plate, were 12 feet long, 6 feet wide, and 2 feet 2 inches high. Three days were required to distill a charge. Hartley shale if withdrawn from the retort and cooled when half distilled resembles stiff rubber in appearance and elasticity.

These rich shales were used for gas making; the poorer shales above and below the rich seam were retorted, yielding crude oils, as indicated in Table 8.⁶⁶

TABLE 8.—Yield of products from Australian shales.

[Quantities calculated to tons of 2,000 pounds and to United States gallons.]

| | Hartley shale. | | Capertee shale. | |
|---|------------------|--------------------------|------------------|--------------------------|
| Oil (gallons per ton)..... | 48.2 | | 51.5 | |
| Specific gravity..... | 0.890 | | 0.900-0.905 | |
| Setting point, °F..... | 70 | | 72 | |
| Refined products from crude: | <i>Per cent.</i> | <i>Specific gravity.</i> | <i>Per cent.</i> | <i>Specific gravity.</i> |
| Spirit..... | 4.02 | 0.720-0.780 | 1.2 | 0.730-0.780 |
| Burning oil..... | 10.2 | .800 | | |
| Gas oil..... | 16.98 | .850-.860 | 58.0 | .850 |
| Blue oil..... | 40.77 | .900 | 16.4 | .925 |
| | | <i>Melting point.</i> | | <i>Melting point.</i> |
| Paraffin scale..... | 4.41 | | 4.1 | 122-3° F. |
| Coke..... | 3.59 | | | |
| Loss in refining..... | 20.03 | | 20.3 | |
| Ammonium sulphate (pounds per ton)..... | 2.9 | | 9.4 | |

At Capertee the splint was kept apart from the retorting shale, as the distillate produced from splint is a sticky resinous substance that clogs the condensers and gives the wax a persistent yellow color, making it impossible to refine to whiteness.

The Commonwealth Oil Corporation's plant consisted of a bench of 64 Pumphreston retorts. Each retort had a capacity of 2 to 2½ tons of shale a day. This bears out the belief that the capacity of a Scottish retort must be rated in terms of oil production rather than shale throughput (p. 72).

In 1921 the plant operated by John Fell & Co. consisted of 32 Fell patent retorts. These are vertical and of original design. Each retort has a daily capacity of 6 to 7 tons of 80 imperial-gallon shale. A gyratory crusher breaks the shale for the retort into pieces 5 by 2 inches or smaller. The operation of the retorts differs little from that of present-day plants in Scotland. Enough permanent gas is formed to heat the retorts and make the use of gas producers unnecessary. A continuous oil-distillation battery of stills is used for the crude—three continuous horizontal boiler-type stills working in

⁶⁶ Table derived from Steuart, D. R., work cited, p. 163.

series and discharging from the last into a coking still. Acid is recovered from the refinery tars and used in making ammonium sulphate. Distillations and treatments generally correspond with Scottish refining practice.

For the year ending September 30, 1920, this plant retorted 16,090 long tons of shale, the average production of oil per long ton being 79.7 imperial gallons. The crude oil yielded finished products about as follows:

| <i>Finished products from crude oil.</i> | |
|--|-----------|
| | Per cent. |
| Benzine and spirits----- | 5 |
| Kerosene----- | 20 |
| Gas oil----- | 25 |
| Fuel oil----- | 20 |
| Wax----- | 5 |
| Refining loss and coke----- | 25 |
| | <hr/> 100 |

The yield of ammonium sulphate is about 19.5 pounds per ton. The oil coke produced is sold for household use.

PRESENT STATUS OF THE OIL-SHALE INDUSTRY IN AUSTRALIA.

Financially, oil-shale operations in Australia have not been particularly successful so far, largely because of extremely high labor costs and peculiar labor conditions. In 1917 the Federal Government of Australia provided for a royalty to be paid for the production of mineral oil and this acted as a stimulant to the industry. In 1919 the royalty paid by the Australian Commonwealth on crude shale oil produced for the year ending June 30, amounted to £26,406. It was earned by one firm in New South Wales.⁶⁷

In 1921 experiments were undertaken in an attempt to produce oil from the oil shale in place. Part of a deposit was ignited and by control of the rate of combustion the heat generated was expected to produce oil from the shale. The vapors were led to condensers on the surface through suitable openings. The preliminary experimental work promised favorable commercial results, but evidently work on a large scale was disappointing, for recent reports indicate that the new method has been abandoned.

Reports reaching the writer early in 1923 indicate that the only company hitherto producing commercial quantities of shale oil in Australia has decided to stop shale operations but will use the refineries for treating imported petroleum. The high cost of labor and the low price of imported oil are given as the main reasons why the rich shales of Australia can not profitably be worked at present.

⁶⁷ Petroleum World: The shale-oil bounty, Vol. 16, November, 1919, p. 458.

Table 9 shows the production of oil shale in New South Wales from 1865 to 1921:

TABLE 9.—*Production of oil shale in New South Wales from 1865 to 1921.*^a

| Year. | Quantity
(long tons). | Year. | Quantity
(long tons). | Year. | Quantity
(long tons). |
|-----------|--------------------------|-----------|--------------------------|-------------------------|--------------------------|
| 1865..... | 570 | 1884..... | 31,618 | 1903..... | 34,776 |
| 1866..... | 2,770 | 1885..... | 27,462 | 1904..... | 37,871 |
| 1867..... | 4,079 | 1886..... | 43,563 | 1905..... | 38,226 |
| 1868..... | 16,952 | 1887..... | 40,010 | 1906..... | 32,446 |
| 1869..... | 7,500 | 1888..... | 34,869 | 1907..... | 47,331 |
| 1870..... | 8,580 | 1889..... | 40,561 | 1908..... | 46,303 |
| 1871..... | 14,700 | 1890..... | 56,010 | 1909..... | 48,718 |
| 1872..... | 11,040 | 1891..... | 40,349 | 1910 ^b | 68,239 |
| 1873..... | 17,850 | 1892..... | 74,179 | 1911..... | 75,104 |
| 1874..... | 12,100 | 1893..... | 55,660 | 1912..... | 86,018 |
| 1875..... | 6,197 | 1894..... | 21,171 | 1913..... | 16,983 |
| 1876..... | 15,998 | 1895..... | 59,426 | 1914..... | 50,049 |
| 1877..... | 18,963 | 1896..... | 31,839 | 1915..... | 15,474 |
| 1878..... | 23,371 | 1897..... | 34,090 | 1916..... | 17,425 |
| 1879..... | 32,519 | 1898..... | 29,689 | 1917..... | 31,661 |
| 1880..... | 19,201 | 1899..... | 39,719 | 1918..... | 32,395 |
| 1881..... | 27,894 | 1900..... | 22,862 | 1919 ^c | 25,453 |
| 1882..... | 48,065 | 1901..... | 54,774 | 1920..... | 421,004 |
| 1883..... | 49,250 | 1902..... | 62,880 | 1921..... | 32,489 |

^a Carne, J. E., *Kerosene shale deposits of New South Wales: Memoir Geol. Survey New South Wales*, 1903, p. 286.

^b Figures for years 1910 to 1918, both inclusive, from *Handbook of the mineral products of New South Wales: New South Wales Dept. of Mines*, 1920, p. 14.

^c Figures for 1919 calculated from data furnished by R. M. Cambage.

^d *Petroleum Times*, May 20, 1922, p. 638.

HISTORY OF THE OIL-SHALE INDUSTRY IN SCOTLAND.

The oil-shale industry in Scotland apparently originated independently of that in France. In 1850^a James Young, who with a partner had been refining the oil obtained from a small seep in Derbyshire, England, conceived the idea that oil was a product of coal and had been distilled from the latter by the action of subterranean heat. He began experimenting with the production of oil from coals, and finally produced from cannel coal an oil containing solid paraffin. In the course of his experiments a highly bituminous material, called "Boghead coal" or "Torbanehill material," or simply torbanite, was discovered, which yielded upward of 135 gallons of oil to the ton. In 1850 a plant was erected by Young and two associates at Bathgate, Linlithgowshire, for the production of oil from Boghead coal, and was producing crude oil in 1851. Young's original patent^a covering the process of the production of oil from bituminous mineral substances was the subject of much controversy and litigation, caused mostly by arguments as to whether Boghead coal was a coal or a shale. Finally it was decided that whatever the material actually was, the lessors had acquiesced in its being treated as within the terms of the lease, and that legally it was coal;

^a Data on the early history of the Scottish oil-shale industry have been taken largely from Redwood, I. I., *Mineral oils and their by-products*, London, 1897, p. 33 et seq.; on the late history from personal communications and interviews with Scottish operators.

^b English patent 18292, Oct. 17, 1850.

the decision was purely on a point of law and scientific evidence was practically disregarded.

In 1862 the deposits of Boghead coal became exhausted, and since then the oils obtained in Scotland have been produced almost entirely from the oil shales of the "lower Carboniferous" series of Midlothian and Linlithgow. These shales were first worked in 1862 by Robert Bell at Broxburn.

The industry developed rapidly in Scotland, though many of the companies organized to treat oil shales failed financially or, in order to attain success, combined with others.

Since 1851 over 140 different individuals or companies have entered the oil-shale industry in Scotland. In 1871 there were 51 active companies, but of these the number in existence in 1910 was only 6. In 1919 the 4 largest companies—Pumpherstons Oil Co. (Ltd.), Broxburn Oil Co. (Ltd.), Young's Paraffin Light & Mineral Oil Co. (Ltd.), and the Oakbank Oil Co. (Ltd.)—with a smaller company, Jas. Ross & Co., Philipstown Oil Works (Ltd.), which produced crude oil only—consolidated under one company, Scottish Oils (Ltd.). Thus practically there is but one oil-shale company in Scotland at present working, although the individual companies still maintain their identity.

The years following 1864 virtually paralyzed the oil-shale industry in Scotland. That year marked the beginning of the import of cheap petroleum oils from America and witnessed the failure of a great number of Scottish oil-shale companies and the consolidation of many of the remainder. Such consolidations, the development of more efficient and economical apparatus and methods for retorting the shales and refining the oils, and the strictest economies in mining, retorting, refining, and marketing are undoubtedly the reasons why the industry has survived in the face of the competition caused by the ever-growing petroleum industry.

The growth of the oil-shale industry in Scotland can be seen at a glance in Table 10, which shows the amount of shale mined in Scotland by years up to and including 1920. The yield of oil per ton of shale retorted has steadily decreased, and although complete official statistics for this in recent years are unavailable, Table 11 gives some data for earlier years that are of interest. The explanation for this decrease of oil yield per unit of shale mined is the fact that though the richer shales have been pretty well worked out, improvements in mining methods and the technique of surface operations—that is, handling, retorting, and refining—have made the working of the thicker, more easily mined, but leaner shales, economically possible. Thus the history of the Scottish oil-shale industry is the story of the development of the present apparatus and

methods for retorting the shale and, to a lesser extent, for refining the products.

TABLE 10.—*Production of oil shale in Scotland from 1873 to 1922.**

| Year. | Quantity
(long tons). | Year. | Quantity
(long tons). | Year. | Quantity
(long tons). |
|-----------|--------------------------|-----------|--------------------------|-----------|--------------------------|
| 1873..... | 524, 095 | 1890..... | 2, 212, 250 | 1907..... | 2, 680, 028 |
| 1874..... | 262, 747 | 1891..... | 2, 361, 119 | 1908..... | 2, 882, 039 |
| 1875..... | 437, 774 | 1892..... | 2, 089, 037 | 1909..... | 2, 967, 057 |
| 1876..... | 603, 538 | 1893..... | 1, 956, 520 | 1910..... | 3, 130, 280 |
| 1877..... | 801, 701 | 1894..... | 1, 986, 385 | 1911..... | 3, 116, 803 |
| 1878..... | 788, 704 | 1895..... | 2, 246, 865 | 1912..... | 3, 184, 826 |
| 1879..... | 783, 748 | 1896..... | 2, 419, 525 | 1913..... | 3, 280, 143 |
| 1880..... | 837, 805 | 1897..... | 2, 223, 745 | 1914..... | 3, 268, 086 |
| 1881..... | 958, 255 | 1898..... | 2, 137, 093 | 1915..... | 2, 998, 652 |
| 1882..... | 1, 030, 915 | 1899..... | 2, 210, 824 | 1916..... | 3, 009, 232 |
| 1883..... | 1, 167, 943 | 1900..... | 2, 282, 221 | 1917..... | 3, 117, 658 |
| 1884..... | 1, 518, 871 | 1901..... | 2, 354, 356 | 1918..... | 3, 080, 867 |
| 1885..... | 1, 770, 413 | 1902..... | 2, 107, 534 | 1919..... | 2, 758, 555 |
| 1886..... | 1, 728, 503 | 1903..... | 2, 009, 602 | 1920..... | 2, 827, 792 |
| 1887..... | 1, 411, 378 | 1904..... | 2, 333, 062 | 1921..... | 2, 867, 000 |
| 1888..... | 2, 076, 469 | 1905..... | 2, 496, 785 | 1922..... | 2, 604, 027 |
| 1889..... | 2, 014, 860 | 1906..... | 2, 546, 522 | | |

* Figures from 1873 to 1910 taken from paper by D. R. Stuart, *The oil shales of the Lothians. Part III, The chemistry of the oil shales: Memoir Geol. Survey Scotland*, 2d ed., 1912, p. 139. Figures from 1910 to 1922 from H. R. J. Conacher, personal communication; 1920 figures from unpublished Consular Report, † 1921 production was low on account of miners' strike.

TABLE 11.—*Production of oil shale and oil and ammonium sulphate therefrom in Scotland for various years.^a*

| Year. | 1886 | 1890 | 1900 | 1910 | 1918 |
|--|-------------|-------------|-------------|-------------|-------------|
| Shale produced, long tons..... | 1, 728, 503 | 2, 212, 250 | 2, 282, 221 | 3, 130, 280 | 3, 080, 867 |
| Ammonium sulphate produced, long tons.. | 18, 080 | 24, 730 | 37, 267 | 69, 113 | 68, 311 |
| Yield of ammonium sulphate per long ton
of shale, pounds..... | 23. 4 | 25. 0 | 36. 6 | 42. 2 | 42. 4 |
| Yield of crude oil per long ton of shale
treated, imperial gallons ^b | 31. 2 | 34. 7 | 58. 4 | 25. 0 | 23. 0 |

^a Based on data in Home Office Report: *Mines and quarries. Part III, London, 1910, p. 276; H. M. Alkali Inspector's annual reports, London; and from personal communications from H. R. J. Conacher.*

^b These figures are approximations, for the years 1871, 1879, 1887, 1893 and 1918, since no official data are available regarding the production of crude oil.

In the early years good profits were made by some of the companies, and there were recurring periods of prosperity. Generally, however, the financial status of the companies did not become satisfactory until about 1900. During the World War and in 1919 operating costs became higher, labor difficulties increased, and the production of oil from the shales now being worked became so low that the refineries could not operate to capacity. Under these circumstances the severe fall in prices in 1919 brought about the reorganization of the industry, as mentioned, and the formation of Scottish Oils (Ltd.), a subsidiary of the Anglo-Persian Oil Co. (Ltd.). It is reported that the unutilized capacity of the refineries will treat petroleum imported by the Anglo-Persian Oil Co. from the Persian fields.

During a large part of 1921 retorting operations in Scotland were greatly restricted because of the lack of coal supplies following the

coal miners' strike, and the resulting adverse economic conditions. The relatively small output of oil shale mined in that year, as shown in Table 10, was due to this shut-down. When operations were resumed there was a reduction in retort capacity caused by the scrapping of obsolete retorts which had been kept going during the World War; also other retorts had been damaged by repeated cooling. This reduction in capacity, however, has been largely made up by the improved efficiency of the remaining retorts. It is reported that the existing Scottish refineries are now in full operation; one of them, representing the saving in buildings and equipment through the concentration of operations, is now employed on imported petroleum. The centralization of management carried out in 1919 has resulted in substantial economy and improved efficiency. It is probable that the industry would have been unable to continue operations profitably under present conditions without this reorganization.

THE SCOTTISH OIL-SHALE INDUSTRY.¹⁰

The oil-shale industry has been developed more in Scotland than in any other part of the world. The capital invested in leases, mines, works, refineries, and the like in Scotland exceeds \$12,500,000. In the opinion of the writer, those who are engaged or who contemplate investing or becoming engaged in the oil-shale industry in the United States will make no mistake by studying in detail the technique of the Scottish industry. That industry has been in existence since 1850, and although its technique may not be wholly applicable to American shales and conditions, the experience of Scottish operators and their methods of solving difficulties in commercial operations can be of great value to the future oil-shale industry of the United States.

SAMPLING AND DETERMINING EXTENT OF OIL-SHALE DEPOSITS.

When a new deposit or seam of oil shale is to be worked, careful exploration and sampling are undertaken. The dip of the stratum at the surface is determined, although this may be of little value, as the shale beds in Scotland usually are badly folded and faulted. The next step is careful and extensive sampling of the seam, or seams, by core drilling to ascertain the position of the seam, its extent, thickness, and often evidence of folding and faulting. For this work, diamond core drills are now used.

If the evidence obtained with the core drill is satisfactory, a trial shaft is sunk to the shale bed, in order to obtain accurate informa-

¹⁰ This section, which deals with the technical development and present status of the Scottish oil-shale industry, has been reviewed, corrected, and revised by Messrs. James Bryson, H. R. J. Conacher, and A. C. Thomson, all officials of Scottish Oils (Ltd.), Glasgow, Scotland.

tion on its thickness and position and to get 50 to 100 tons of the shale for a large-scale retorting test.⁷¹ If the evidence is favorable, the bed is opened in the regular manner.

Companies starting oil-shale operations in the United States should thoroughly sample and measure their deposits with core drills or otherwise, in order to determine the amount of shale available, its position, and its richness, before installation of expensive plant is attempted. A little time and money spent in this kind of investigation may prevent much money being wasted in plant development not based on accurate information as to the amount and value of the shale in the deposit.

MINING.⁷²

In Scotland oil shales are almost invariably mined underground. Quarrying or open-pit operations are now rarely possible. The shale beds frequently dip at rather steep angles, and, as a rule, several seams overlie one another and in many places are worked from one main entry. An entry or "mine" is sunk from the surface to the seam or seams, often as an incline along one of the seams from the outcrop. From this entry horizontal crosscuts are run to the other seams, each being worked independently, and in turn the crosscuts serve as means of communication and transit. In other mines the shaft may be vertical and several overlying seams worked in turn from it. With modern equipment it is considered cheaper to work a shaft than an incline.

In most respects the mining of oil shale is similar to the mining of bituminous coal. Usually the roof has to be supported with timber props, and frequently the walls of the main galleries and the entries are bricked. Structural steel and reinforced concrete beams are coming into general use for permanent main gallery roof supports. Provision is made for drainage, ventilation, and haulage to main galleries. By legal regulation, all shale mines must have two independent entries. Coal-mining regulations are applied to shale workings, and are rigidly enforced.

Shale is usually mined by the "pillar and stall" or "stoop and room" method, although the "longwall" system has been used to some extent. In the former method the galleries are first driven to the limits intended, and then the pillars, which have been left between the galleries and are much larger than the latter, are removed, retreating toward the incline or shaft. In many mines the roof is supported by timbers as pillars are removed; later the tim-

⁷¹ Caldwell, W., *The oil shales of the Lothians. Part II, Methods of working the oil shales: Memoir Geol. Survey Scotland*, 2d ed., 1912, p. 97.

⁷² Much of the following description is based on the above paper, pp. 95-135.

bers are removed and used again. The roof, as a rule, sinks in gradually behind the maximum removal of shale, or after the removal of the supporting timbers, if these are used, and this movement is transmitted ultimately to the surface of the ground. Where several overlying seams are mined from one main entry, the upper or No. 1 seam is sometimes first worked out, then No. 2, and so on; but frequently the reverse order is followed. An average of 25 per cent of the shale is left in the mine as small pillars or as finely divided refuse and shale of poor quality.

Shale produced at the working face is loaded into cars holding from 1,100 to 1,600 pounds. In horizontal entries the cars are moved to the main entry by manual labor or by horses; in inclined workings various gravity systems are used, the loaded car pulling an empty car to the face. Where the distance is great, electric haulage is used. Where the main entry is inclined the haulage is usually by endless, overrunning steel cables, driven by electric motors on the surface. In shafts a skip is used. Mine cars are run on the skip, hoisted, emptied in the building above the shaft, and returned to the gallery on the skip.

Electric power and lighting are used generally in the mines except at the most remote places. Current is generated by a power plant at the shale-retorting works. The exhaust steam from this power plant is used in the retorting process.

Drilling is usually done with hand ratchet drills, similar to ships' augers, but in recent years electric rotary auger drills have replaced the hand machines in some mines. Holes are drilled at a fair rate by either method. Gunpowder in paper cartridges is the explosive generally used. It is important that the proportion of fines produced be as small as possible, as fines, unless well distributed, tend to pack in the retort and hinder the passage of gases. On an average, about 0.72 pound of powder is required per ton on shale mined.

A man working at the face produces about $4\frac{1}{2}$ tons of shale in an eight-hour day. By "man working at the face" is meant a miner and his helpers. Every miner has one helper and some miners have more.

COST OF MINING.

Present costs in the Scottish oil-shale industry are lower than they were during the World War and in the two years following it, but they are yet much higher than before it. Adjustment is still going on, but is complicated by serious labor difficulties. For these reasons, and because costs in Scotland can not be expected to apply to American oil-shale operations and conditions, actual figures for that country are not given in this report. However, in 1919, mining opera-

tions accounted for 53 per cent of the total costs of producing marketable products from the shale.

Approximately, the cost of mining is similar to the average cost of mining coal in the British Isles. If anything, it is a little lower, largely because of the average greater thickness of the shale seams now being mined.

In 1919 the Scottish shale miner was paid on either a minimum rate basis or on a straight tonnage basis. The rates per ton, based on the character and thickness of the seam, etc., varied widely. Frequently the miner must do work that is essential to the development, drainage, or ventilation of the mine, but which yields only a small tonnage of shale.

In 1919 the miners paid their helpers, who were also paid a war-time bonus by the companies. The miner furnished his own powder, which was sold to him at cost by the companies. He also supplied his own lamp and oil; also his drills, and paid for dressing them. Coal proportioned to the power in the mine amounted to about 50 pounds per ton of shale. During and since the war the cost of timber has been a large item.

MINE VENTILATION AND GASES.

The mining laws of Great Britain compelled, forced, or induced ventilation of shale as well as coal mines. Dangerous quantities of harmful gases are not commonly encountered in the shale workings in Scotland, but often enough is present to be hazardous if proper ventilation is not effected. Both methane (marsh gas or fire damp) and carbon dioxide (choke damp or black damp) are met in the workings. In many crosscuts methane escapes from fissures in the working face, especially when a crosscut is approaching the shale bed. Usually there is not enough gas in the workings to cause the miners to use closed lights and naked flames are almost invariably used in shale mining, but occasionally, especially in raises, closed lights must be used.⁷³

According to Gray⁷⁴ the percentage of carbon dioxide and methane in air at the working face ordinarily ranges from 0.29 to 0.09 and 0.02 to 0.06 per cent, respectively. Samples taken in the return airway of a shale mine in Scotland showed carbon dioxide 0.32 to 0.05 and methane 0.09 per cent to none.

Ventilation is effectively provided for in the Scottish shale workings.⁷⁵ As each mine has two independent openings, one serves as an air inlet and the other as an outlet. Ventilating fans are of the blower or the exhaust type, depending on whether they are placed at

⁷³ Caldwell, W., *The oil-shales of the Lothians. Part II, Methods of working the oil shales*: *Memoir Geol. Survey Scotland*, 2d ed., 1912, p. 119.

⁷⁴ Caldwell, W., work cited, p. 120.

⁷⁵ Caldwell, W., work cited, pp. 122-126.

the inlet or outlet shaft. By doors, stoppings, or cloth brattices the air is given a definite circulation throughout the workings; the brattices are chemically treated to make them fireproof. In steep workings the general circulation sometimes is temporarily insufficient just after blasting, to provide satisfactory ventilation in a raise; then a hand blower, usually operated by a boy, often makes up the deficiency. The blower is set a suitable distance from the working face, to which it supplies air through a wooden or iron pipe.

On account of the more simple system of the workings, the ventilation of longwall workings is cheaper than that of stoop-and-room workings.

WATER IN MINE WORKINGS.

As the shale beds are impervious only small quantities of water are encountered.¹⁶ It is usually handled by pumps, driven by electric motors.

EXPLOSIBILITY OF SHALE DUST.

According to McLaren and Clark,¹⁷ the dust of Scottish oil shale in the mines tends to retard explosions rather than to propagate them. Careful experiments were made with dust collected from the Broxburn and Pumpherston mines to determine if it were inflammable in the presence of a methane gas explosion. These tests indicated conclusively that shale dust was nonexplosive and actually had a tendency to retard a gas explosion. However, it will not do to take for granted, in view of the nonexplosiveness of Scottish shale dust, that dust in shale mines in the United States will also be nonexplosive nor that dangerous quantities of explosive gases will not be encountered. Indeed, the writer has witnessed experiments that lead him to doubt the nonexplosibility of some shale dusts. Until it is known positively that such precautions are not necessary the writer recommends that shale mining be conducted with the same care to avoid explosions that is taken in coal mining. (See p. 113.)

CRUSHING.

Cars carrying the shale are examined at the mine mouth by an inspector, who rejects unsuitable shale; then the cars are weighed and the weight credited to the miner who has filled them. Each car has attached to it a leather or metal tag, serving as an identification mark for the miner. After being weighed, the cars are dumped directly into the crusher, or are hauled to it by endless cable if the retorts are not close to the mine. Automatic tippers are usually em-

¹⁶ Caldwell, W., work cited, p. 127.

¹⁷ Caldwell, W., work cited, p. 120.

ployed for dumping the mine cars into bins above the breakers, but if the shale is carried in standard-gauge railway cars a hydraulic ram is used.

The crushers consist of horizontal toothed rolls that revolve slowly toward each other. They are of heavy cast iron, made up of hollow cylinders about 2 feet long and 3 feet in diameter, the total length of each roll being 9 feet. The clearance between the ends of the teeth is about 1 inch. The teeth are about 2 inches long when new, 1 inch square at the base, and set 4 or 5 inches apart each way. The maximum size of shale passing the rolls is about 5 by 5 by 12 inches, and the average of the larger pieces is about the size of an ordinary brick, but thinner. The capacity of a pair of rolls is between 480 and 780 short tons a day. Fines usually are not screened out after the shale has passed the rolls, because if they are properly mixed they can be satisfactorily treated in the retorts. The rolls rotate slowly and are belt or gear driven by steam engines or electric motors. It should be noted that the shale is both broken and crushed by the rolls. From the breakers the shale is hauled in cars by endless cable or chain up an incline to the top of the retort benches and there discharged by hand into the retort hoppers.

SCOTTISH OIL-SHALE RETORTS.

DEVELOPMENT OF THE SCOTTISH RETORTS.

As has been noted, one of the dominating factors in the success of the Scottish oil-shale industry has been the development of economical and efficient retorts for the production of oil. A statement regarding the course of this development seems worth while because the present status of oil shale in the United States bears a striking resemblance to the early days of the industry in Scotland, when many retorts were tried, and abandoned, or improved, until the perfection of the present types in the years 1895 to 1902. The reader should note that the trend in development has been to devise apparatus and methods to produce, as cheaply as possible, the highest yields of the products for which there was the greatest demand or which could be most readily marketed.

Up to the year 1865 no one seems to have suspected that ammonia could be recovered from oil shales, consequently up to that time all efforts were bent on producing higher yields of the best oils from the shales. The first retorts constructed by Young⁷⁸ were ordinary D-shaped coal-gas retorts. They were charged and heated, and after the oil had been driven off were discharged and recharged. These horizontal retorts were of many different sizes and shapes,

⁷⁸ English patent 13292 (1850).

all designed to utilize the heat efficiently. Their charging capacity ranged from about 550 to 900 pounds, although a few had a capacity of over 2½ tons.

It was soon learned that control of the heat in such intermittently operated retorts was difficult and one inventor devised a horizontal retort, heated to a high temperature at one end only. The shale was continually charged at one end from a hopper, conveyed through the retort by a helical screw, and when spent was discharged from the other end into a sealed water lute. This type of retort, although adopted by Young's company, proved quite unsatisfactory and was largely replaced by vertical retorts made of cast-iron pipe 10 or 11 feet high, oval or oblong in cross section, and tapering downward. Top dimensions were 12 by 24 inches. Inasmuch as the rich shale then being treated expanded somewhat during distillation, and also intumesced or fused together considerably, this retort frequently clogged, and its manipulation was difficult.

Young and Brash then devised⁷⁹ tapered cast-iron retorts of circular cross section, set with the larger end down and the discharge water sealed. In the retort a vertical helical screw, gear driven from the top, was kept slowly revolving while the shale was going through. This apparatus largely overcame difficulties due to the expansion and sintering of the shale, and was successfully used for about eight years.

About this time Young's company introduced a vertical retort said to have been suggested by Doctor Price.⁸⁰ This retort had an elliptical cross section and a taper much greater than that before used. The large part was placed downward, and the discharge passed, as usual, into a water seal. The retort was charged from a hopper, and about 16 hours after it had been first charged and heated a small quantity of spent shale was raked out of the water seal and an equal amount of fresh shale was charged from the hopper. The hopper was recharged every 3 hours. Six or eight of these retorts were set in a common furnace, and it is to be noted that the vertical screw conveyers were no longer necessary.

This retort was developed in 1860, and its advantages over the horizontal retorts quickly became evident. Although various types of horizontal retorts were used for many years, they were gradually superseded by the verticals, until to-day the verticals are the only type used. These increased the yield of oil from a given shale by more than 10 per cent, and although the oil from the verticals required a distillation before chemical treatment, whereas oils from the horizontals did not, even then the cost of the "vertical" oil ready for chemical treatment was less than that of oil from the horizontal retort

⁷⁹ English patent 625 (1866).

⁸⁰ Redwood, Boverton, *A treatise on petroleum*. London, vol. 2, 3d ed., 1913, p. 94.

at the same stage. The yield of naphtha from the "horizontal" was more than twice as large as that from the vertical, but the yield of illuminating and lubricating oils were higher and the yield of paraffin obtained from the "vertical" oil was nearly 50 per cent more than that from the "horizontal" oil. Inasmuch as paraffin wax was, and still is, the most valuable product obtained from oil in Scotland, the advantage of the vertical retort is evident.

In 1861 steam²¹ was first used in the Scottish retorts for the purpose of removing the oil vapors from the retorts as quickly as possible and thus preventing undue decomposition. Steam in vertical retorts not only increased the yield of oil upward of 2 per cent, but also produced an oil with a greater percentage of paraffin wax than when it was not used. Since 1861 steam has been generally used in vertical retorts. In addition to its effect on the oil, the use of steam made heat transfer more effective and the distribution of heat more uniform, thus making control of temperature less difficult. In the horizontal retorts the use of steam does not present such great advantages, because of the difficulty of obtaining effective distribution.

In the Price type of vertical retort coal was used as fuel—about 500 pounds of coal per ton of shale retorted. The average life of these retorts was four and a half years; then the bottoms were distorted and cracked that the whole retort had to be replaced.

The construction and operation of this type marked the greatest advance in the development of the present Scottish retort. The continuous vertical retort with an elliptical cross section was adopted in preference to the horizontal, and the use of steam in retorting was introduced.

In 1865 Robert Bell,²² of Broxburn, accidentally discovered that the water produced in retorting the shale was valuable as a fertilizer. Up to that time this water, considered a necessary nuisance, had been wasted. Bell noticed that the growth of grass was extraordinarily luxuriant in a field into which the waste water had been turned. An investigation drew attention to the ammonia content of the water, and the development of methods for the production of ammonium sulphate followed. Incidentally, it may be mentioned that the production of ammonium sulphate has been the life-saver of the industry in Scotland, as the oil yield alone would have been insufficient to make the industry pay.

From 1860 to 1873 various forms of horizontal and vertical retorts were tried, some designed to operate continuously and others intermittently, but none of these are particularly noteworthy and the

²¹ Redwood, Beverton, work cited, p. 96.

²² Redwood, I. I., *Mineral oils and their by-products*. London, 1897, p. 169.

use of the vertical types increased. In 1873 Henderson²² patented his first vertical retort, which marked a second great step in the development of the Scottish retort.

The features of the 1873 Henderson retort are (1) the spent shale was used as a fuel for heating the retort; (2) discharge of the spent shale was greatly facilitated; (3) the distillation products passed out at the bottom of the retort, instead of the top as in previous practice; and (4) the fixed or uncondensable gases resulting from the shale distillation were, after being scrubbed to remove naphtha and ammonia, led back to the retort and burned as fuel. The cast-iron retorts were vertical and of elliptical cross section; four were set in a common oven and heated by two furnaces. The shale was charged at the top from cars, and after 16 hours was discharged in a spent condition into one of the furnaces below, in which was also burned the retort gas, as has been noted. Periodically the shale ash was dropped from the furnace into a car, which took it to the dump. Operation of the retorts was intermittent, a retort being emptied completely before a fresh charge was added. Volatile products were removed downward and passed from the retort through a vapor line in the bottom. Steam, somewhat superheated, was passed into the retort at the top, which, on account of the design of the furnace, was the hottest and the bottom the coolest part of the retort.

There is a difference of opinion as to the real economic value of the combustion of the spent shale for fuel purposes in the 1873 Henderson retort. Since the shale was discharged comparatively cool, considerable heat was undoubtedly required to bring it to and keep it at combustion temperature. The spent shale from this retort contained considerable free carbon (9 to 14 per cent), as retort temperatures were too low to permit the action of the steam on the carbon to progress to any important extent. This fixed carbon was, of course, burned in the furnace, and probably supplied more heat than was needed to keep it burning. In any event, the burning of the spent shale and the fixed gases, as fuel, reduced the amount of coal required in retorting about 50 per cent. On account of the ease of charging and discharging, the labor required was cut 33 per cent.

The yield of oil from the first Henderson was about the same as that from the older vertical retorts, but the total yield of finished oil products, especially wax, was notably greater in the former. The yield of ammonia from the Henderson was, however, only about half as much as from the old verticals. The life of these retorts, under proper care, was upward of five years.

²² English patent 1327 (1873).

About 1880⁸⁴ Young and Beilby discovered that the recovery of ammonia could be greatly increased if after the removal of the volatile matter from the shale the latter was heated to a high temperature and at the same time submitted to the action of superheated steam. It was soon determined that the yield of ammonia was increased as the carbon was removed from the shale by this action, and that the maximum amount of ammonia could be recovered only if all the carbon were so removed.

In line with these discoveries, Young and Beilby, working independently, constructed retorts designed to recover the oil from the upper part of the retort, where the temperature was kept relatively low; ammonia was formed at the bottom of the retort where the temperature was high. The work of these men demonstrated the advantages of their discoveries, but imperfections in retort design and construction caused the failure of their retorts. They then combined their ideas, and in 1882 patented the Young and Beilby, or "Pentland" composite retort,⁸⁵ which was the first appearance of the modern Scottish retort. With minor modifications, this type of retort is still in use in Scotland.

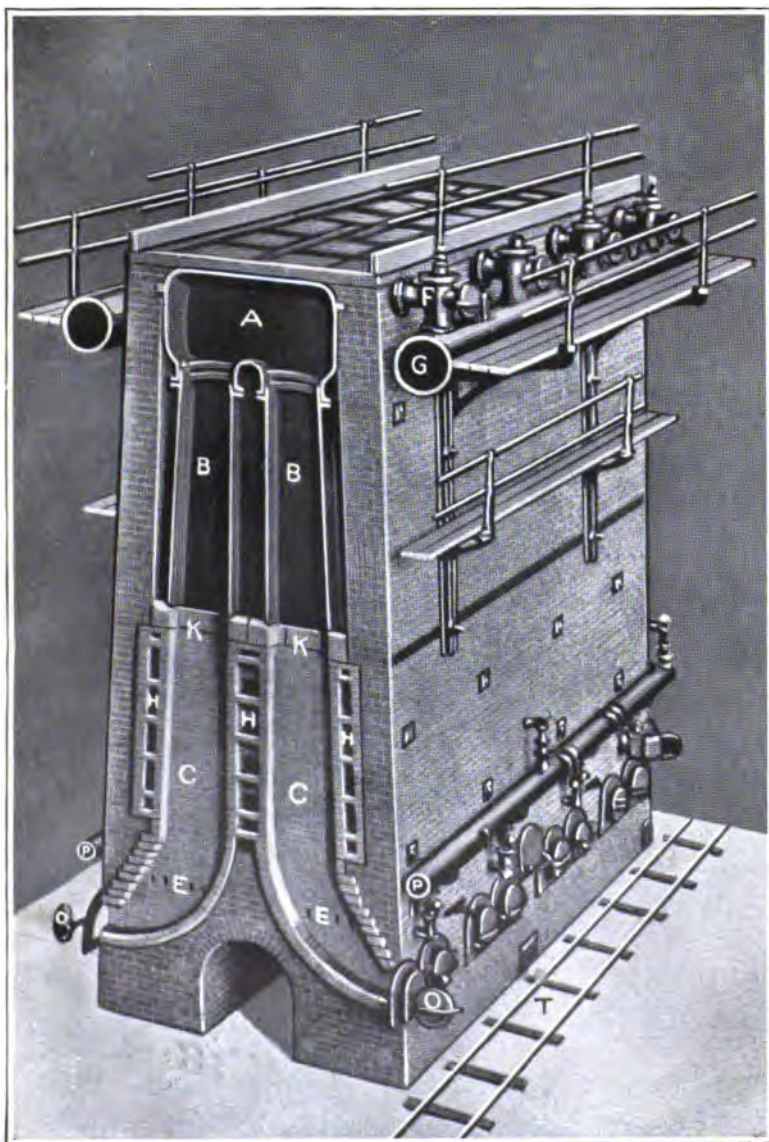
THE YOUNG AND BEILBY RETORT (1882).

Plate V is a reproduction of a sectional model of a bench of Young and Beilby retorts. This retort is also known as the "Pentland" composite retort, because the single retort performed two definite operations—(1) distillation of the oil from the shale at relatively low temperatures; and (2), after distillation of the oil, production of ammonia from the nitrogen of the shale at a comparatively high temperature with the aid of steam.

In operation a hopper or "jumbo" A (Plate V), which was common to and served four retorts, was filled with shale. The hopper had four lids for charging. The shale passed into the cast-iron part B of the retorts, which was of circular cross section and tapered slightly from top to bottom. In this part of the retort the oil was practically all distilled from the shale at a temperature of about 900° F. The shale residue then passed into the fire-brick part of the retort C (joined to iron part B with a fire-clay joint K), in which the shale reached a temperature of 1,300° F., and was subjected to the action of superheated steam entering the retorts at E. In later retorts low-pressure steam without superheat was used. In this part of the retort much of the nitrogen of the shale was converted into ammonia by the action of the steam; at the same time and in the same reaction the bulk of the carbon in the residue was converted

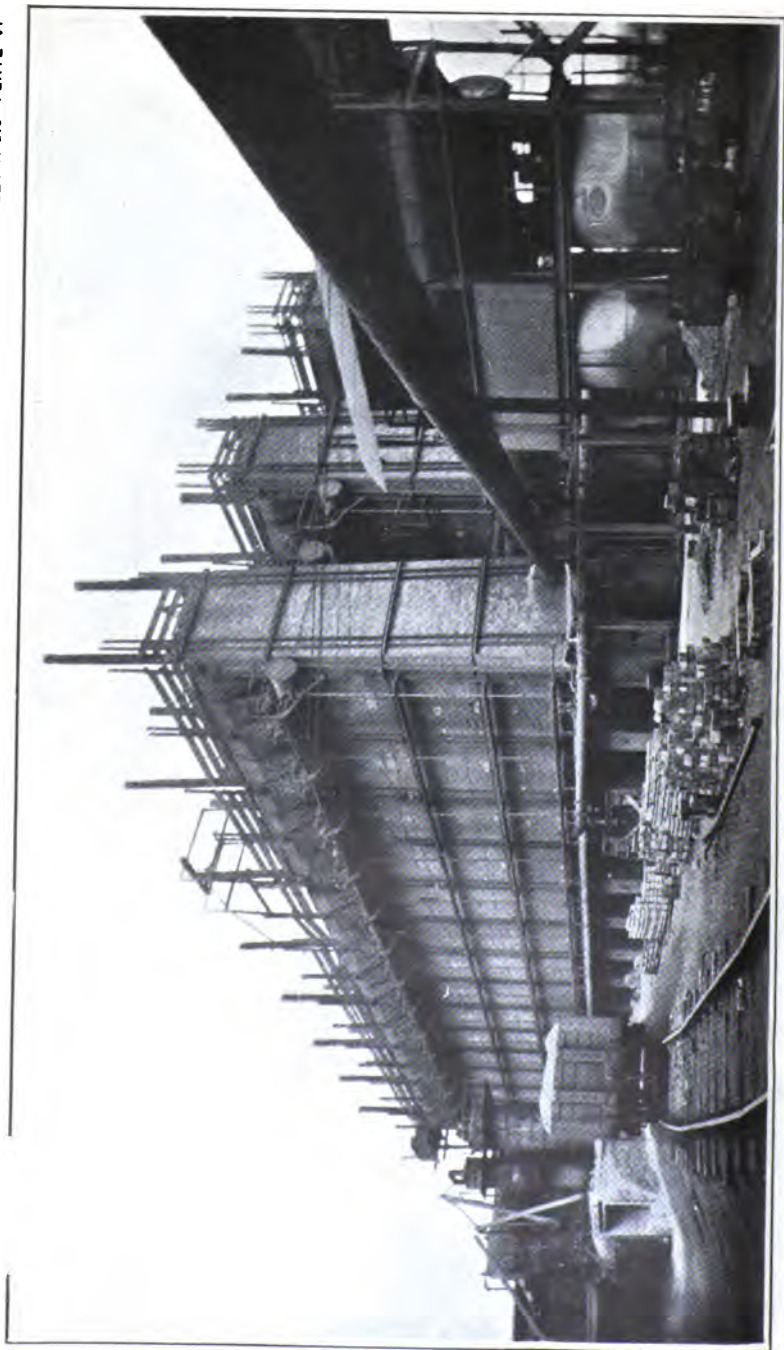
⁸⁴ Redwood, Boverton, *A treatise on petroleum*. Vol. 2, 3d ed., London, 1913, p. 96.

⁸⁵ English patents 1377 and 5084 (1882).



Courtesy Mr. H. M. Cadell.

YOUNG AND BEILBY'S "PENTLAND" RETORTS, 1881.



BENCH OF PUMPFERSTON RETORTS

Courtesy Scottish Oil, Ltd.

into water gas. Discharge doors O were opened every six hours to remove the spent shale, corresponding in volume to 850 pounds of broken raw shale, which passed into the retort with the removal of the spent shale. Gases and vapors resulting from the distillation were removed from the top of the hopper A after passing through the fresh shale therein, through vapor off-take F, into vapor main G, and thence into the condensers. Some type of exhauster was used in the vapor-line system to keep a slight suction in the retorts.

The passage of the distillation vapors through fresh shale in hopper A was expected partly to refine the oil and make the primary refining distillation unnecessary, but this arrangement did not accomplish what was expected of it.

Gas was used as fuel; it consisted of the fixed gases from shale distillation supplemented by producer gas made from dross coal. For each eight shale retorts there were two gas producers, which were so operated that the ammonia produced from the coal was recovered with the ammonia from the shale. Gas supplied by pipes P was admitted into and burned in flues H. Spent shale discharged from the retorts dropped into cars running on track T and was carried to the spent-shale dump.

A bench of 80 retorts had a capacity of 110 to 135 tons of shale per 24 hours; each retort thus put through 1.38 to 1.68 tons of shale a day. Each retort held about 3,300 pounds of shale, so about 24 hours was required for a particle of shale to travel from top to bottom of the retort. The amount of steam used varied with the quality of the shale, but averaged about 108 gallons of water, as steam, per ton of shale. The steam was commonly the exhaust from the works power plant and quickly became superheated in passing through the hot spent shale in the bottom of the retort. (See p. 73.)

The Young and Beilby retort had a life of six or seven years, if operated carefully. As in modern retorts, care had to be exercised in heating, especially to prevent the pulling apart of the joint between the iron and fire-brick parts of the retort. Coal consumption in this retort was 100 to 300 pounds per ton of shale retorted. The oil produced was a little less in quantity but somewhat better in quality—especially in yield of paraffin and lubricating oil—than that produced in the old Henderson retort, but the yield of ammonia was two and one-half to four times as great as from the latter. The Young and Beilby retort, therefore, soon practically superseded all earlier types, until it was itself replaced by the more modern Young and Fyfe, Henderson, and Pumpherston (or Bryson) retorts. All these latter types are, however, based on the principle of the Young and Beilby, and essentially differ from the latter only in dimensions and in being continuous in operation instead of intermittent.

THE HENDERSON RETORT OF 1889.

In 1889 Henderson patented a new retort based on the Young and Beilby principle.⁶⁶ This retort in its improved form,⁶⁷ which is shown in Plate VII, is used to the exclusion of others by the Broxburn Oil Co. (Ltd.), now a part of Scottish Oils (Ltd.).

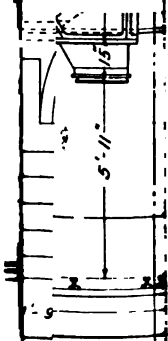
Plate VII, *B*, represents an end section of a bench of the improved Henderson or "Broxburn" retorts, showing two retorts in section. One malleable iron hopper (1) is provided for each retort. These hoppers have a capacity of about 54 cubic feet and hold a supply of broken shale sufficient to run the retort for about 18 hours. The cast-iron part 6 of the retort is rectangular, measuring 2 feet 9½ inches by 1 foot 2¾ inches at the top and increasing with constant taper to 3 feet ¼ inch by 1 foot 5½ inches at the bottom, where it is joined to fire-brick section 8 by a fire-clay joint 7. The length of the iron part of the retort is 14 feet. The fire-brick part of the retort is made up of specially shaped brick, and the length of this part is usually 20 feet. Its cross section is rectangular, measuring 3 feet ¼ inch by 1 foot 5½ inches at the top, where it is joined to the iron part. At the bottom the cross-sectional measurements have increased to 4 feet 8 inches by 1 foot 10 inches. The shale in the retort rests on a pair of toothed rolls 9, about 4½ feet long, at the bottom of the retort. These rolls rotate slowly toward each other at a regulated rate, permitting a controlled throughput of shale. Spent shale is discharged continuously by the rolls into hoppers 10, one for each retort. These hoppers are discharged periodically into cars on tracks 25 running beneath the retort benches. The over-all height of the retort bench is about 65 feet.

Vapors pass out of the tops of the retorts through vapor lines 3, into 30-inch headers 5, and into condensers as subsequently described (Pl. IX). A slight vacuum is maintained in the retorts by some form of exhauster, often a fan, in the condenser system. Steam is admitted into the bottom of the brick part of the retort in about the same proportions as indicated for the Young and Beilby retort. Practically all of the oil is distilled from the shale in the cast-iron part 6 of the retort, at a temperature of about 900° F. The spent-oil shale reaches a maximum temperature probably exceeding 1,500° F. in the bottom of the fire-brick part 8, but is discharged from the hoppers relatively cool, having been cooled by the incoming steam.

The new Henderson retort puts through, in a day of 24 hours, 4 to 4½ tons of the grade of shale now being mined in Scotland. The shale remains in the heat of the retort 24 to 30 hours, and thus is heated very gradually. As compared with older types, the retort yields

⁶⁶ English patent 6726 (1889).

⁶⁷ English patent 28647 (1901).



B. End 94

ALERT.

slightly more oil and notably more ammonium sulphate. On account of the greater length of the retorts, depreciation is slower, and the life of a retort is 7 to 10 years.

The retorts are heated by the fixed gases from the distillation of the shale supplemented by producer gas made from dross coal, after both gases have been scrubbed for gasoline and ammonia. Four retorts are set in 1 oven, and 16 ovens comprise a bench, 2 retorts wide and 32 long. At the crude-oil works of the Broxburn Oil Co. (Ltd.) $3\frac{1}{2}$ benches, or 224 retorts, of this type are in operation. The Henderson retort of 1889⁸⁸ differed from the improved type mainly in being shorter, and the discharge mechanism had one roll instead of two.

THE YOUNG AND FYFE RETORT.

Young's Paraffin Light & Mineral Oil Co. (Ltd.) uses a modified Young and Beilby retort called the Young and Fyfe.⁸⁹ This type is the old Young and Beilby retort made continuous, but inasmuch as its efficiency is lower and operating costs higher than both the Henderson and the Pumpherston no detailed description is given here. The top hoppers are provided with rocking shafts to which chains are attached to facilitate the regular passage of raw shale onto the retorts. Spent shale is discharged by a type of double screw into a large combustion chamber at the bottom of the retort. In this chamber the residual fixed carbon of the shale is burned out, but apparently this does not increase the efficiency of the retort or decrease the cost of fuel to any appreciable extent.

THE PUMPHERSTON RETORT.

The Pumpherston or Bryson retort, invented by Bryson, Fraser, and Jones, of the Pumpherston Oil Co. (Ltd.),⁹⁰ is undoubtedly the most efficient and best retort that has been devised to treat Scottish oil shales. Plate VI shows the external appearance of this type of retort, which is used exclusively by the Pumpherston and Oakbank Oil Companies (Ltd.), both now part of Scottish Oils (Ltd.). The Pumpherston company was, before the consolidation, the largest oil-shale company in Scotland, and during the past decade has paid the largest dividends of any of the present six producing oil-shale companies.

The Pumpherston retort is vertical and circular in cross section. It consists of five essential parts (see Pl. VIII, A); a hopper, a; a cast-iron part, b; a fire-brick retort part, d, joined to b at c; a discharge mechanism, e; and a spent-shale hopper, f.

⁸⁸ English patent 6726 (1889).

⁸⁹ English patents 13665 (1897) and 15236 (1899).

⁹⁰ English patents 8871 (1894), 7113 (1895), 4249 (1897).

slightly more oil and notably more ammonium sulphate. On account of the greater length of the retorts, depreciation is slower, and the life of a retort is 7 to 10 years.

The retorts are heated by the fixed gases from the distillation of the shale supplemented by producer gas made from dross coal, after both gases have been scrubbed for gasoline and ammonia. Four retorts are set in 1 oven, and 16 ovens comprise a bench, 2 retorts wide and 32 long. At the crude-oil works of the Broxburn Oil Co. (Ltd.) 3½ benches, or 224 retorts, of this type are in operation. The Henderson retort of 1889⁸⁸ differed from the improved type mainly in being shorter, and the discharge mechanism had one roll instead of two.

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⁸⁸ English patent 6726 (1889).

⁸⁹ English patents 13665 (1897) and 15236 (1899).

⁹⁰ English patents 8871 (1894), 7113 (1895), 4249 (1897).

Hopper a (one for each retort) is of wrought iron and holds enough shale that the retorts need not be charged during the night or over Sunday. The top or cast-iron part of the retort is 11 feet high, circular in cross section, tapered, 2 feet in diameter at the top and 2 feet 4 inches at the bottom. This part is a single casting, about 1 inch thick. At the top is an 8-inch vapor offtake, j, cast integral with the retort; through this the distillation products of the shale pass from the retorts into the vapor header k. The angle between the retort and the vapor line j is 45° , and a lip, l, keeps the shale from clogging the offtake.

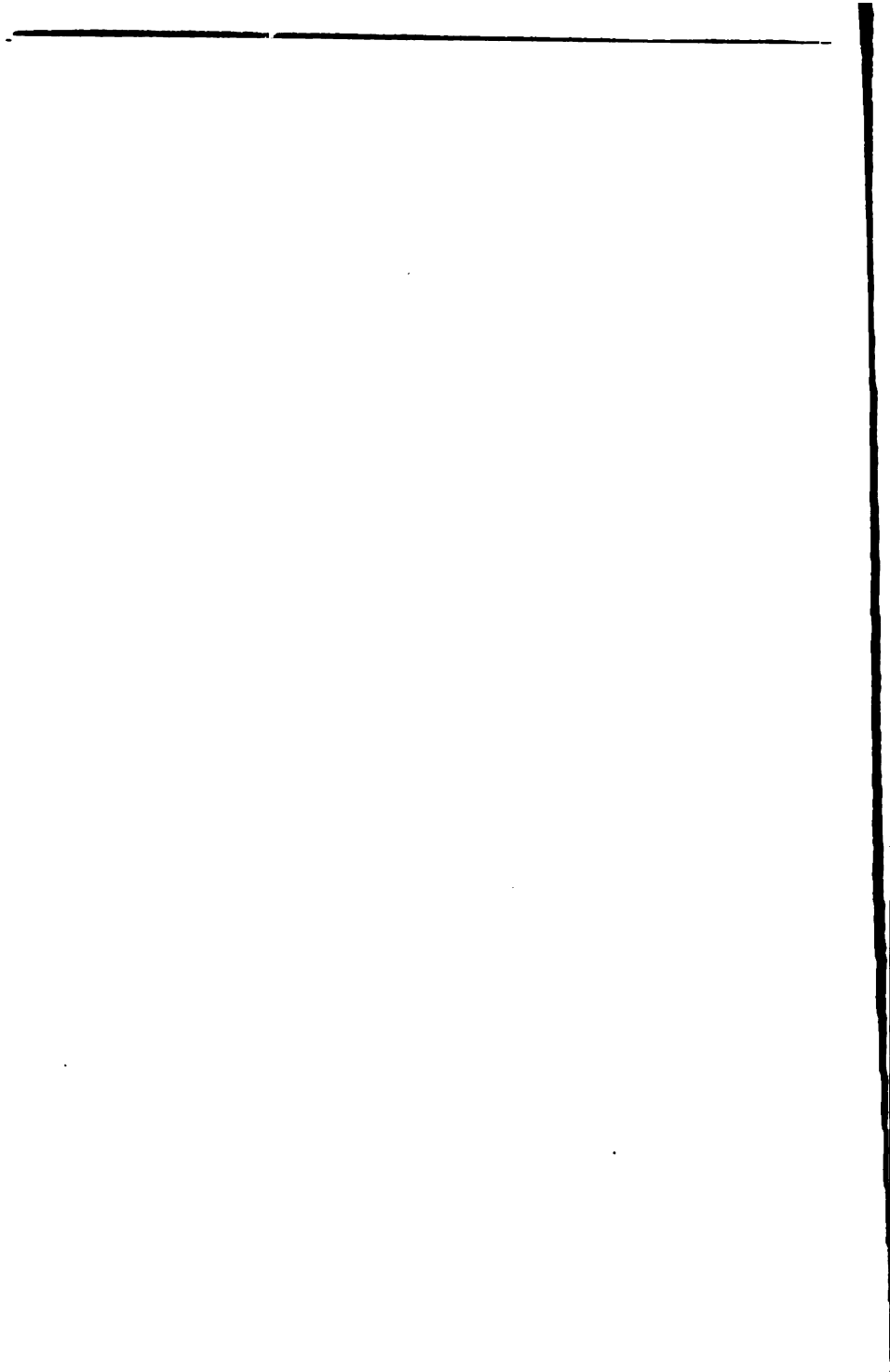
At c the cast-iron part of the retort rests on, and is joined to, fire-brick part d. The joint is made with a special fire-clay mixture. The fire-brick part d is 18 feet high, of circular cross section, and, like the iron part, tapers at a constant rate from top to bottom. It is 2 feet 4 inches in diameter at the top where it joins the iron part, 3 feet in diameter at the bottom, and is made of a single tier of brick.

The brick section is constructed of specially shaped fire brick, made of exceptionally fine fire clay found in the vicinity of the shale fields. The bricks are laid with a thin coat of fire clay. At points a, Plate VIII, A, are special bricks extending from this fire-brick part and touching similar bricks, m, in the outer wall of the retort. These bricks are not tied to each other, are free to move and expand and contract, but provide a support for the retort itself. The whole retort is in no way tied to the outer wall at any place, but each retort is free to expand and contract as a unit, without putting strain on any part of the bench. Just above the bottom of the retort is a constriction n, below which is a circular iron table, o, that supports the shale in the retort. Above this table is a curved arm, p, which revolves above the table, being driven by a system of ratchets and pawls as indicated at x, (Pl. VIII, B). As it revolves, this arm thrusts the spent shale from the retort into spent-shale hopper q, common to two retorts.

Since the shale falls apart and expands about 25 per cent in volume during retorting, and also tends to stick and bridge, the retort is tapered, as noted, to permit the shale to move freely.

The revolving arm ordinarily makes four revolutions an hour, but its rate, and consequently the rate of discharge of spent shale, can be regulated. The arm, besides discharging spent shale from the retort, gives the shale a certain amount of movement in the retort and discourages any tendency of the shale to bridge, clog, or stick. Hopper f is discharged periodically into spent-shale car g by lowering bell s. It is to be noted that one track beneath the bench





of retorts takes care of the spent shale from all the retorts of the bench, because of the design of the discharge hoppers.

Steam—exhaust from the power plant—is admitted into spent-shale hoppers at t, and passes upward into the retorts. Gas—used as fuel—enters combustion spaces at the bottom of the brick section, and these combustion chambers are divided into four segments by baffle walls which cause the flames to circle the outside part of the retort. Usually the hot gases from the brick section pass to the top of the oven v, surrounding the iron parts, and strike an arch, w, which directs them downward into a flue, y, that starts at the bottom of the housing for the cast-iron section. Two smokestacks, r, serve a unit of four retorts, set in a common furnace; each unit of four is separated from the next unit by fairly thick brick walls. Sixteen units constitute a bench of retorts, 2 retorts wide and 32 long.

The Pumpherson retort proper holds $4\frac{1}{2}$ tons of shale and its present throughput of shale is 4 to $4\frac{1}{2}$ tons a day. Therefore a piece of shale requires 24 hours to pass from top to bottom of the retort. Inasmuch as the raw-shale hopper at the top of the retort holds $4\frac{1}{2}$ tons in addition, a piece of shale requires 48 hours to travel from the raw to the spent-shale hopper.

In operation, freshly broken shale is fed from time to time from cars u (Pl. VIII, A) into hopper a, from which it gradually passes into the iron part b, where practically all of the oil is distilled at a temperature probably not exceeding 900° F. The shale then passes into the brick part d of the retort, where it is subjected to the action of steam and reaches a final temperature in the lower part of the combustion chamber, certainly not in excess of 1,800° F. The amount of steam used varies with the quality of the shale, but usually amounts to about 100 gallons of water—as steam—to each ton of shale treated; that is, to each 25 gallons of oil produced.

The spent shale discharges into the spent-shale hopper f, and is finally dumped from the latter, as described, into cars g. It usually contains only $1\frac{1}{2}$ to 2 per cent of fixed carbon, which could be removed by the action of heat and steam, but this amount is allowed to remain as the economic limit with respect to fuel consumption, throughput of shale, and ammonia and gas yield. Probably a larger yield of ammonia and gas per unit weight of shale could be obtained by reducing the amount of fixed carbon in the spent shale through the use of more steam and longer retorting time, but with decreased throughput and increased operating costs.

OPERATING TEMPERATURES.

Temperature measurements within the retort itself have never been made. A series of heat measurements in different parts of the com-

bustion chambers immediately surrounding the retorts gave the following temperatures, which can probably be considered as averages:

Temperatures in combustion chambers.

| Section of furnace: | °F. |
|----------------------------------|-----------|
| Bottom----- | 1,800 |
| Middle of brick part----- | 1,650 |
| Top of brick part----- | 1,500 |
| Middle of iron part----- | 1,100 |
| Top of iron part----- | 900-1,350 |
| Average of top of iron part----- | 1,100 |

The fires and temperatures in the different zones of the retorts are controlled by one man per shift, who determines retort temperatures visually through peepholes at various levels of the retort.

CAPACITY OF THE RETORT.

The capacity of a Pumpherston retort has been given as 4 to 4½ tons of shale per 24-hour day. This is true as to the shale now being treated, but experiments have indicated that the capacity of the retort is one of oil production rather than of shale throughput. It may be stated that the maximum daily capacity of a Pumpherston retort is about 96 gallons of oil, when operating under best conditions; that is, producing all the ammonia possible and the greatest yield of the best oil possible (this does not include scrubber naphtha). Therefore, in round figures, a Pumpherston retort will handle 10 tons of 10-gallon shale and 1 ton of 100-gallon shale a day. Table 12 indicates this general tendency, the figures being derived from actual test runs.

TABLE 12.—*Capacity of Pumpherston retorts—effect of oil yield of shales on throughput.*

| Yield of shale
(gallons per
ton). | Throughput of
shale (tons
per day). | Oil yield per
retort per day
(gallons). |
|---|---|---|
| 16.1 | 3.42 | 72.7 |
| 19.5 | 4.25 | 81.4 |
| 24.2 | 3.81 | 92.0 |
| 29.6 | 4.32 | 129.5 |
| 28.2 | 4.32 | 121.2 |
| 35.4 | 2.63 | 92.7 |
| 61.4 | 3.26 | 207.5 |
| 135.8 | 1.45 | 197.3 |

a Australian shale; throughput very slow, as the shale tended to intumesce and sinter in the retort.

LIFE OF THE RETORTS.

A Pumpherston retort has a life of over 20 years; many have now been working continuously for that time. The average life is probably not so long, but with minor repairs is well over 12 years. The

cast-iron parts fail by overheating and swelling, which later produces cracking, and the brick part fails by cracking and wearing out. The scouring action of the shale in the lower part of the retort slowly wears that part thinner, hence the bricks used are selected for their ability to resist abrasion rather than for their heat conductivity. Much care must be taken in starting and operating the retorts to avoid sudden temperature changes, local overheating, or explosions.

LABOR FOR RETORTING.

A bench, or 16 units of 4 retorts each, 64 retorts in all, is operated by 1 charger, 1 dropper, 1 tip man, and 1 man to clean pipes and to regulate heats, etc., a total of 4 men a shift, or 12 men a day. In May, 1920, the shale workers were granted a 7-hour day in place of 8 hours.

REPAIRS.

As a unit consists of 4 retorts, when 1 is being repaired the 4 must shut down. After repairing a retort about 24 hours is required to heat the unit up to operating temperature. If a bench is shut down, the time required for heating up is two weeks. When a retort has been off for repair it is heated slowly, and preferably on spent shale, but often fresh shale is charged in at once and distillation commences immediately. In charging an empty retort care must be taken to prevent disastrous explosions. Vapor mains are usually provided with explosion doors to provide for such accidents.

USE OF STEAM IN RETORTING.

As stated before, all modern Scottish oil-shale plants submit their shale to the action of steam in the retorts at a relatively high temperature. The steam serves many purposes, and its use in these retorts is essential. Exhaust steam entering the bottom of the retorts cools the hot spent shale passing from them and becomes superheated; then, passing into the retort, the steam takes back much heat that would otherwise be wasted. The steam also distributes the heat evenly throughout the shale in the retort, thus helping to produce uniform temperatures, and sweeps the oil vapors out of the retort before they can decompose much. (See p. 177.) At the temperature of the base of the retort, 1,300° to 1,800° F., steam reacts with the fixed carbon of the shale to produce a mixture of carbon dioxide, carbon monoxide, and hydrogen. The two latter are combustible gases, valuable as fuels, and are made by the action of steam on the fixed carbon of the shale, which otherwise would not be utilized. The use of steam greatly increases the ammonia yield.

Some ammonia is produced from shales at low temperatures even without the use of steam, possibly by the decomposition of amines and the like; but when the carbon is removed from the shale by the use of steam much of the nitrogen of the spent shale is either set free in such condition that it can combine with the hydrogen produced by the interaction of steam and carbon, or, more likely, amino or imino compounds are freed, which are reduced to ammonia by the hydrogen. The interaction of the nitrogen and hydrogen, or the reduction of the nitrogen-containing compounds, produces ammonia, which, at the temperature of that part of the retort, would decompose completely into its elements if it were not protected by excess steam and hydrogen and the excess gas and steam did not immediately sweep it into a cooler zone. Under proper conditions practically all of the nitrogen of oil shales can be converted into ammonia by the use of a large excess of steam and relatively high temperatures, but in practical retorting in Scotland only about 60 per cent of the nitrogen in the shale is converted into ammonia.

The yield of paraffin from oil shales has been increased experimentally by Irvine,²¹ who passed ammonia into the retorts. A possible explanation is that part of the ammonia decomposed, giving free hydrogen, and the increased concentration of hydrogen in the oil-producing zone inhibited, to a certain extent, the decomposition of the shale oil. It hardly seems possible that hydrogenation of the oil takes place in the modern Scottish retort.

CONDENSATION AND SEPARATION OF PRODUCTS.

Large and comparatively low-speed rotary suction fans pull the vapors and gas from the top of the shale retorts and into the condensing and scrubbing system. These fans are placed between the last (oil) scrubbers and a large 2-foot gas header, to which the separate retort gas burners are connected; that is, the fans draw the vapors and gas from the retorts, through the condensers, water scrubbers, and oil scrubber, and deliver the gas into the burner-supply header. Gas holders are not used, since the total volume of the vapor pipes, condensers, scrubbers, and gas headers is large enough to take care of pressure fluctuations. The suction of the fans should not be more than 9 or 10 inches of water at the fan, but at the retort outlet it is ordinarily not over five-sixteenths inch. The whole retorting system, including retorts, vapor lines, condensers, and scrubbers thus operates under a slight negative pressure. The suction is naturally greatest at the fans, and least at the outlets of the spent-shale hoppers. Hence, in general, any leaks in this system are

²¹ Steuart, D. B., *The oil shales of the Lothians, Part III, The chemistry of the oil shales*: Memoir Geol. Survey, Scotland, 2d ed., 1912, p. 177.

of air leaking in, rather than of vapors and gases leaking out. Large leaks are detected by gauge readings and by an increase of the nitrogen content (nitrogen from the air) of the gases coming from the scrubbers.

From the retort the vapors and fixed gases pass into large vapor mains, usually 30 inches in diameter (the size varies, however, with the number of retorts connected to a main), which conduct the vapors to the condensers. At some plants boiler feed-water heaters, serving as economizers, are placed in the vapor mains, but this practice is not general.

The condensers (Pl. IX) are air cooled and vertical. They consist of cast-iron pieces 9 feet long and 4 inches in diameter, making an inverted U at the top, and fitting into headers at the bottom. The condensers are four joints of pipe and are about 40 feet high. For a 110-ton shale plant, 1,392 cast-iron pipes are used in addition to the headers, giving a total of 16,200 square feet of cooling surface, in addition to headers and return U's. In general, 15 square feet of cooling surface is used in Scotland for 1,000 cubic feet of vapors and permanent gases per 24 hours. About 147 square feet of cooling surface is provided for each short ton of shale retorted per day. This is suitable for Scottish shales yielding up to 38 gallons of oil per ton. Generally a tubular water heater is installed between the gas mains and condensers.

The condensers separate from the vapors and fixed gases the bulk of the oil and all of the water produced in distilling the shale, also the shale dust. The water and oil flow from the condensers to separating boxes or tanks, where they are separated by gravity settling. The water, which contains in solution nearly all of the ammonia produced from the shale, together with certain fixed nitrogen compounds formed in retorting, flows into a tank from which it is piped to the ammonium sulphate plant. The crude oil flows to another tank, whence it is either piped to the refinery or shipped there in tank cars, depending on the distance of the refinery from the retorting plant. The crude oil has a high congealing point, 84° to 92° F., through its high content of wax, and thus its transportation in pipe lines is attended with difficulties. In some plants the oil flows in a jacketed line to the refinery, the inner pipe carrying the oil and the outer pipe the hot water recovered from the condensers on its way to the ammonia plant. The ammonia plant is commonly operated in connection with the refinery.

The gases passing from the condensers still contain small quantities of ammonia and light hydrocarbons not removed by the condensers. In order to separate these, the gases are passed into vertical scrubbing towers; in the first they are scrubbed with water to re-

move ammonia, and in the second with an intermediate oil that moves the lighter hydrocarbons (scrubber gasoline or naphtha).

The number and size of the scrubbers vary with the size of plant. As a rule they are vertical towers about 30 feet high and 3 feet in diameter. At a retorting plant there are usually two or three such towers for water scrubbing, and two or more for oil scrubbing. The towers are partly filled with coke or wooden checkerwork to insure intimate contact of the washing liquid with the gases. The gases enter the towers at the bottom and the scrubbing liquid enters at the top; the gases leave the towers at the top and the liquid at the bottom. In the first towers the gases are thus washed with water and thereby ammonia, equivalent to about 3 pounds of ammonium sulphate to a ton of shale retorted, is recovered. From the scrubbers the ammonia-containing water is piped to the sulphate plant, where it is combined with the condenser water.

In the oil scrubbers the gas is washed with an intermediate oil fraction of shale oil distilling between kerosene and the first lubricating fraction, and about 2.4 gallons of gasoline are thus recovered from the gases of a ton of shale retorted. From the scrubbers the scrubber oil, containing the dissolved gasoline, flows to continuous steam stills, in which the gasoline is distilled from the oil. The latter is then cooled and reused for scrubbing, the same oil being used successively. The scrubbing of shale gases for gasoline is similar to methods used in the United States for obtaining gasoline from natural gas by the absorption method.²²

Sometimes the fan is placed between the water and oil scrubber but usually the permanent gases pass from the scrubbers to the fan and are forced under low pressure into the large header, which leads them back to the retorts where they are burned as fuel in the furnaces.

PRIMARY OR CRUDE PRODUCTS.

At this stage the crude products obtained are permanent gases; crude oil, with which may be included scrubber naphtha; ammonia water from scrubbers and condensers; and spent shale.

SHALE GAS.

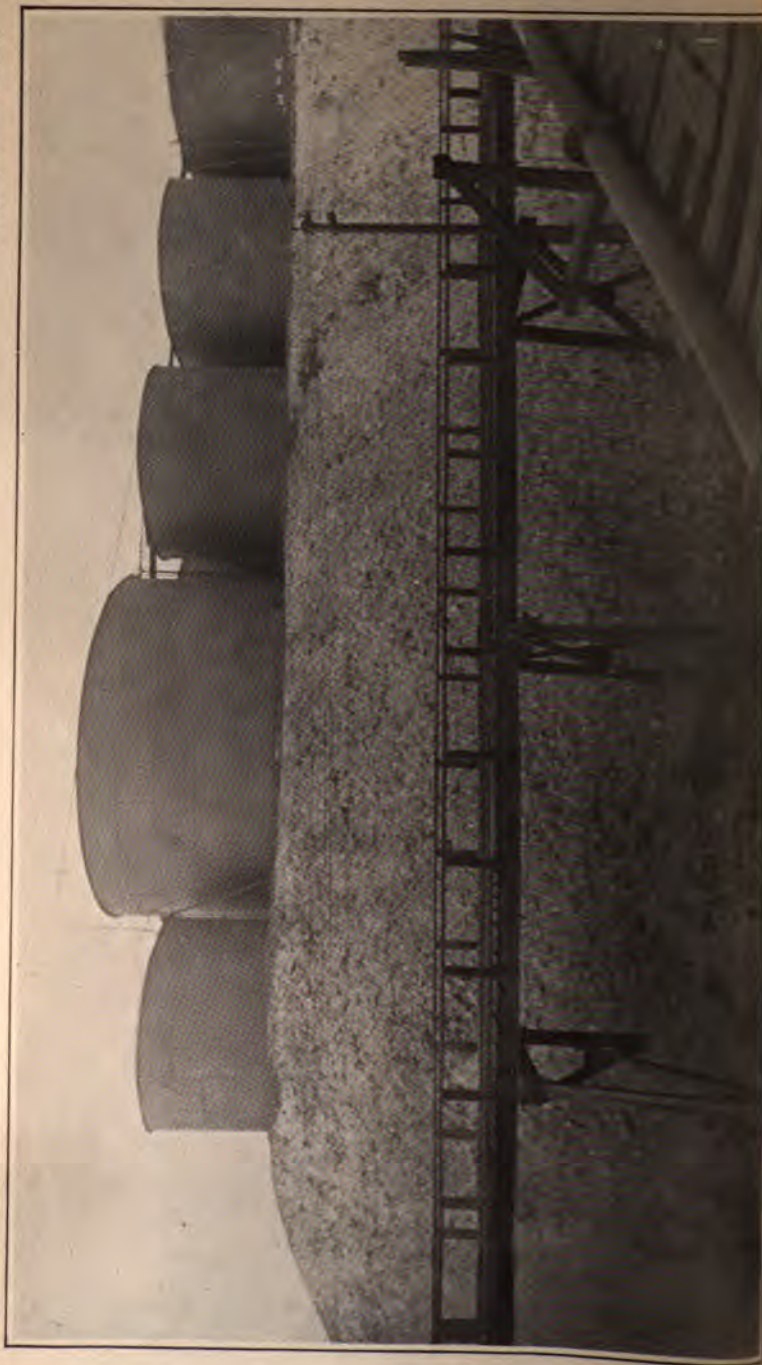
Shales treated in Scotland in 1921 produced about 9,800 cubic feet of gas per ton retorted. A typical analysis of the gas is given in Table 13.

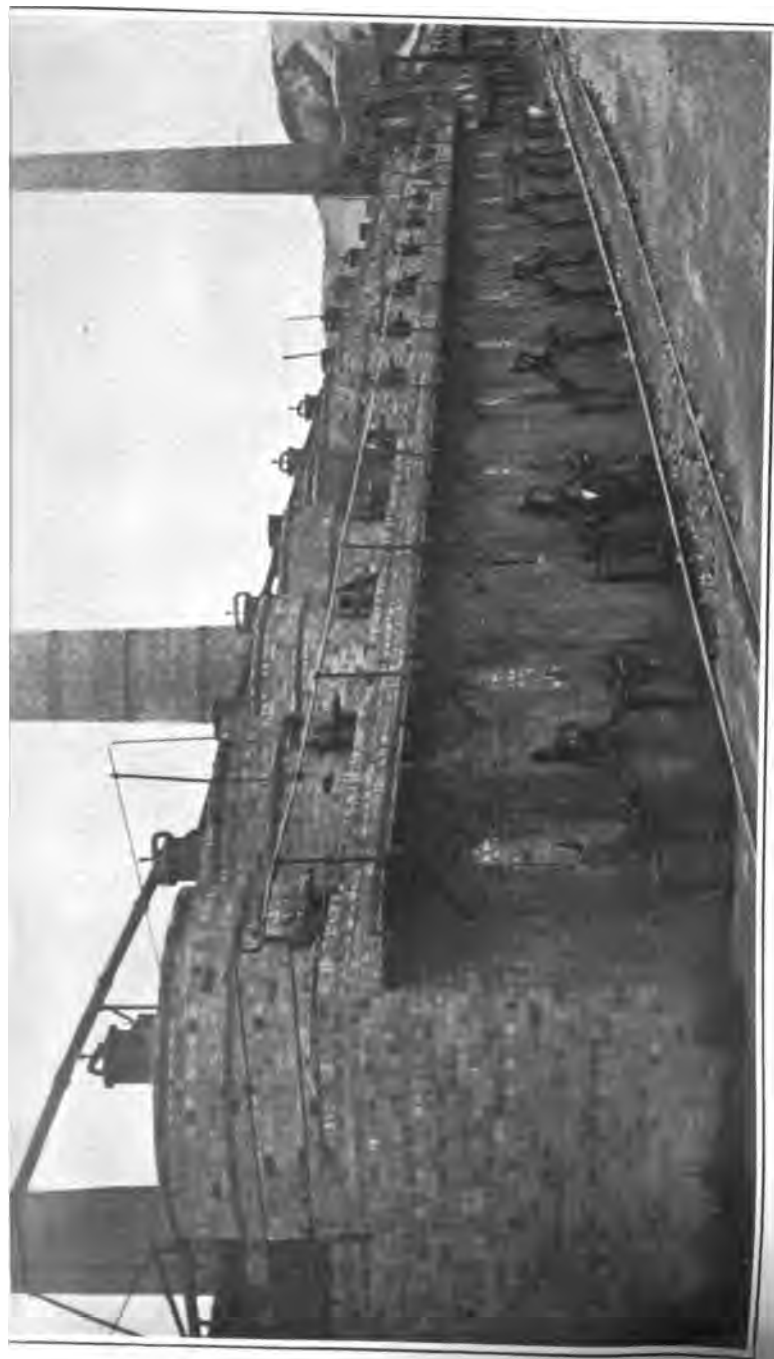
²² Dykema, W. P., Recent developments in the absorption process for recovering gasoline from natural gas: Bull. 176, Bureau of Mines, 1919, 90 pp.



ATMOSPHERIC CONDENSERS FOR CRUDE OIL.

Courtesy Scottish Oils, Ltd.





SCOTTISH SHALE-OIL DISTILLING PLANT.

Courtesy Scottish Oil, Ltd.



TABLE 12.—*Typical analysis of gas produced in retorting Scottish oil shale.*

| Constituent: | Per cent. |
|---|-----------|
| Carbon dioxide (CO ₂) | 20.00 |
| Carbon monoxide (CO) | 4.28 |
| Hydrogen (H ₂) | 34.21 |
| Methane (CH ₄) | 10.80 |
| Olefines (C ₂ H ₄) | 3.06 |
| Oxygen (O ₂) | 5.10 |
| Nitrogen (N ₂) | 22.55 |
| Total | * 100.00 |
| Calorific value, gross, 302.5 British thermal units per cubic foot. | |
| Calorific value, net, 271.4 British thermal units per cubic foot. | |

* The gas always contains a trace or more of hydrogen sulphide (H₂S).

CRUDE OIL.

Crude oil obtained from the shale now being worked in Scotland (1921) amounts to about 24.5 gallons of oil (including 2.4 gallons of scrubber naphtha) per ton treated. The crude has a specific gravity varying from 0.870 to 0.895 (31.1 to 26.6° A. P. I.) and a setting or congealing point of about 84° to 91° F.

The oil is green, greenish brown, or brown, and is a complex mixture of organic compounds. (See p. 88.) It has a peculiar odor, faintly resembling garlic, but not particularly disagreeable.

AMMONIA WATER.

The ammonia water from the condensers and scrubbers has a yellowish color, and usually has an odor of pyridine or quinoline. In addition to dissolved ammonia it contains fixed nitrogen-containing compounds, such as pyridines, cyanides, and the like. In 1919 the shale treated in Scotland produced about 35.7 pounds of ammonium sulphate per short ton. This represents about 60 per cent of the total nitrogen in the raw shale.

SPENT SHALE.

The spent shale, or residue from the retorts, which amounts to 75 or 80 per cent of the weight and 125 per cent of the volume of the raw shale treated, has no value. Its removal involves an expense, and often land on which to dump it must be leased. All the Scottish plants are marked by immense spent-shale dumps or "bingas." (See Pl. X.)

Small particles of spent shale are black, because of their fixed carbon; large pieces are bluish white on the edges, where the carbon has been completely removed. The fixed carbon in the spent-shale dump slowly oxidizes, and the small amount of iron present changes

from the ferrous to the ferric form; thus the color of the bing slowly changes from grayish black to a reddish color, and because of these reactions the dumps remain warm and give off an unpleasant odor.

Spent shale has been tried for road and brick making in Scotland, but with poor results. The colloidal properties of the silica in the shale have been destroyed by the heat of the retort, consequently spent shale has no value for making brick even when it is oxidized, as the binding properties of the clay are entirely destroyed.

YIELD OF PRODUCTS.

Table 14 shows the crude or primary products yielded by 1 ton of Scottish oil shale of the kind treated in 1919 (retorted with 100 gallons of water, as steam).

TABLE 14.—*Crude products from 1 ton of Scottish oil shale (1919).*

| Product. | Quantity. | Properties. | Disposal. |
|-------------------------------|-------------------------|--|--|
| Gas..... | 9,800 cubic feet..... | Heat value 270 B. t. u. per cubic foot. | Used as fuel in retort furnaces. |
| Crude oil ^a | 24.5 gallons..... | Specific gravity 0.860; setting point 86° F. | Sent to refinery. |
| Ammonia water ^b .. | 35.7 pounds..... | | Sent to sulphate plant for making ammonium sulphate. |
| Spent shale..... | 1,500-1,600 pounds..... | | Dumped as waste. |

^a The crude oil includes scrubber naphtha. The crude and scrubber naphtha are not ordinarily mixed and are kept separate during refining operations. If mixed in proper proportions, however, the specific gravity of the mixture is as given. The specific gravity of the crude oil alone is about 0.885.

^b Expressed in terms of equivalent $(\text{NH}_4)_2\text{SO}_4$.

REFINING SHALE OIL.

The refining of shale oil in Scotland resembles, in general principles, the refining of petroleum in other parts of the world but is more involved than the equivalent refining of petroleum, because shale oil is, chemically speaking, more reactive than the average petroleum, and it contains a greater percentage of those substances that must be removed in refining to produce marketable products. Shale oil refining is more costly than petroleum refining, chiefly because of this difference in the composition of the two oils.

The most valuable product derived from Scottish shale oil has always been paraffin wax. Therefore conditions for retorting the oil shales are such as will produce, within economic limits, an oil containing the maximum amount of wax, and refining operations are such as will produce, as cheaply as possible, the maximum amount of marketable wax from the crude oil.

A Scottish shale-oil refinery differs from the ordinary American petroleum refinery of equal capacity in having (1) a much greater number of small stills, a large proportion of which are intermit-

tent or batch stills; (2) a greater number of small and often horizontal agitators; (3) an extraordinarily large wax plant; (4) a separate condensing tank for each still; and (5) most of the oil in various stages of refining is handled by compressed air instead of pumps. This necessitates the use of strong horizontal cylindrical tanks, many of them on the ground level. Another difference noticeable to the petroleum refiner is the absence of "look boxes" for watching the flow of distillates in the delivery lines from condensers. The streams of distillates are always in sight, however, as they flow from the condensers to the distributing lines. Plate XI gives a general view of the distillation plant of a Scottish shale-oil refinery.

Very little is known regarding the actual chemical composition of shale oils (p. 88). They contain, however, a large percentage of compounds that must be removed before the products are of commercial grade. Probably most of these compounds are members of the various unsaturated series of hydrocarbons, phenols, and the like, and various nitrogen-containing bodies, such as homologues of pyridine. Although the effort is made to fix as much as possible of the nitrogen of the shales as ammonia, yet undoubtedly the crude oil contains a considerable percentage of nitrogen compounds. This is evidenced by the fact that a 10 per cent sulphuric acid solution will remove about 10 per cent of the volume of the crude oil. The objectionable substances mentioned are removed by treatment with sulphuric acid and caustic soda or soda ash, so most of the refinery loss is in the form of acid and alkali tars and sludges. Because of the high percentage of objectionable compounds acid and alkali treatments are more numerous than in ordinary petroleum refining, and redistillations of unfinished products are more frequent.

Plate XIII gives the ordinary procedure at a shale-oil refinery.⁹² The chart shows that the crude products of shale distillation are ammonia liquor, crude oil, gas, and spent shale. The gas, as before mentioned, is scrubbed for ammonia and naphtha, and the scrubber oil, containing the naphtha, is distilled and crude naphtha recovered therefrom. The crude naphtha is then treated with acid and alkali and redistilled, giving various grades of finished naphtha and still bottoms, which are added to the crude burning oil from the crude.

The crude oil, as Plate XIII shows, is first distilled, giving (1) coke, (2) a crude distillate (green oil), and (3) crude naphtha. The naphtha is treated with acid and soda, and yields (4) several

⁹² Some of the terms used in refinery procedure to describe the oils, such as "blue oil" and "green oil," date from the time of the first shale-oil refinery, that of Young, when much secrecy was thrown around the industry. These and similar terms were used to mislead the curious.

finished naphthas, and (5) bottoms, which are added to the crude burning oil fraction of the green oil.

The green oil (2) is treated with acid and soda and again distilled, yielding (1) coke, (6) crude burning oil, to which is added (5) bottoms from crude naphtha distillation, and (7) heavy oil containing paraffin wax. The crude burning oil (6) is chemically treated and fractionated into (8) motor-boat oil (a distillate fuel for internal-combustion motors of the Diesel type), burning oil (9), and light gas or distillate fuel oil (10). Burning oil (9) is again treated and yields the finished lamp oil or kerosene (11). The heavy oil carrying paraffin wax (7) is cooled in refrigerators, pressed in filter and hydraulic presses, and yields blue oil (12) and hard paraffin scale (13), which is mixed with soft scale (22), obtained in further processing of the blue oil. The hard scale (13) is sweated in sweating houses (Pl. XV) and yields an oil (14) which is added to the green oil (2), or crude distillate; wax mixed with strainings (15), which is added to more hard scale (13), and again sweated; and various grades of waxes (16), which are melted and filtered, or treated with fuller's earth, after which they are ready for the market.

The blue oil (12) is treated with acid and alkali, and subsequently distilled to coke, yielding finished heavy burning oil (19), light gas oil (18), heavy gas oil (17), unfinished lubricating oils (20), and coke (21). A residual oil is also obtained and used in making greases. The heavy gas oil (17) is separately cooled and pressed, yielding finished gas or fuel oil (24) and soft scale (22), or wax, which is sweated, the final product being miners' wax (23); or the soft scale (22) is mixed with hard scale for the production of finished wax. The unfinished lubricating oils (20) are also separately treated in a similar manner, yielding like products, and lubricating oils (25), which are again treated with acid and alkali before being ready for the market.

The diagram and description are necessarily brief, and not entirely complete, as fractions often overlap; for example, fractionation of the crude burning oil yields a certain amount of heavy naphtha or gasoline, which is added to the naphtha previously obtained.

The scrubber naphtha is handled separately, as may be noted on the diagram. On fractionation, however, it yields still bottoms, which are added to the crude burning oil.

Figure 3 shows diagrammatically the process of refining shale oil and indicates the approximate percentage of products yielded from a ton of shale. The rectangles are intended to be in proportion to the yields.²⁴ Plate XII is a view of the largest oil-shale retorting plant in Scotland, and Plate XIV shows the typical layout of a complete retorting and refining works.

²⁴ The writer is indebted to Mr. H. M. Cadell for permission to use this diagram.

NOTES ON REFINING.

1. The crude oil is settled in tanks for 12 to 18 hours to separate water and shale dust before being run to the stills.
2. In the more modern refineries a combination of continuous and batch stills is used for fractionations. Some arrangement such as

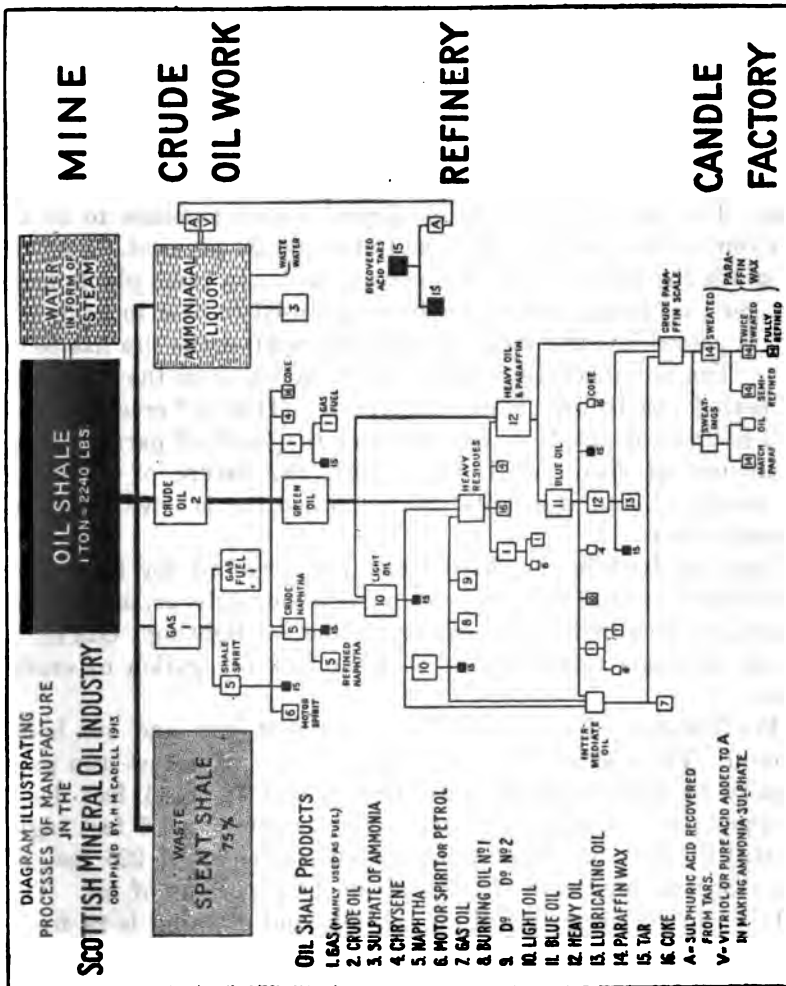


FIGURE 3.—Diagram of shale-oil refining process. Rectangles indicate appropriately the proportion of the product obtained from 1 ton of shale.

described below is often used. The center still of a battery of five stills receives the oil and removes the highest fraction. From this still the residue flows into similar stills on each side, which remove heavier fractions. These stills in turn feed to two more, where another fraction is removed. These finally deliver the residuum into heavy cast-iron pot or coking stills, operating intermittently. Each battery of five continuous stills has six or more of the pot stills, of which

three or more are on each side of the continuous battery. The final continuous still discharges into one pot still until the latter is full, and the stream is then switched to a second pot still and the oil in the first is distilled down to coke. Meanwhile a third pot still is cooling and being cleaned of coke, making ready to receive residuum from the continuous battery when the second pot still has filled up. The coke, amounting to about 2 per cent of the crude, is sold as fuel or is used for the manufacture of arc-light carbons, electrodes, and the like (see, however p. 86). In many refineries the continuous battery consists of three stills instead of five. Sometimes only batch or pot stills are used in refining the oil.

3. Steam is admitted to the bottom of the stills during all distillations. The quantity used, as condensed water, amounts to 10 to 25 per cent of the total distillate and averages 20 per cent. Part of the steam is the exhaust from the power plant. In some plants heat exchangers, set in the still vapor lines, are used to heat ingoing oil. Steam is passed into the still for three hours after the fire has been drawn. The only exception to the use of steam is in the pot stills for a period just before coking takes place. Thus a "cracking" of the oil is brought about, which increases the yield of paraffin wax. The amount of steam used varies with the nature of distillate being produced, and is usually greatest when the heavier fractions are coming over.

4. Gases evolved in the distillations are scrubbed for light oils, and are either used as fuel for heating retorts or stills or, on account of their high illuminating and calorific value, for lighting. Gas from the stills amounts to more than 1 cubic foot per gallon of crude treated.

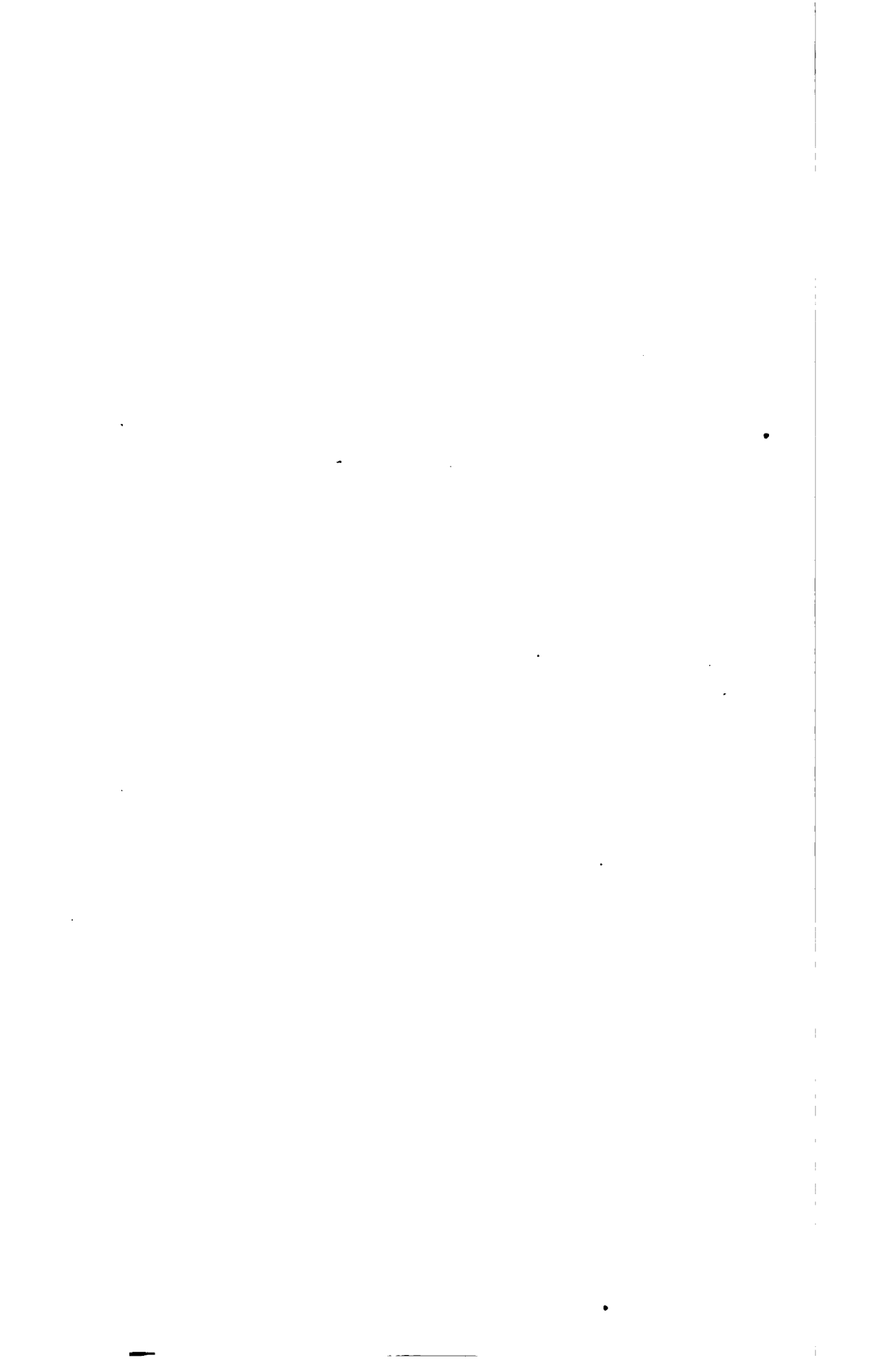
5. The bottoms of the pot stills are of cast iron and are hemispherical. The casting is about 2 inches thick. The cast-iron part is usually $8\frac{1}{4}$ feet in diameter at the top and $3\frac{1}{4}$ to $4\frac{1}{2}$ feet deep. The upper part, of steel, is of the same diameter and 6 feet high. Each still of this size has a water-cooled condenser of 225 feet of 4-inch cast-iron pipe, and charges about 2,400 gallons of oil. The time taken in charging, distilling, cooling, and cleaning is 24 to 30 hours. The bottoms of the stills, with minor repairs, last about a year. A new bottom can be put on without much difficulty.

The continuous stills are of the horizontal boiler type, usually 8 $\frac{1}{2}$ feet in diameter and 30 feet long. Their condensers are water cooled and provide about 850 feet of condensing surface.

6. The quantity of sulphuric acid used in refining amounts to about 3 per cent by volume of the crude oil. The strength of the acid is about 66° B. (93 per cent H_2SO_4). Some of the refineries have their own acid-making plants. Agitators for partly refined products are small; few, if any, charge more than 6,000 to 7,000

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✓
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gallons of oil. The crude distillate (green oil) is treated in larger agitators, which charge as much as 50,000 gallons. Agitation is usually by compressed air. Acid of proper strength and proportion is usually added in batches, or sometimes all at once, and agitated 15 minutes to an hour. Tar is allowed to settle for 2 to 18 hours, depending upon the grade of oil under treatment—the average being about 6 hours. The acid-treated oil is then run to soda agitators and similarly treated with a solution of caustic soda or soda ash. The oils are not water washed before distillation. Caustic soda is more commonly used than soda ash.

It is impossible to discuss in detail the quantity and strength of chemicals used in treating the different oils, since these factors differ for nearly every fraction. An acid tar from one treatment is often used as the first acid batch in the treatment of another oil. Also dilute acid recovered from the acid tars is frequently used in treating. The strength of the sulphuric acid used for various oils and in various stages of refining varies from 40 to 66° B. (48 to 93 per cent H_2SO_4).

About $1\frac{1}{2}$ per cent by weight of crude caustic soda, or its equivalent in soda ash, is used in treating and refining; thus about 11 pounds of caustic is used per 100 gallons of crude oil. Certain distillations, especially of heavy oils, are made over caustic soda. The strength and the quantity of alkali solution used in treating vary with the nature of the fractions under treatment; the specific gravity of the solution ranges from 1.01 to 1.38. Soda tars from one treatment are frequently used in treating another oil.

7. Acid tars are recovered, washed, and steamed to recover free acid. In this way 50 to 60 per cent of the acid used in making ammonium sulphate is obtained. The acid-free tars as well as the soda tars are used as fuel under the stills. The tars or sludges obtained in refining amount to about 16 per cent of the crude treated.

8. Just before distillation ceases in the coking stills a small amount of chrysene, a peculiar reddish product like resin, is obtained. It is not further purified except to float off water and oil adhering to it. It is usually considered a loss, but is sometimes used in laying wooden paving blocks.

9. Particular attention is paid to the recovery and treatment of the paraffin wax. The methods now generally used are largely the inventions of James Bryson, of the Pumpherston company, and the late N. M. Henderson, of the Broxburn Oil Co. (Ltd.).⁹⁵ The method varies somewhat in different plants but is essentially as follows:⁹⁶

⁹⁵ English patents 9557 (1884), 1291 (1887), and 11799 (1891).

⁹⁶ This method is followed all over the world generally by petroleum refiners who owe much to the Scottish oil-shale industry in this and other features of petroleum refining.

The heavy oil is cooled in tanks in open sheds and subsequently by freezing machines, usually of the ammonia absorption type, or compressors with anhydrous ammonia, operating with calcium chloride brine. Some coolers are directly cooled with anhydrous ammonia, notably those at the Pumpherston and Oakbank works, where compressors are used. Cooling takes place slowly to preserve the crystalline structure of the wax as well as possible. The chilled mass of paraffin, now a pasty mixture of wax and oil, is then pumped into filter presses, which are in refrigerated rooms. As a rule, low-pressure filters are used, so the wax cakes from these filters are then pressed in cloths in hydraulic presses, after which the wax is termed "hard scale." The oil separated from the wax is the "blue oil" mentioned on page 80. The scale is now melted and run on strainers of rather fine screen in long shallow trays. The wax is floated in the trays by water, but when it cools and solidifies the water is run off, leaving the cake of wax on the screen. The trays are built in special houses (sweating stoves), heated with steam coils, and then these houses are closed and sweating is begun. The temperature is raised so high that the oil and soft paraffin melt and run away from the harder wax, until the latter has the desired purity and melting point. The wax remaining on the trays is then melted by direct steam and run to filters, filled with boneblack or fuller's earth. The sweatings are rerun for softer or lower melting-point wax.

Henderson also designed a sweating apparatus which is said to be an improvement on the tray types. This apparatus consists of several vertical cylinders 9 feet high and 17 inches in diameter, within each of which is a cylinder 7 inches in diameter, open at top and bottom. The warm wax is run into the space between the two cylinders and floated on water until it has solidified. The water is then run off and the cylinder slowly heated as in the other types of sweaters.

In the early history of the Scottish oil-shale industry many processes were proposed and used for the recovery and treatment of wax. All these improvements were intended to insure better and cleaner recoveries and to lessen plant and labor costs. The Bryson type of coolers and Henderson's sweating stoves have practically superseded all other types. They are used in the United States and elsewhere under different names. Plate XV illustrates a typical installation of sweating sheds in a shale plant.

Much of the wax produced by the shale operators is made directly by them into candles, but the larger part is sold in bags or slabs to independent candle makers.



SWEATING STOVES OR HOUSES FOR THE PRODUCTION OF WAX.

Courtesy Scottish Oils, Ltd.

YIELD OF PRODUCTS.

Early in the history of the industry certain products were made and a market was created for them, and since then the operators have not striven greatly to make others. In fact, really little is known as to the possibilities of making products other than those now being made. With a definite market for their products, the operators strive to make a crude oil that will yield the maximum of these products at lowest cost. Conditions have been fixed for producing such a crude oil, and it is known that if retorting conditions different from these were applied a less satisfactory crude would be made, resulting in a smaller yield of refined products.

In 1919 the average yield of refined products from typical Scottish crude shale oil was about that shown in Table 15, following:

TABLE 15.—Yield of refined products from Scottish shale oil in 1919.

| Products: | Yield, per cent |
|---|-----------------|
| Naphtha (gasoline), including scrubber naphtha, maximum end point 450° F..... | 9.9 |
| Illuminating and burning oils..... | 24.8 |
| Gas and fuel oils..... | 24.4 |
| Lubricating oils..... | 6.6 |
| Paraffin wax ^a | 9.5 |
| Still coke..... | 2.0 |
| Loss ^b | 22.8 |
| Total..... | 100.0 |

^a The quantity of wax of various melting points recovered, amounts to about 17.7 pounds or 2.4 gallons per ton of shale, or 9.5 gallons of wax obtained from 100 gallons of crude oil.

^b Loss includes chrysene, tars, and permanent gases.

The products, marketed under various trade names, are:

1. *Gasoline and naphthas*.—These (sold in three or four grades) are usually motor spirits, cleaners' naphtha, and solvent naphtha.

2. *Burning oils*.—A good illuminating oil for domestic lamps is made, also oil specially prepared for burning in lighthouses, railway signal lamps, and marine and agricultural oil engines.

3. *Intermediate oils (gas and fuel oils)*.—In this class are oils used for fuel purposes and for making and enriching gas. Much of this oil is sold as fuel for Diesel and semi-Diesel engines.

4. *Lubricating oils*.—Usually lubricating oils of three different specific gravities are made, all of which have low viscosities. They are usually sold unblended, but are sometimes mixed with animal or vegetable oils.

5. *Paraffin wax*.—This is used in making various kinds of candles and tapers. It also finds a use in the manufacture of matches, explosives, waterproof coating, impregnated paper, etc.

Still coke.—In addition to the above products, still coke and is sold as a fuel. The coke from soda-treated oils is contaminated with soda salts (some distillations are made over soda solution) is usually not pure enough to be used for purposes other than requiring a high-grade fuel. Coke made in the primary distillation and in certain other distillations is pure enough for making electric-furnace electrodes and the like.

Soda and acid tars.—Inasmuch as these are used as feed for refineries, they can also be considered products, although they are not marketed. Formerly they were sold for grease making. Acid oil, often recovered in rerunning the lubricating fractions, is also used for grease making.

CHARACTER OF PRODUCTS.

In Table 16 is given the properties of some of the refined products from Scottish shale oil. Some of this information was obtained from operators, the remainder from tests made by the author or obtained from Scotland. For purposes of comparison, the distillation table of two typical American gasolines, meeting the requirements of the United States Government Committee on Standardization of Petroleum Specifications^a is included in the table. Particular attention is called to the homogeneity of the shale-oil products as shown by distillations.

TABLE 16.—Properties of Scottish shale-oil products.

| | Motor spirit. | Naphtha. | Burning oil. | Fuel oil. | Lubricating oils. | | |
|---|---------------|----------|--------------|-----------|-------------------|-------|--|
| Specific gravity..... | 0.720 | 0.742 | 0.799 | 0.863 | 0.899 | 0.893 | |
| Gravity, °A. P. I..... | 65.0 | 59.2 | 45.6 | 32.5 | 25.9 | 27.0 | |
| Flash point, °F..... | | | 103 | 240 | 360 | 334 | |
| Distillation ^b first drop, °C... | 52 | 71 | 150 | 226 | 270 | 228 | |
| Per cent: | | | | | | | |
| 10..... | 75 | 85 | 173 | 266 | 336 | 299 | |
| 20..... | 81 | 103 | 187 | 284 | 356 | 323 | |
| 30..... | 86 | 109 | 196 | 294 | 360 | 337 | |
| 40..... | 92 | 116 | 204 | 308 | 362 | 346 | |
| 50..... | 97 | 122 | 213 | 312 | 364 | 352 | |
| 60..... | 102 | 129 | 221 | 324 | 368 | 360 | |
| 70..... | 108 | 137 | 231 | 336 | 370 | 370 | |
| 80..... | 117 | 148 | 242 | 344 | 372 | 372 | |
| 90..... | 134 | 162 | 261 | 378 | 377 | 376 | |
| 95..... | | | 278 | | | | |
| Dry residue..... | 147 | 180 | 285 | 382 | 384 | 378 | |
| Viscosity..... | | | | | 215 | 140 | |
| Setting point, °F..... | | | | | 26 | 25 | |

^a Bureau of Mines method. See Tech. Paper 214, 1920, 33 pp. Samples meet Government specifications (1920).

^b Engler-Ubbelohde method on 100 c. c.

^c Losses not given.

^d Redwood at 70° F.—215 Redwood corresponds to about 255 Saybolt Universal and 140 mm. 150 Saybolt Universal. See also Table 15 (p. 88) for comparative data on viscosities.

Table 17 is quoted from a Scottish authority as representing properties of average samples of the usual products from Scottish shale oil.

^a Report of Committee on Standardization of Petroleum Specifications: Bulletin 19, 8 pp.

TABLE 17.—*Properties of average Scottish shale-oil products.**

| Product. | Specific gravity. | Flash point range, °F. ^b | Tint No. 600 glass Lovibond 2-inch cell. | Boiling point range, °F. ^c | Viscosity, seconds. ^d |
|-------------------------|-------------------|-------------------------------------|--|---------------------------------------|----------------------------------|
| Shale oil..... | 0.715 | | () | 120-300 | |
| Naphtha..... | .735 | () | () | 140-320 | |
| or water-white oil..... | .740 | | () | 180-380 | |
| | .785 | 106-120 | | 310-500 | |
| | .800 | 115-125 | 0.5 | 340-575 | |
| | .805 | 110-135 | .75 | 310-580 | |
| | .810 | 145-160 | 1.0 | 390-573 | |
| | .830 | 200-230 | 2.0 | 450-675 | |
| | .840 | 220-240 | 5.0 | 500-700 | |
| | .855 | 200-230 | | 450-710 | |
| | .870 | 275-300 | | 500-720 | |
| | .865 | 275-300 | 15 | 575-710 | 44 |
| | .875 | 300-310 | 18 | 575-800 | 47 |
| | .885 | 320-340 | 15 | 650-870 | 55 |
| | .895 | 330-350 | 16 | 720-875 | 70 |

* D. R., The oil shales of the Lothians. Part III. The chemistry of the oil shales: Memoir of the Geological Survey of Scotland, 2d ed., 1912, p. 191.

^b Apparatus—closed cup for light-burning oils, open cup for others.

^c By Engler-Ubbelohde method.

^d Viscosity—viscometer not specified. Companies in 1912 each had own standard viscosity pipette.

^e At temperature.

^f Less

half-inch cell.

NAPHTHA.

According to Steuart,⁸⁸ 60 to 70 per cent of the naphthas can be removed by treatment with fuming nitric or sulphuric acid. In regard, however, to the strength and quantity of acid used is so added that although all highly unsaturated compounds are removed, a bulk of the olefins remain in the finished products. This also applies to products other than naphtha. On testing a Scottish shale oil with spirit for unsaturation, by the method used by the Bureau of Mines,⁸⁹ the writer found that 22 per cent of the oil was removed by sulphuric acid (sp. gr. 1.84). Shale naphtha has been shown to be soluble in benzene and toluene.⁹⁰ B. Steuart¹ found the naphthenes, cyclo-tetramethylene, pentamethylene, and hexamethylene, in shale naphtha. Heusler,² in addition, found metaxylene and cumene in shale naphtha product.

The specific gravities of the shale naphthas range from 0.700 to 0.895 (70.6 to 57.2° A. P. I.) or more.

BURNING OILS.

The burning oils are colorless, especially those prepared for domestic kerosene lamps. The heavier oils may, however, become

Steuart, D. R., Work cited, p. 190.

Dean, E. W., and Hill, H. H., The determination of unsaturation in gasoline: Tech. Rep. 181, Bureau of Mines, 1919, 25 pp.

Steuart, B., Contribution to the composition of shale naphtha: Jour. Soc. Chem. Ind., 9, 1900, pp. 986-989.

Heusler, F., Über der Zusammensetzung der schottische Schiefertheeröle: Ber. Deut. Gesell., Jahrg. 30, 1897, pp. 2748-2752.

Still coke.—In addition to the above products, still coke is made and is sold as a fuel. The coke from soda-treated oils is contaminated with soda salts (some distillations are made over soda solutions) and is usually not pure enough to be used for purposes other than those requiring a high-grade fuel. Coke made in the primary crude distillation and in certain other distillations is pure enough to be used for making electric-furnace electrodes and the like.

Soda and acid tars.—Inasmuch as these are used as fuels in the refineries, they can also be considered products, although they are not marketed. Formerly they were sold for grease making. A *residual oil*, often recovered in rerunning the lubricating fractions, is used for grease making.

CHARACTER OF PRODUCTS.

In Table 16 is given the properties of some of the refined products from Scottish shale oil. Some of this information was obtained from operators, the remainder from tests made by the author on samples obtained from Scotland. For purposes of comparison, the distillation table of two typical American gasolines, meeting the 1920 specifications of the United States Government Committee on Standardization of Petroleum Specifications^a is included in the table. Particular attention is called to the homogeneity of the shale-oil products as shown by distillations.

TABLE 16.—*Properties of Scottish shale-oil products.*

| | Motor spirit. | Naphtha. | Burning oil. | Fuel oil. | Lubricating oils. | | American gasoline. ^a | |
|------------------------------------|---------------|----------|--------------|-----------|-------------------|-------|---------------------------------|-------|
| Specific gravity..... | 0.720 | 0.742 | 0.799 | 0.863 | 0.899 | 0.893 | 0.741 | 0.735 |
| Gravity, °A. P. I..... | 65.0 | 59.2 | 45.6 | 32.5 | 28.9 | 27.0 | 59.5 | 55.9 |
| Flash point, °F..... | 103 | 103 | 240 | 300 | 300 | 334 | 80 | 8 |
| Distillation b first drop, °C..... | 52 | 71 | 150 | 226 | 270 | 228 | 80 | 8 |
| Per cent: | | | | | | | | |
| 10..... | 75 | 95 | 173 | 266 | 336 | 299 | 87 | 74 |
| 20..... | 81 | 103 | 187 | 284 | 356 | 322 | 100 | 94 |
| 30..... | 86 | 109 | 196 | 294 | 360 | 337 | 110 | 106 |
| 40..... | 92 | 116 | 204 | 305 | 362 | 346 | 120 | 120 |
| 50..... | 97 | 122 | 213 | 312 | 364 | 352 | 129 | 134 |
| 60..... | 102 | 129 | 221 | 324 | 368 | 360 | 139 | 147 |
| 70..... | 108 | 137 | 231 | 336 | 370 | 370 | 152 | 164 |
| 80..... | 117 | 148 | 242 | 344 | 372 | 372 | 168 | 184 |
| 90..... | 134 | 162 | 261 | 378 | 377 | 376 | 188 | 210 |
| 95..... | | | 278 | | | | 204 | 225 |
| Dry c..... | 147 | 180 | 285 | 382 | 384 | 378 | 224 | 225 |
| Viscosity..... | | | | | 215 | 149 | | |
| Setting point, °F..... | | | | | 26 | 25 | | |

^a Bureau of Mines method. See Tech. Paper 214, 1920, 33 pp. Samples meet Government specifications (1920).

^b Engler-Ubbelohde method on 100 c. c.

^c Losses not given.

^d Redwood at 70° F.—215 Redwood corresponds to about 258 Saybolt Universal and 140 Redwood to 165 Saybolt Universal. See also Table 18 (p. 88) for comparative data on viscosities.

Table 17 is quoted from a Scottish authority as representing the properties of average samples of the usual products from Scottish shale oil.

^a Report of Committee on Standardisation of Petroleum Specifications: Bull. No. 1, 1919, 8 pp.

TABLE 17.—*Properties of average Scottish shale-oil products.**

| Product. | Specific gravity. | Flash point range, °F. ^b | Tint No. 500 glass Lovibond 2-inch cell. | Boiling point range, °F. ^c | Viscosity, seconds. ^d |
|--|-------------------|-------------------------------------|--|---------------------------------------|----------------------------------|
| Motor spirit..... | 0.715 | | () | 120-300 | |
| Spirit or naphtha..... | .735 | (e) | () | 140-320 | |
| Do..... | .740 | | () | 180-380 | |
| Special oil or water-white oil..... | .785 | 108-120 | | 310-500 | |
| Crystall oil..... | .800 | 115-125 | 0.5 | 340-575 | |
| No. 1 burning oil..... | .805 | 110-135 | .75 | 310-580 | |
| Lighthouse oil..... | .810 | 145-160 | 1.0 | 390-573 | |
| Mineral or marine sperm..... | .830 | 200-230 | 2.0 | 450-575 | |
| Mineral coals..... | .840 | 220-240 | 5.0 | 500-700 | |
| Gas oil..... | .855 | 200-230 | | 450-710 | |
| Gas oil, grease oil, cleaning oil..... | .870 | 275-300 | | 500-720 | |
| Lubricating oil..... | .885 | 275-300 | 15 | 575-710 | 44 |
| Do..... | .875 | 300-310 | 18 | 575-800 | 47 |
| Do..... | .885 | 320-340 | 15 | 650-870 | 55 |
| Do..... | .895 | 330-350 | 16 | 720-875 | 70 |

* Stewart, D. R., The oil shales of the Lothians. Part III. The chemistry of the oil shales: Memoir Geol. Survey Scotland, 2d ed., 1912, p. 191.

^b Abel apparatus—closed cup for light-burning oils, open cup for others.

^c Probably Engler-Ubbelohde method.

^d Viscometer not specified. Companies in 1912 each had own standard viscosity pipette.

^e Lowest temperature.

^f Colorless

^g One-half-inch cell.

NAPHTHA.

According to Steuart,²² 60 to 70 per cent of the naphthas can be removed by treatment with fuming nitric or sulphuric acid. In refining, however, the strength and quantity of acid used is so adjusted that although all highly unsaturated compounds are removed the bulk of the olefins remain in the finished products. This also applies to products other than naphtha. On testing a Scottish shale motor spirit for unsaturation, by the method used by the Bureau of Mines,²³ the writer found that 22 per cent of the oil was removed by sulphuric acid (sp. gr. 1.84). Shale naphtha has been shown to contain benzene and toluene.²⁴ B. Steuart¹ found the naphthenes, methyl-tetramethylene, pentamethylene, and hexamethylene, in shale naphtha. Heusler,² in addition, found metaxylene and cumene in the same product.

The specific gravities of the shale naphthas range from 0.700 to 0.750 (70.6 to 57.2° A. P. I.) or more.

BURNING OILS.

The burning oils are colorless, especially those prepared for domestic kerosene lamps. The heavier oils may, however, become

²² Steuart, D. R., Work cited, p. 190.

²³ Dean, E. W., and Hill, H. H., The determination of unsaturation in gasoline: Tech. Paper 181, Bureau of Mines, 1919, 25 pp.

²⁴ Steuart, B., Contribution to the composition of shale naphtha: Jour. Soc. Chem. Ind., vol. 19, 1900, pp. 986-989.

² Heusler, F., Über der Zusammensetzung der schottische Schiefertheeröle: Ber. Deut. chem. Gesell., Jahrs. 30, 1897, pp. 2748-2752.

more or less colored on exposure to sunlight. Shale naphthas and burning oils are characterized by a strong bluish fluorescence. From 30 to 40 per cent of the volume of the oil can be removed by fuming sulphuric acid.* The specific gravities of the burning oils range from 0.790 to 0.830 (47.6° to 39.0° A. P. I.) and flash points from 105 to 230° F.

INTERMEDIATE OILS—GAS OILS, ETC.

The intermediate oils are used extensively in gas making and for fuel in internal-combustion engines. The British Navy uses large quantities for fuel. These oils are also sold for cleaning purposes. Their specific gravity varies from 0.840 to 0.870 (37.0° to 31.1° A. P. I.).

LUBRICATING OILS.

The lubricating oils can hardly be considered viscous enough for high-duty work. They are sometimes blended with animal or vegetable oils, although most are sold unblended. The specific gravity of the lubricating oils ranges from 0.865 to 0.910⁴ (32.1° to 24.0° A. P. I.).

Table 18 below will be found of interest, as it gives the viscosities at different temperatures of various American lubricating oils particularly adapted for use in internal-combustion engines, compared with the viscosities of a Scottish shale-oil lubricating oil at corresponding temperatures.

TABLE 18.—*Viscosities of petroleum and Scottish shale lubricating oils at various temperatures.*

| Oil. | Saybolt viscosity at— | | |
|---------------------------------|-----------------------|--------|---------|
| | 40° C. | 60° C. | 100° C. |
| Eastern extra heavy motor | 1,325 | 451 | 106 |
| Eastern medium motor | 502 | 195 | 63 |
| Eastern heavy motor | 326 | 136 | 53 |
| Eastern light motor | 164 | 81 | 42 |
| Do. | 132 | 71 | 41 |
| Scottish shale | 140 | 68 | 43 |

PARAFFIN WAX.

Paraffin wax is made in various grades; the usual melting points are 130–132°, 125–127°, 118–120°, 110–112°, and 98–102° F.

CHEMICAL COMPOUNDS FOUND IN SHALE OIL.

Comparatively little is known of the chemical constituents of shale oil; however, its composition varies with the nature of the shale from

* Stewart, D. R., *The oil shales of the Lothians. Part III, The chemistry of the oil shales*: Memoir Geol. Survey Scotland, 2d ed., 1912, pp. 191–192.

⁴ Stewart, D. R., *Work cited*, p. 192.

which it was obtained and with the method by which it was produced. Inasmuch as shale oil is produced by destructive distillation of substances that undoubtedly differ in different shales, this is self-evident.

Some studies of the composition of shale oil have been made. In addition to members of the aromatic and naphthene hydrocarbons (p. 87), Gray^{*} found traces of phenol, ortho and meta cresols, xylenols, and guaiacol in Scottish shale oil. The amounts present were too small to warrant their commercial extraction. Several investigators^{*} have studied the basic nitrogen constituents of shale oil, and have identified pyrrol (C_4H_5N), pyridine (C_5H_5N) and several of its homologues, and members of the quinoline (C_8H_7N) and isoquinoline (C_9H_7N) series.

AMMONIUM SULPHATE.

The ammonia water recovered from the retort condensers and scrubbers is pumped through a heater, in which it is heated by the spent or waste water flowing from the still, and passes to the top of a tower still, which is circular in cross section and approximately 30 feet high. Details of the ammonia stills vary in different plants, but in general they have a series of cast-iron trays about 2 feet apart from top to bottom.⁷ Steam, at a pressure of 30 to 40 pounds, is admitted to the bottom of the tower and passes upward through sealed hoods, cast on the trays; the liquid, flowing downward, passes from tray to tray by overflow traps. The steam carries off the volatile ammonia; in addition, milk of lime is added about halfway down the tower to break up and set free the ammonia contained in fixed compounds from which it can not be removed by simple steaming. Sulphur compounds in the water also break down in the tower and form deposits of sulphur which must be removed from time to time. The spent water runs from the bottom of the still into a tank containing pipe coils, through which flows fresh ammonia water on its way to the still, thus the incoming ammonia water is preheated by the spent water from the still.

The ammonia gas from the still is passed into sulphuric acid, much of which is recovered from the acid tars of the refinery, as has been noted (p. 83). The vessels in which the sulphate is formed are large lead-lined tubs or saturators, or "cracker boxes." The gas bubbles into the acid through holes in a lead coil in the bottom of the

^{*} Gray, Thos., The phenols from shale oil: *Jour. Soc. Chem. Ind.*, vol. 21, 1902, p. 845.

^{*} Williams, G. C., On the presence of pyridine in naphtha from the bituminous shale at Dorsetshire: *Phil. Mag.*, 4th ser., vol. 8, 1854, pp. 209-212; On the volatile bases produced by the destructive distillation of the bituminous shale of Dorsetshire: *Jour. Chem. Soc.*, vol. 7, 1854, pp. 97-107.

Robinson, G. C., and Goodwin, W. L., New bases of the leucoline series: *Trans. Roy. Soc. Edinburgh*, vol. 28, 1879, p. 561; vol. 29, 1880, pp. 265 and 273.

Garrett, T. C., and Smythe, J. A., The bases contained in Scottish shale oil: *Jour. Chem. Soc.*, vol. 81, 1902, pp. 449-456; vol. 83, 1903, p. 763.

⁷ Cosacher, H. R. J., Personal communication, 1920.

vessel, and sulphuric acid constantly flows into the box. In the older plants the ammonium sulphate thus formed, $2\text{NH}_3 + \text{H}_2\text{SO}_4 = (\text{NH}_4)_2\text{SO}_4$, falls into a well at the center of the bottom of the vessel, and a steam ejector lifts it from the well into cars with perforated bottoms. The excess acid is drained from the cars, falls on a lead-lined table, and flows back to the saturators. The sulphate in the cars is then sent to storage bins, and, after standing a while to dry, is ready for the market. More modern practice evaporates the sulphate solution to saturation in vacuum pans; the precipitated salt is dried in a centrifuge, and all traces of adherent acid are neutralized.

The acid recovered from the refinery goes to separate saturators where ammonia is passed into it until the saturation point has been reached. Then the solution is drawn from the saturators, run to settling tanks for the removal of tar, and then run into the continuous saturators—where it serves to dilute the incoming fresh acid to the proper point—or is also concentrated to saturation in vacuum pans and dried in centrifuges.

The exhaust steam and waste gases from the saturators are passed to the retorts and utilized for the formation of ammonia from the shale. The spent water from the ammonia stills is pumped to the spent-shale dump and filters through it before being allowed to escape from the works.

Most works purchase their acid, but two of the companies have their own acid plants.

The ammonium sulphate produced by the older method, though not perfectly dry, commands a ready market. As mentioned above, in the most modern plants, the sulphate is concentrated in vacuum pans, washed, and finished in centrifuges, and is of very good quality. The sulphate is sold and rated on a basis of 25 per cent ammonia content. About 1,850 pounds of sulphuric acid (sp. gr. 1.72) is required to make a ton (2,000 pounds) of ammonium sulphate. From 50 to 60 per cent of this acid is recovered from the refinery acid-tars.

COST OF AN OIL-SHALE PLANT IN SCOTLAND.

It is difficult to estimate, with any degree of accuracy, the cost of an oil-shale retorting and refining plant at the present time because conditions are abnormal and no complete plants have been erected in Scotland for over 15 years. According to Scottish operators, however, a retorting plant, including shale breaker, tips, retorts, condensers, tankage for crude products, haulage for spent shale, ammonium sulphate works, and a part of the power plant, would cost in Scotland, under the conditions, prices, and exchange rates prevailing in 1919, at least \$5,000 per retort. As it seems that a retort

should be rated by daily output of oil rather than throughput of shale (p. 72), the investment cost of a Scottish retorting plant, complete as described above, is \$5,000 per 96 gallons of oil a day. (It is not safe to consider that the average oil production of a Scottish retort, under present methods of treating Scottish shales, is over 96 gallons a day.) The investment cost of a retorting plant is therefore approximately \$52 per gallon of oil a day, or about \$2,190 per barrel (42 gallons) of oil a day. This is much less than the average "well cost" per barrel of petroleum, but the cost of producing petroleum is much lower than that of producing an equivalent quantity of shale oil.

Therefore the writer calculates that a Scottish oil-shale retorting plant, capable of receiving shale from the mine, producing 1,000 barrels of crude oil and a corresponding quantity of finished ammonium sulphate in a day of 24 hours, and disposing of the spent shale would cost approximately \$2,190,000. A refinery capable of completely refining 1,000 barrels of crude petroleum a day would cost about \$1,000,000; a complete shale-oil refinery probably more. Thus a complete plant of the Scottish type capable of treating 1,000 tons of 42-gallon oil shale per day and making a complete line of finished products, including ammonium sulphate, would cost about \$3,190,000. This figure does not include cost of land for plant site, mine, mine equipment, opening up mine, land for spent-shale dump, town for workers, and the like. It does include complete retorting and refining equipment, ammonium sulphate works, and auxiliary buildings and equipment.

MARKETING.

Oil-shale companies in Scotland market their products in a manner somewhat similar to that of the petroleum companies in the United States. Most of the shale oil is marketed in the British Isles, particularly in Scotland. The oil-shale companies and many marketing concerns and large consumers own their own fleet of tank cars. The cars are much smaller than those commonly used in the United States. A part of the paraffin wax is exported and, on account of its high purity, is in good demand.

SUCCESS OF THE SCOTTISH INDUSTRY.

All of the oil-shale companies in Scotland have not been commercially successful. In 1871 there were 51 companies engaged in the industry, but in 1919 there were only 6, and in 1919 these were consolidated. The successful companies have been those that, efficiently organized and well financed, have been able to employ the best technical skill, utilize labor-saving devices, and work on a large

scale. The industry has been favored by several factors, such as the relatively low cost of labor, large yields of ammonium sulphate, high cost of competing petroleum products, and situation in a densely populated industrial region—near Glasgow and Edinburgh—where supplies were readily available and a market was close at hand.

From 1871 to 1916 the production of shale oil increased from 593,310 to 1,965,000 barrels, whereas the production of ammonium sulphate in the same period rose from 2,350 to 59,400 long tons. These figures indicate that increasing effort has been directed toward the recovery of ammonium sulphate. Undoubtedly the increased recovery of this product has been a large factor in the success of the industry in Scotland.

In spite of the favorable factors which undoubtedly aided the success of the Scottish oil-shale industry, even the larger companies have had a hard struggle to compete with petroleum products imported from Russia, Rumania, Mexico, Dutch East Indies, and the United States. Retorts were improved to produce a better oil and a higher yield of ammonia. Mechanical labor-saving devices were invented, developed, and installed, the refinery operations were cheapened and improved, and economies of many kinds were introduced. The chemicals used in refining were recovered; and the recovered tars, the removal of which had involved expense, became a source of profit as fuel.*

Even now the struggle for existence continues, especially as the lessened price of ammonium sulphate, because of its growing production from by-product coking plants, gas producers, etc., has increased the difficulties of the operators. Labor costs have increased and the prices of refined oils decreased. In 1919, when the shale miners struck for an increased pay equivalent to the scale awarded the coal miners in Great Britain, the companies suspended operations of some of the retorts until the miners returned to the same scale. Later, the shale companies conceded the new terms voluntarily. Finally, in 1919, four of the existing companies consolidated, as a subsidiary of the Anglo-Persian Oil Co., in the hopes that a centralized and more efficient organization would permit more economical operation. Also, arrangements were made to have the surplus refining capacity of the shale works handle crude petroleum, imported by the Anglo-Persian Oil Co., which should assist economic operation and tend to reduce overhead costs.

According to reports,^o in 1920 the amalgamation was expected to remove, for the present at least, all danger of failure of the Scottish

* Stewart, D. R., *The oil shales of the Lothians. Part III, The chemistry of the oil shales: Memoir Geol. Survey Scotland*, 2d ed., 1912, p. 187.

^o *Petroleum World*, vol. 17, No. 238, July, 1920, pp. 301-302.

industry. It has been possible under the consolidation to decrease the miner's time of employment from an eight to a seven hour day, and to continue the old scale of wages. As has been noted on page 55, it is probable that the industry as a whole could not have continued profitable operations had there been no reorganization. The shale-oil refineries are now reported to be operating partly on imported Persian petroleum; and although the amount of petroleum imported is increasing, it still appears profitable to produce shale-oil products under the conditions now existing in Great Britain. It should be noted that, though prices of petroleum products in the British Isles are much lower than they were during and immediately after the World War, they are still about twice as high as those of similar products in the United States.

Table 19 summarizes the financial statements of the four leading companies for the financial years 1916-17, 1917-18, and 1918-19. Points of note in the table are the small capitalization, fair profits and dividends, and strong working capital. No plants have been erected or materially enlarged within the past 15 years, and repairs in that time have probably been routine and not particularly expensive.

Under the reorganization, the Anglo-Persian Oil Co. (Ltd.), guaranteed to pay or advance the amount necessary to meet the dividends on the 7 per cent participating preferred shares of the new company, Scottish Oils (Ltd.), which were exchanged for the ordinary shares of the subsidiary companies, until December 31, 1922; any sum so paid or advanced is to be repaid from profits of the new company for subsequent years, after payment of 7 per cent on the preference and ordinary shares of the new company. The Anglo-Persian Oil Co. was to subscribe and pay in cash for the ordinary shares. For the year ending March 31, 1920, 7 per cent dividends were paid on both ordinary and preference shares of the new company, but since then dividends have not been paid on the ordinary shares.

Of the subsidiary companies, Broxburn paid $13\frac{1}{4}$ per cent in 1920, $7\frac{1}{2}$ per cent in 1921 and 1922, and 15 per cent in 1923; Oakbank paid $11\frac{1}{4}$ per cent in 1920, 15 per cent in 1921, 10 per cent in 1922, and 15 per cent in 1923; Young's paid $2\frac{1}{2}$ per cent in 1920, no dividend in 1921 or 1922, and 3 per cent in 1923; Pumpherston paid 30 per cent in 1920, 20 per cent in 1921, 10 per cent in 1922, and 20 per cent in 1923. The above dividends are on ordinary shares. As to preference shares, Broxburn, Oakbank, and Pumpherston have paid 6 per cent annually since 1920.

TABLE 19.—Financial statements of the four principal oil-shale companies Scotland.*

| Name. | Year. | Capital. | | | | | Bank
interest |
|--|---------|---------------------------|-----------------------------------|--------------------------|---------------------------|--|------------------|
| | | Authorized common shares. | Authorized preference shares. | Common shares issued. | Preference shares issued. | | |
| Broxburn Oil Co. (Ltd.). | 1916-17 | 235,000 at £1.. | 10,000 at £10 bearing 6 per cent. | 235,000 at 17s. paid up. | 10,000 | | |
| Do..... | 1917-18 | 235,000 at £1.. | 10,000 at £10 bearing 6 per cent. | 235,000 at 17s. paid up. | 10,000 | | |
| Do..... | 1918-19 | 235,000 at £1.. | 10,000 at £10 bearing 6 per cent. | 235,000 at £1 paid up. | 10,000 | | |
| Oakbank Oil Co. (Ltd.). | 1916-17 | 200,000 at £1.. | 100,000 at £1.... | 200,000 at 17s. paid up. | 100,000 | | |
| Do..... | 1917-18 | 200,000 at £1.. | 100,000 at £1.... | 200,000 at 17s. paid up. | 100,000 | | |
| Do..... | 1918-19 | 200,000 at £1.. | 100,000 at £1.... | 200,000 at 17s. paid up. | 100,000 | | |
| Pumphreston Oil Co. (Ltd.). | 1916-17 | 300,000 at £1.. | 150,000 at £1.... | 285,500..... | 150,000 | | |
| Do..... | 1917-18 | 300,000 at £1.. | 150,000 at £1.... | 285,500..... | 150,000 | | |
| Do..... | 1918-19 | 300,000 at £1.. | 150,000 at £1.... | 285,500..... | 150,000 | | |
| Young's Paraffin Light & Mineral Oil Co. (Ltd.). | 1916-17 | 175,000 at £4.. | | 113,202..... | | £128,800 at 5 per cent. | |
| Do..... | 1917-18 | 175,000 at £4.. | | 113,202..... | | £173,200 at 6 per cent.
£128,800 at 5 per cent. | |
| Do..... | 1918-19 | 175,000 at £4.. | | 113,202..... | | £173,200 at 6 per cent.
£128,800 at 5 per cent. | |

| Name. | Year. | Operating profit | Balance from previous year. | Total available | Dividends. | |
|--|---------|------------------|-----------------------------|-----------------|-----------------------------|--------------------|
| | | | | | Common shares. ^b | Preference shares. |
| Broxburn Oil Co. (Ltd.). | 1916-17 | £65,938 | £12,657 | £78,595 | £29,263 | £10,000 |
| Do..... | 1917-18 | 81,140 | 15,409 | 96,549 | 30,263 | 10,000 |
| Do..... | 1918-19 | 78,157 | 14,283 | 92,441 | 35,250 | 10,000 |
| Oakbank Oil Co. (Ltd.). | 1916-17 | 76,224 | 12,457 | 88,681 | 26,508 | 10,000 |
| Do..... | 1917-18 | 67,727 | 26,663 | 94,390 | 26,508 | 10,000 |
| Do..... | 1918-19 | 57,379 | 25,289 | 82,668 | 26,500 | 10,000 |
| Pumphreston Oil Co. (Ltd.). | 1916-17 | 202,458 | 17,931 | 220,458 | 114,200 | 10,000 |
| Do..... | 1917-18 | 196,207 | 22,501 | 218,708 | 114,200 | 10,000 |
| Do..... | 1918-19 | 190,816 | 30,508 | 221,324 | 114,200 | 10,000 |
| Young's Paraffin Light & Mineral Oil Co. (Ltd.). | 1916-17 | 99,816 | 6,408 | 106,228 | 22,640 | 10,000 |
| Do..... | 1917-18 | 111,094 | 10,380 | 121,454 | 27,168 | 10,000 |
| Do..... | 1918-19 | 107,894 | 10,554 | 118,448 | 27,168 | 10,000 |

| Name. | Year. | To reserve funds, etc. | For depreciation. | Forward to next year. | Current assets. ^c | Current liabilities. ^f | Net working capital. |
|--|---------|------------------------|-------------------|-----------------------|------------------------------|-----------------------------------|----------------------|
| | | | | | | | |
| Broxburn Oil Co. (Ltd.). | 1916-17 | £14,099 | £16,294 | £12,409 | £329,010 | £134,028 | £194,982 |
| Do..... | 1917-18 | 27,570 | 15,794 | 14,383 | 465,496 | 254,932 | 210,564 |
| Do..... | 1918-19 | 15,250 | 15,450 | 14,491 | 512,082 | 260,553 | 251,529 |
| Oakbank Oil Co. (Ltd.). | 1916-17 | 17,517 | 12,000 | 26,668 | 246,818 | 144,151 | 102,667 |
| Do..... | 1917-18 | 12,601 | 25,600 | 25,289 | 295,631 | 167,067 | 128,564 |
| Do..... | 1918-19 | 5,000 | 30,000 | 26,169 | 353,380 | 208,374 | 145,006 |
| Pumphreston Oil Co. (Ltd.). | 1916-17 | 55,339 | 15,000 | 22,501 | 505,588 | 293,426 | 212,162 |
| Do..... | 1917-18 | 50,000 | 15,000 | 30,508 | 869,076 | 545,288 | 323,788 |
| Do..... | 1918-19 | 44,000 | 18,084 | 36,059 | 886,498 | 495,922 | 390,576 |
| Young's Paraffin Light & Mineral Oil Co. (Ltd.). | 1916-17 | 42,421 | 25,000 | 10,380 | 314,436 | 127,223 | 187,213 |
| Do..... | 1917-18 | 46,732 | 26,000 | 16,554 | 559,798 | 265,915 | 293,883 |
| Do..... | 1918-19 | 41,732 | 30,000 | 10,548 | 630,123 | 326,521 | 303,602 |

* Abstracted from balance sheets. £1 equals \$4.86 normal exchange.

* 15 per cent on Broxburn and Oakbank (income tax free); 40 per cent on Pumphreston; 5 per cent on 6 per cent on Young's.

* 6 per cent.

* 4 per cent contingent on 6 per cent bonds.

* Cash position strong in all companies. Investments in war loans accounts for gains in assets.

/ Includes provision for imperial taxes.

DISTRIBUTION OF COSTS IN THE SCOTTISH INDUSTRY.

Table 20 shows the percentage of costs attributable (in 1919) to different items entering into the total cost of producing refined shale-oil products. The distribution as applied to retorting does not include interest, depreciation, or certain other fixed charges, which, because of the investment represented by the retorts, amounts to a considerable proportion of the total cost of retorting.

TABLE 20.—*Percentage distribution of costs in the Scottish oil-shale industry, 1919.*

| Total mining costs. ^a | | Per cent. |
|--|-------|-----------|
| Miners' wages, measurements, and extras | ----- | 36.67 |
| On cost (below) (dead-work expenses) | ----- | 13.33 |
| On cost (above) (dead-work expenses) | ----- | 7.50 |
| Coal | ----- | 7.50 |
| Stores, including rails, castings, and wire rope | ----- | 10.00 |
| Ties and mine timbers | ----- | 8.33 |
| Royalties (lordship) | ----- | 3.33 |
| Right of way | ----- | 1.67 |
| Overhead, general, and fixed charges | ----- | 11.67 |
| Total | ----- | 100.00 |

Total retorting costs.^b

Includes taking shale from where it is delivered from mine to the spent-shale "bing" and making crude oil, crude scrubber naphtha, and ammoniacal liquor.

| | Per cent. |
|---|-----------|
| Total labor (wages) | 42.86 |
| Total stores, including coal for producer gas and steam | 57.14 |
| Total ^c | 100.00 |

Total refining costs.

| | Per cent. |
|--|-----------|
| Coal | 26.67 |
| Chemicals | 33.33 |
| Wages (labor) | 20.00 |
| Stores | 3.33 |
| Total fixed, general, and overhead charges | 16.67 |
| Total | 100.00 |

Total ammonium sulphate production costs.^d

| | Per cent. |
|--|-----------|
| Wages (labor) | 11.67 |
| Stores, including lime | 5.00 |
| Sulphuric acid | 60.00 |
| Coal, general, fixed, and overhead charges | 23.33 |
| Total | 100.00 |

^a Pre-war costs of mining were 45 per cent of 1919 costs.

^b Pre-war retorting costs were 43 per cent of 1918 costs.

^c This distribution does not include overhead, general, and fixed charges, which the writer estimates would be at least 20 per cent of the total retorting cost.

^d Pre-war costs of refining were 43 per cent of 1919 costs.

Table 21 shows the quantity of products finished from 1 ton of shale of the quality being worked in 1919, and Table 22 the percentage distribution of the total cost of producing from 1 ton of shale the finished products shown in Table 21. The distribution shown in Table 22 does not include fixed, general, and overhead charges on retorting but does include these items in the costs of other operations.

TABLE 21.—Quantity of products finished from 1 ton of Scottish oil shale in 1919.

| Product: | Quantity. |
|---|-----------|
| Naphtha (450° F. end point), gallons..... | 2.45 |
| Burning oils, gallons..... | 6.13 |
| Gas and fuel oils, gallons..... | 5.88 |
| Lubricating oils, gallons..... | 1.72 |
| Wax, pounds..... | 17.70 |
| Coke, pounds..... | 3.62 |
| Ammonium sulphate, pounds..... | 35.70 |

TABLE 22.—Percentage distribution of costs of making finished products from 1 ton of Scottish oil shale in 1919.

| Operation: | Percentage of total cost. |
|-----------------------------------|---------------------------|
| Mining..... | |
| Retorting*..... | |
| Refining..... | |
| Production ammonium sulphate..... | |
| Total..... | 100 |

* Does not include fixed, general, and overhead costs of retorting.

APPLICATION OF THE SCOTTISH INDUSTRY TO AN AMERICAN OIL-SHALE INDUSTRY.

The only oil-shale industry of commercial importance in the world at present is in Scotland, where it has been in existence over 60 years. Operations in France and Spain are still conducted on a small scale, and the plants of Australia which receive Government bounty on oil produced are reported as gradually discontinuing work on oil shale. The Scottish retort and retorting and refining procedure have been undergoing improvements for over 50 years and have changed little in the past 15 years. This evolution took place under extremely practical conditions, especially as the Scottish operators were under the pressure of severe foreign competition. It is evident, therefore, that the present retort is the best apparatus known to the Scottish operators for their particular shale and their particular conditions. It seems equally evident that the United States, where an oil-shale industry is just beginning, should not fail to take advantage of the knowledge gained during the many years of Scottish practice and should endeavor to adapt

the mistakes made in the early stages of the industry in Scotland. This is said with full realization that shales and conditions in the two countries are dissimilar. The knowledge and practice of Scotland should be applied and tried in this country as far as they can be applied successfully, and from this stage our own experience should guide us.

The possible use of Scottish practice in this country is discussed on pages 117 to 124, but it seems proper to indicate here some lessons that may be learned from a study of the Scottish oil-shale industry and applied in part, at least, to the potential industry of this country.

From a technical standpoint, we find that—

(1) The character of the crude oil produced, and thereby the nature, quality, and quantity of its products, is influenced greatly by the conditions under which it is produced.

(2) Inasmuch as the margin of profit in oil-shale operations is small, every possible technical economy must be applied in the plants.

(3) The refining operation must be suited to the nature of the oil; petroleum-refining methods probably will not apply entirely, or well, to shale-oil refining.

With respect to business practice, it is particularly evident that oil-shale operations require—

(1) The best executive and technical ability; (2) a strong, economical, efficient, and highly organized control; (3) a large amount of capital, on which only a nominal yield can be expected; and (4) large-scale operations, conducted with the utmost regard for economic conditions and prevention of waste.

HISTORY AND PRESENT STATUS OF THE OIL-SHALE INDUSTRY IN THE UNITED STATES.

In 1860 there were 53 companies, mostly in the eastern part of this country, producing oil by the distillation of various kinds of bituminous substances.¹⁰ Many of the companies operated under licenses from the Young company of Scotland. The methods used were crude, and the materials treated ranged from bituminous and cannel coals to some true oil shales. The desired product was kerosene or "coal oil"—the term still surviving from the time when most of the kerosene used in this country was really derived from coal and the like. Some of these companies were just getting started when the American petroleum industry came into being and reduced the price of kerosene so much that "shale-oil" operations be-

¹⁰ Bacon, R. F., and Hamor, W. A., *The American petroleum industry*. Vol. 1, pp. 203-212.

Baskerville, C. E., *Economic possibilities of American oil shales*: *Eng. and Min. Jour.*, vol. 88, 1909, pp. 149-154.

came unprofitable, and the plants were abandoned or adapted for refining petroleum.

In 1914 Woodruff and Day¹¹ published the result of an investigation of the oil shales of the Green River formation in Colorado. That a mineral from which oil could be obtained by destructive distillation existed in parts of Colorado, Utah, and Wyoming was well known; but this report, followed by those of Winchester,¹² which have been widely quoted, gave rise to a remarkable interest in oil shales in the United States during 1916, and the beginning of an attempted development of an oil-shale industry. Since that year an increasing amount of work has been done on oil shale, but up to the time of writing (April, 1923) the only operations approaching a commercial scale are those being conducted by the Catlin Shale Products Co., at Elko, Nev.; an oil-shale industry can hardly be said to exist in the United States, except in the literature of promotion concerns.

Many retorts of various designs have been proposed, several patented, and a few have been erected, but most of these have been so small that they must be considered as experimental or demonstration plants. Many investigators and organizers are working in good faith, but the potential industry has suffered much harm by the fake promoter and his promotion companies. It is reasonable to say that over 100 companies have been organized, ostensibly for the purpose of developing oil shale in some way or another but really for the purpose of fattening the pocketbooks of the promoters.

It is not difficult to see the reason for this situation. The country, in the years 1918-1920, was in the height of a speculative fever. Many persons during the war had made their first investments by purchasing Government bonds and the like, and not having learned to differentiate between an investment and a speculation were easily convinced that their present investments should be changed into ones yielding a higher return; nor did they know the earmarks of a fake industrial stock. This speculative fever seems to be abating at present; nevertheless, the country is literally "wild over oil." In the popular imagination, anything connected with oil is a source of immediate wealth; and stock promoters dealing in oil-shale stock do not hesitate to make the prospect more dazzling by presenting all sorts of impossible estimates of assured profits in oil-shale operations.

¹¹ Woodruff, E. C., and Day, D. T., Oil shale in northwestern Colorado and northeastern Utah: U. S. Geol. Survey Bull. 581, 1914, pp. 1-21.

¹² Winchester, D. E., Oil shale in northwestern Colorado and adjacent areas: U. S. Geol. Survey Bull. 641, 1917, pp. 189-198; Oil shale of the Uinta Basin, northeastern Utah, and results of dry distillation of miscellaneous shale samples: U. S. Geol. Survey Bull. 691, 1918, pp. 27-55.



A. VERTICAL OIL-SHALE RETORT, COLORADO.



B. PLANT WITH INCLINED RETORT, COLORADO.



A. BATTERY OF VERTICAL RETORTS, NEVADA.



B. BATTERY OF VERTICAL RETORTS,
UTAH.



C. PLANT WITH VERTICAL RETORTS,
UTAH.

Such tactics can not but hinder the development of legitimate oil-shale operations. To those who are sincerely assisting the development of an oil-shale industry by laboratory research and by the attempted development of well-founded processes, well knowing that the chances at present are not strongly in favor of making an immediate return on the investment, much credit is due. They are the ones who are really building a firm foundation for an oil-shale industry in this country.

TENDENCY OF RETORT DESIGN IN THE UNITED STATES.

In the neighborhood of 75 proposed processes for the treatment of oil shale have come to the attention of the writer during the past five years. Small plants, mostly for experimental and demonstrative work, have been erected in various parts of the country. Two or three are in Hoboken, N. J.; one or more in Illinois and in Kentucky; one in Kansas City; several in Denver, Colo.; a half dozen or more in the region between Denver and Salt Lake City, Utah; two or three in Salt Lake City; one near Dillon, Mont.; and at least two in Nevada and in California. Plates XVI and XVII show the size and general appearance of typical oil-shale installations in this country. None of these plants can be considered as ever having operated commercially; except one or two they are not large enough to be considered commercial plants, in the sense of being able to produce shale oil in commercial qualities and quantities.

Study of the proposed processes and plants reveals a tendency to get away from the Scottish type of vertical retort; only two or three are similarly designed, and even these are modified, mostly in essential details.

OBJECTIONS TO THE USE OF THE SCOTTISH OIL-SHALE RETORT IN THE UNITED STATES.

The chief objections to the use of the Scottish retort in this country are (1) its low throughput per unit, with consequent large initial plant cost; (2) its design, for the purpose of producing a maximum quantity of ammonia from Scottish shale; and (3) the doubt that it will treat richer American shales, based largely on the operation in Colorado in 1916 and 1917 of a so-called modified Henderson retort. These objections should not condemn the Scottish retort for trial in this country, especially as the objections seem to have little or no basis in fact.

1. The capacity of the Scottish retort seems to be low, yet the experience of operators in Scotland, working for the maximum yields of refined oils and ammonia that are consistent with the highest economic return, would indicate that the shales must be heated slowly if good yields, both as to quantity and quality of finished products,

are to be obtained. In the Appendix (p. 175) the effects of different conditions of retorting are discussed, so it will suffice to note that the retorting conditions applied in Scotland are probably those that yield there the greatest profit per unit of shale. Under other conditions might, and probably would, yield a larger amount of crude oil; a higher recovery of finished products might be obtained; or more ammonia might be produced, but the net profit would probably be reduced. It is pertinent here to mention that higher yields of crude oil does not necessarily lead to largest production of the products of highest quality. (See p. 175.)

Altogether aside from the influence of slow retorting on the products, other factors are to be considered. The Scottish retorts treat shale in large lumps—except for a small amount of fines well mixed with the coarse material—and therefore the rate of heating is much slower than if fine material were used. It would hardly be possible to treat fine shale in a vertical retort, because of the difficulty of moving vapors. In any retort, when finely ground shale is distilled, especially if the retorting rate is rapid, a large amount of dust passes into the condensers, and what does not “hang up” in the condensers passes out with the oil. This makes necessary frequent cleaning of the condensers and removal of the dust from the oil before the oil is sent to the refinery. Even with the Scottish vertical retorts, dust troubles are not entirely avoided. Since large quantities of shale are treated, the cost of crushing is small. The cost of grinding the richer American shales will probably be relatively great. These are some of the advantages of the Scottish retorts which tend to offset its low capacity. (See also p. 121.)

The vertical retort offers many advantages over an inclined or horizontal type, inasmuch as passage of shale through and from the retort is easily insured by simple mechanical devices. Complicated conveying and discharge apparatus should be avoided as much as possible in retorts operating at the temperatures of, and under the conditions obtaining in, oil-shale distillation. For conveying shale, it is much cheaper to utilize the force of gravity than complicated mechanical devices prone to get out of order; and finally the slow rate of depreciation, infrequent repairs, and continued operation with occasional shutdown offset somewhat the high installation cost of the Scotch type of retort.

Retorts better than the Scottish type will probably be developed, or that type may be modified, but the writer sees so many favorable features in it, coupled with successful commercial operation for many years, that he believes it worthy of serious consideration and a thorough trial in America.

2. The objections to the ammonia-production features of the Scottish retorts are: (a) It is necessary to use steam in retorting, and

an additional condensing surface for condensing this steam; and (b) it is the general belief that American shales do not contain enough nitrogen to warrant the production of ammonia from them. Attention is called to the fact that steam serves many functions in the retort other than that of acting as an ammonia producer (p. 48), and its use is probably essential in vertical retorts in order to obtain oils of good quality. As to the nitrogen content of American shales, Table 5 (p. 30) indicates that they average well with Scottish shales. The writer does not venture to predict whether the recovery of nitrogen as ammonia from American oil shales will be profitable, but the point is that these shales contain comparatively large quantities of nitrogen.

3. In 1916 a small retort, said to be a modification of the Scottish Henderson, was erected near De Beque, Colo. Except for minor details, this retort was all that the modern Henderson is not. It was of cast iron with 1-inch walls, 15 feet high and tapering, 12 inches in diameter at the top and 20 inches at the bottom. A piece of 12-inch casing 8 feet long, set vertically at the top, served as a hopper, and one toothed roller at the bottom, operated intermittently by hand, was intended as a discharge mechanism. The retort was set in a brick furnace, not at all like that used with the Henderson retort, and was heated by combustion of the gases produced in retorting, supplemented by raw shale burned in a fire box. A small exhaust pump drew vapors away from the retort. Provision was made for passing steam into the bottom of the retort.

The retort was filled with rich, massive shale, and heated. As is probably true of many rich shales heated rapidly, this shale intumesced and adhered to the walls of the retort, stopping operations. From the results of work with this one retort the idea has arisen that Scottish retorts can not be used for rich American shales. Reference to the description of the Henderson retort (p. 68) shows that the retort under discussion was a crude attempt to imitate it. The test was by no means fair and should not be considered conclusive. One of the features of the Scottish retort is the fact that the shales are heated slowly and the products removed rapidly by the use of excess quantities of steam. As far as coking or adhering of shales to retort walls is concerned, experiments of the Bureau of Mines show that richness of the shales is not the property that determines extent of coking; some very rich shales show no tendency to coke. Little difficulty is experienced with many coking shales if they are heated slowly and the products removed rapidly as formed, especially with steam. It is believed, however, that certain coking shales will be troublesome in any type of retort.

Later tests with the retort mentioned above, somewhat modified, have given more favorable results, but even now it is no more than a crude attempt to imitate the modern Scottish retort.

In 1919, near Elko, Nev., a unit of four Pumphreyston experimental retorts was erected by a private corporation; the Bureau of Mines cooperated to the extent of assigning one of its consulting engineers to direct the construction and operation of the plant. The retorts proper were constructed in accordance with plans obtained from Scotland, but after a short period of operation were closed down.

An examination of the plant after it had been closed down showed that the brick parts of the retorts had separated from the iron parts, that vertical joints in the brickwork, mostly in the upper parts, had pulled apart, and that slag had formed in the lower portions of the brick parts of the retorts. These conditions seem to have been caused by (1) imperfect workmanship in parts of the construction; (2) probably too rapid heating to working temperature; (3) operation probably being forced in an effort to increase throughput, with resultant high temperatures and the formation of slag; and (4) too rapid cooling when the plant was closed down. Although repairs could have been made at relatively low cost, plans for putting the plant in shape came to nothing and it was finally dismantled.

Under the circumstances the abandoning of this project should not be taken to indicate that the Scottish type of retort is not suitable for treating American shales. Scottish operators stress the necessity of using specially selected fire clay in the manufacture of retort bricks and in building the brick parts of the retort, and also the danger of damage to the brickwork if the retorts are not heated slowly when first put into operation or after they have been down for repairs. It is also necessary to cool slowly when the retorts are being shut down after a period of operation.

GENERAL DISCUSSION OF TYPES OF AMERICAN OIL-SHALE RETORTS.

It has been noted that the present trend in the development of oil-shale retorts in this country is away from the Scottish type. The principal features of proposed American retorts may be briefly outlined by dividing them into several general types. Table 23 (p. 107) gives the processes which have been most commonly mentioned in connection with the development of an oil-shale industry in this country. The table also classifies the processes according to the classification given below.

1. HORIZONTAL RETORTS WITH CONVEYER SYSTEMS FOR MOVING AND DISCHARGING SHALE.

A. CYLINDRICAL RETORTS.

Either one or several superimposed horizontal cylinders are provided with conveyers of various kinds, usually Archimedian screws. If one cylinder is used, the shale is fed at one end, subjected to rising temperature on its travel through the retort, and discharged in a spent condition. If more than one cylinder is used, the shale passes in series through them and is subjected to a higher temperature in each. With this type of retort the vapors are often removed at several different points, on the theory that oils of different qualities are distilled from the shale at different temperatures, and that by taking advantage of this condition a fractional "eduction" will result, yielding refined or partly refined fractions direct from the shale. There is undoubtedly some truth in the theory, but the extent of such fractionation does not appear to be sufficient to be of practical value. (See p. 174.) In many tests what has seemed to be fractional "eduction" has really been fractional condensation, taking place either within the retort or in the condensers.

Retorts of this general type were tried in Scotland early in the history of the industry (p. 62) and were abandoned as difficult to control and producing an unsatisfactory oil. This does not necessarily indicate, however, that they will be unsatisfactory for American shales and conditions.

B. RECTANGULAR RETORTS.

Usually the rectangular horizontal retort consists of a flat hearth, heated from below, and covered with a hood or top that leads the oil vapors to the vapor mains. The shale, which is ordinarily crushed to a size that will pass a 10-mesh screen, is usually fed upon the hearth by a screw conveyer. The feed end of the hearth is generally the coolest part, and the temperature increases progressively toward the discharge end. The shale is moved across the hearth and discharged from the retort by a series of rabbles, rakes, toothed conveyers, or the like. Discharge is usually into a water seal. Provision is often made to secure whatever benefits there may be in the fractional "eduction" theory. Retorts of the Wedge ore-roaster type with only one hearth are included in this class.

2. INCLINED RETORTS WITH CONVEYERS OR AGITATORS.

The title given is nearly self-explanatory. In the various types proposed the retorts are inclined at different angles to the horizontal, ranging from 5 or 10 degrees to perhaps 45 degrees. In different

retorts, the cross section is rectangular, circular, semicircular, or a segment of a circle. Various types of agitators or conveyers are used to keep the shale in motion through and discharge it from the retort. These range from scrapers that oscillate from side to side and scrape the bottom of the retort to belt or bucket conveyers that actually carry the shale through and from the retort.

3. CONTINUOUS, EXTERNALLY HEATED VERTICAL RETORTS.

A. MODIFICATIONS OF SCOTTISH TYPE

At least two modifications of the Scottish vertical retort have been proposed. One is based on the Pumpherston retort, but has two separate chambers—the upper being heated by passing through it combustion gases from the lower furnace, and the lower being externally heated, like the Pumpherston. The two chambers, or retorts, are connected by a duct, normally closed by a manually operated valve. In the upper chamber the shale is preheated and dehydrated by direct contact with the hot combustion gases. Then the shale drops into the lower retort, where destructive distillation takes place. Steam is admitted into the lower retort, which may be of silica brick, and the discharge mechanism is similar to that of the Pumpherston retort.

Another retort, a modification of the Henderson, in commercial size will have about the same dimensions as the Henderson and have the same type of discharge mechanism. The modification consists of perforated vapor pipes extending into and through the upper or iron part of the retort; their purpose is to remove the distillation products as rapidly as they are formed and to gain the supposed advantages of fractional eduction.

B. OTHER CONTINUOUSLY OPERATING VERTICAL TYPES.

One inventor proposes a vertical retort made of two concentric castings heated from an inside and an outside flue. One casting is stationary and the other is rotary about a vertical axis. The outer surface of the inner casting and the inner surface of the outer casting are provided with rows of short horizontally placed lugs or shelves. The shale feeds in at the top on the lugs, from which it is scraped by the lugs on the moving cylinder, falls to lower lugs, and is finally discharged.

Another vertical retort consists of an inner and outer metal shell of zigzag section. The inner shell rotates. The vapors pass through it at several places into an inner vapor duct. The whole is placed in a furnace and heated externally.

Other vertical types are based on the principles of the Wedge or Herreshoff furnaces for roasting ores. In these types shale is fed upon an externally heated horizontal hearth, where it is moved by a series of plows or rabblers connected to a central rotating vertical shaft. The shale progressively falls from one hearth to a second, is scraped over it and discharged to a third, and so on, until finally discharged. Each successive hearth is heated to a higher temperature, and vapor lines usually lead from the level of each hearth.

Still other vertical types, continuously operating, discharge the spent shale from an externally heated metal or fire-brick retort into a furnace where the spent shale is burned. The fixed carbon of the spent shale is supposed to furnish with the fixed retort distillation gases enough fuel to conduct the distillation in the upper distillation retort. Steam may also be admitted into the upper retort. One vertical retort contains a vertical screw conveyer for moving and agitating the shale in the retort.

4. INTERMITTENT EXTERNALLY HEATED VERTICAL RETORTS.

A retort of this type consists of a vertical casting or refractory retort, closed at top and bottom by tight-fitting lids. The lower lid has, passing through it and removable with it, a vapor line that reaches to within a few inches of the top of the retort. This inner vapor line is perforated, and the vapors resulting from distillation by external heating are drawn in through the perforations and downward out of the retort by a suction pump placed somewhere in the gas main or condensing system. This arrangement is supposed to provide better transmission of heat and immediate removal of the products as they are formed from the zone of formation to a zone of lower temperature. This process may also be made continuous.

5. INTERNALLY HEATED RETORTS.

Several types of internally heated retorts have been proposed. Most of them are vertical; some work continuously and others intermittently. Heating is effected by passing superheated steam or fixed distillation gases into one end of the retort and taking them, with the products of distillation, out of the other end. The retorts sometimes are operated under considerable positive pressure. The condensable products are condensed, the gases scrubbed, and the fixed gases used as fuel in the gas or steam superheater. In one plan proposed a series of retorts of this type is operated, the hot gases passing into retort 1, out of 1 and into 2, and so on, as far as feasible. When the shale in retort 1 is spent, it is discharged and the retort is filled with fresh shale. Retort 2 then receives the gas first and retort 1 last.

Two or three proposed vertical retorts are internally heated by the combustion of the spent shale in the lower part of the retort.

Fixed distillation gases, burned with the spent shale within the retort, may be used as a supplementary fuel. Products of combustion pass upward through the retort in contact with the distilling shale. These retorts are usually designed to operate continuously, and steam may be used to aid the distillation.

6. HORIZONTAL OR INCLINED ROTARY CYLINDRICAL RETORTS.

Most of the rotary cylindrical retorts are more or less like a cement clinkering kiln, and are designed to operate continuously. They are externally heated, and are usually inclined about 15° from the horizontal. The retort is set in a furnace and rotated by gears driven by electric motors or steam engines. Shale is fed at the cooler, higher end and passes downward through the retort and out at the lower end. The rotation of the retort, sometimes supplemented by vanes or spiral baffles, causes the shale to move downward at a uniform rate. In some of the retorts provision is made for taking the vapors off at several points. A proposed type of this retort is 60 feet long and 30 inches in diameter.

7. MISCELLANEOUS TYPES OF RETORTS.

One inventor treats his shales in a digestion bath of hot oil, recovering light oils by distillation from the bath direct and heavy oils by removing the bath oil from the digester, separating the spent shale from it, and distilling the bath oil, which now carries the heavier products resulting from distillation of the shale. This process is designed to operate continuously, the refining of the shale oil furnishing the bath oil, a constant amount of which is kept in the system. Another inventor combines this principle with that of ordinary destructive distillation; a bucket conveyer carries the shale into and through the hot oil bath; the same conveyer carries the residual shale from the oil bath to another externally heated vertical retort, in which ordinary destructive distillation takes place.

Other miscellaneous types of retorts are: (1) Closed ovens in which cars containing raw shale are admitted and from which they are withdrawn when the shale is spent, the vapors being taken off through openings in the oven walls; (2) cylindrical retorts externally heated and given an oscillating motion during distillation; (3) one investor proposes to drill vertical holes into the shale deposits and set up intercommunicating passages between holes by exploding powder in the bottoms of the holes. It is then proposed to pass superheated steam down the central hole and recover the products distilled by the steam from the others.

This discussion gives only the barest outline of the more commonly proposed types of retorts for treating oil shales in the United

States. Possibly some have been omitted, as new ones are being proposed continually. Some of the proposed retorts can be rejected by the engineer as being constructed on basically unsound principles; and others, which appear to have favorable features, can not have their commercial value determined until they are operated in a commercial manner. As previously noted, few of the above types have been operated on a scale sufficient to give much of an idea of their feasibility under commercial conditions, and many of them exist only on paper or in the minds of their inventors.

TABLE 23.—Oil-shale retorts proposed for use in the United States.*

| Name. | Type. ^b | Inventor or owner. | Address. |
|---------------------------------|--------------------|---|--|
| Anderson..... | 1b | Anderson Shale Oil Co..... | 160 South Broadway, Denver, Colo. |
| Beam (American Continuous)..... | 1b | American Continuous Retort Co. | 340 Gas & Electric Building, Denver, Colo. |
| Bishop..... | 1b | Bishop, J. A..... | 1285 Lafayette Street, Denver, Colo. |
| Brown..... | 6 | Brown, H. L..... | 265 Washington Avenue, Newark, N. J. |
| Bussey..... | 5 | Bussey, C. C..... | Brooklyn, N. Y. |
| Catlin..... | 5 | Catlin, R. M..... | Franklin Furnace, N. J. |
| Chew..... | 3b | Chew, L. F..... | Eighteenth and Blake Streets, Denver, Colo. |
| Colorado Continuous..... | 3b | Krushnic, E. L..... | 421 Cooper Building, Denver, Colo. |
| Day-Heller..... | 3b | The Day Co..... | 414 Hobart Building, San Francisco, Calif. |
| Del Monte..... | 1a | Prevost, C. A..... | Southern Building, Washington, D. C. |
| Erickson..... | 3a | Rainbow Petroleum Products Co. | 419 Judge Building, Salt Lake City, Utah. |
| Fusion..... | 6 | Fusion Corporation (Ltd.)..... | Cleford, Cheshire, England. |
| Galloupe..... | 3b | Western Shale Oil Co..... | Grand Junction, Colo. |
| Ginet..... | 1a | Ginet, J. H..... | Temple Court Building, Denver, Colo. |
| Hedges..... | 6 | Hedges, E. E..... | University of Kentucky, Lexington, Ky. |
| Jenson..... | 1a | Jenson, J. B..... | 823 McIntyre Building, Salt Lake City, Utah. |
| Johns..... | 2 | Industrial Process Engineering Co. | Rialto Theater Building, St. Louis, Mo. |
| Jones..... | 3b | Jones, Frank..... | Salt Lake City, Utah. |
| Lesley..... | 6 | Lesley, R. L..... | 611 Pennsylvania Building, Philadelphia, Pa. |
| National..... | 6 | National Rotary Retort Co..... | 42 Root Building, Buffalo, N. Y. |
| Porter..... | 3b | Porter Process Co..... | 409 Symes Building, Denver, Colo. |
| Randall..... | 6 | Randall, J. W. H..... | 15 East Fortieth Street, New York, N. Y. |
| Ryan..... | 7 | National Oil Machinery Co..... | 1270 Broadway, New York, N. Y. |
| Scott..... | 3a | American Engineering Associates..... | 979 Woodward Avenue, Detroit, Mich. |
| Simpson..... | 5 | Simpson, L..... | 172 O'Connor Street, Ottawa, Canada. |
| Simplex..... | 2 | Mount Logan Oil-Shale Mining & Refining Co. | De Beque, Colo. |
| Stalman..... | 3a | Stalman, O..... | 319 Ness Building, Salt Lake City, Utah. |
| Wallace..... | 4 | Wallace, G. W..... | 412 Missouri Avenue, East St. Louis, Ill. |
| Wingett..... | 3b | Troy-American Petroleum Co..... | 1214 First National Bank Building, Denver, Colo. |
| Whittaker-Pritchard..... | 5 | Fuel Products Corporation..... | 110 West Fortieth Street, New York, N. Y. |
| Young..... | 2 | Young, A. V..... | De Beque, Colo. |

* This table includes the names of those retorts that have been most frequently mentioned in connection with the treatment of oil shale in the United States. Many others have been proposed. Few of those mentioned have reached the stage of successful demonstration.

^b Type classification refers to that of the foregoing text.

Anyone studying the various designs of retorts in connection with the history of the oil-shale industry of Scotland is struck by the fact that many of the types proposed have been tried and rejected by the Scottish operators as inefficient or impractical, or have been

replaced by better methods. This fact does not necessarily condemn them for use in treating American shales, which undoubtedly differ from the Scottish shales; but it is suggested that persons contemplating the erection of retorts in this country ascertain if the apparatus they propose to erect, or a type similar to it, has ever been tried elsewhere, and if possible determine the reason why it was not used.

Another important fact is that most inventors of oil-shale retorts are paying little attention to the principles of destructive distillation, and seemingly have little regard for the kind of oil or other products their retort will produce. The interrelation of retorting and refining are discussed elsewhere (p. 116), but it is apropos to point out here that the nature of the crude oil, the degree of refining it will require, and the quality and quantity of finished products that can be produced from it depend materially on the conditions under which the crude oil is produced. Many inventors, apparently trying to devise a retort with a large output of crude oil per unit of time and installation, are overlooking the fact that they must produce an oil of good refining quality. (See also p. 174.)

It is possible that different types of retorts or different retorting conditions may be required to produce crude oils containing maximum quantities of different desired products. It is also possible that different shales will require different types of retorts and conditions of retorting for satisfactory treatment. For this reason the fact that a certain type of retort does not prove satisfactory for a particular shale may not indicate that the retort will not handle satisfactorily a different shale.

ACQUISITION OF OIL-SHALE DEPOSITS ON PUBLIC LANDS.

Large oil-shale deposits on public lands were, in 1921, opened for lease by the Federal Government, under the terms of an act of Congress approved February 25, 1920, entitled "An act to promote the mining of coal, phosphate, oil, oil shale, gas, and sodium on the public domain," and known as the mineral leasing act.

The rules and regulations governing the issuance of such leases are given in General Land Office Circular 671, entitled "Regulations Concerning Oil-Shale Leases Authorized by the Act of February 25, 1920. (Public No. 146.)" Persons interested in this subject should address a request for this circular to the Commissioner of the General Land Office, Washington, D. C. If a lease is desired, application should be made to the register of the proper land office, and he will give the necessary instructions and supply the required forms.

The supervision of oil-shale mining operations is under the jurisdiction of the Bureau of Mines. A pamphlet entitled "Operating Regulations to Govern the Methods of Mining Oil Shale, Phosphate, Sodium, and Potash, Under the Acts Approved February 25, 1920 (41 Stat. 437), and October 2, 1917 (40 Stat. 297), and the Safety and Welfare of the Employees in Connection Therewith," has been issued by the bureau for the benefit of those interested and may be had upon application to the Director, Bureau of Mines, Washington, D. C., or to the registrar of the local land office.

Because of the many points of law involved, as well as conflicting claims, it is impossible to give definite rules to pursue in obtaining oil-shale leases, and persons interested in leasing and operating deposits on the public domain should submit their cases to the register of the local land office for individual consideration.

ACQUISITION OF OIL-SHALE LAND BY LOCATION.

Prior to the leasing act of 1920, oil-shale locations were made under placer regulations. The General Land Office has ruled that where such locations were in all respects legal and regular they may go to patent like ordinary mining claims. A few patents have already been granted to oil-shale locators. Other patents may be expected to be issued to those who have fully complied with the law and located their claims before the approval of the act of February 25, 1920. Certain shale lands are of course already owned in fee simple, and therefore it may be possible to acquire deposits from their legal owners.

Considerable caution should be used in purchasing oil-shale lands from alleged owners, as the titles are often not at all clear and many are now in dispute. Many land surveys in the shale districts have been only preliminary, and later surveys may be expected to change some boundaries. It is reported that there have been many cases of claim jumping and fraudulent entry in claiming oil-shale lands, particularly in Utah and Colorado. Therefore, the prospective purchaser of such land is advised to take particular care to ascertain that the title to the land is clear before he purchases. Purchasers are also warned not to deal with companies who claim they can obtain land under a special agreement or arrangement with the United States Government, as no such terms exist or would be allowed.

ECONOMIC IMPORTANCE OF OIL SHALES TO THE UNITED STATES.

The great economic importance of oil shales is that when the industry is properly developed the United States will have a new domestic supply of mineral oils, which can not be cut off in time of war, whether military or economic. Until the oil-shale resources are

developed, the comforting assurance may be had that here is the raw material for a new supply of mineral oils which, come what may, will always be ready to help meet the Nation's demands for oil, no matter what the world's petroleum situation may be. As so many have expressed it, our oil-shale deposits constitute a rear line of economic defense in assuring the United States a home supply of mineral oils.

The oil shales are of great economic importance and interest to the States in which they occur. Mention is made (p. 128) of the supply of miners and other laborers necessary to develop and operate a great oil-shale industry in this country. Most of these laborers will be brought into the shale districts from remote regions, will live in the shale fields, and with their families earn and spend their money in the same locality. The enormous capital to be invested in the shale industry will add to the wealth and taxable properties of the States in which the money is invested. Auxiliary industries will develop, which again will add to the population, prosperity, and wealth of the shale regions. Transportation must be furnished to the shale districts now not adequately supplied with it and this should help also to open up new mining and agricultural districts. Oil products may be relatively cheaper in the regions close to the shale districts than in other parts of the country. The availability of cheap fuels near the shale districts will do much to develop unused agricultural districts of great potentiality which occur close to the shale fields of the West.

With the possible exception of certain products, it is probable that shale oil will not, for a long time, enter into the country's foreign trade. It would seem that all the shale oil that can be produced in this country for a great many years will be consumed within the borders of the United States.

PROBLEMS OF AN AMERICAN OIL-SHALE INDUSTRY.

The oil-shale industry of the United States is still in an experimental stage, and it is to be regretted that much of the experimental work is not being conducted with regard to the problems that will have to be solved in commercial work. The production and refining of shale oil is a chemical manufacturing operation and should be directed and controlled by experts. It is a fact that at present the majority of those working on oil shale have little or no knowledge of the technique and the technical problems presented by the substances they handle and the possible products. The oil-shale industry is one to which much technical knowledge and skill will have to be applied before it can become a successful commercial enterprise of first importance.

MINING.

In an earlier section (Table 22, p. 96) a summary of the relative costs of the different operations of producing refined products from oil shale in Scotland was presented. The cost of mining the shale was more than half (53 per cent) the total cost of producing refined products. It is to be expected that with American shales and under local conditions costs may differ and the same relative distribution not apply. The figures from Scotland, however, should be of interest as a basis for cost considerations and should, in a way, indicate where the greatest efforts toward reducing costs should be made.

Many of our shales may be worked more readily than Scottish shales, but clearly the cost of mining will make up a large part of the total cost of producing and refining shale oil. Mining methods and the possible reduction of mining costs should, therefore, be given serious and thorough consideration. In all probability a saving of 10 per cent in the cost of mining will be much more important than one of 10 per cent in retorting and refining.

Oil shale is tougher than coal; most of the deposits are horizontal or nearly so; and those in the Rocky Mountain region usually outcrop high up in the walls of canyons and lie at altitudes of 5,000 to 8,000 feet. Most mining engineers believe that oil shale from such deposits will be mined by methods similar to those used in coal mining and at about a similar cost. The impression has been created that shale can be mined by open-cut or quarrying methods permitting the use of steam shovels. Although this is undoubtedly true of some shales in the Eastern States, there are probably few deposits in Colorado, Utah, or Wyoming where steam-shovel methods can be employed. In the latter region the workable seams are, as a rule, relatively thin, and are usually covered by comparatively great thicknesses of overburden. It is now believed that most of these oil shales will be mined by underground methods similar to those used in mining coal.

However, the thickness and richness of the western shale beds are accurately known only in a comparatively few places, although most of the seams seem to be rather uniform in thickness and oil yield. Comprehensive core drilling or some other means of determining the thickness of the various seams and of obtaining accurate samples of the geologic column in the shale formations will do much to determine the feasibility of various methods of mining. Core drilling was done by a few companies in the shale district near De Beque, Colo., in 1921 and 1922, but complete results of the work have not yet been made public.

Plans for most contemplated workings in the western shale fields consider that the shale will be mined by drifts entering from the out-

crop, and the retorting plant will be on the hillside between mine entry and valley floor or on the valley floor. Consideration has also been given to the possibility of placing the retorting plants on the tops of the mesas or plateaus and mining the shale by shafts from the mesa top. Shaft mining may often be cheaper than tunnel mining, especially when mining is conducted at a considerable distance from the outcrop. In this case the average underground haul to a centrally located shaft will often be so much less than that through a long drift that it will pay to install expensive hoists.

It is not probable that many western mines will be worked through shafts, at least not until most of the shale near the outcrops has been mined. Although the tops of the mesas provide good sites for works and towns, and ample space for dumping spent shale, which the valleys and hillsides do not always furnish, these advantages are offset by the probable difficulty of securing water supplies on the mesa tops, the severe winter climate and heavy accumulation of snow, and the difficulty of supplying transportation.

In many of the Eastern States the black oil shales overlies coal seams and are removed with the coal in open-pit or stripping operations. In such places, or in similar places where the shale lies close to the surface, it is probable that steam-shovel mining will be feasible. Most of the black shales of the Eastern States are not very rich in point of oil production, and it is likely that reduced mining costs will be largely offset by the necessity of mining more shale to get a yield of oil equivalent to that produced from a smaller quantity of the richer shales of the Rocky Mountain district.

It is expected that the shales will furnish good walls, and that the problem of supports will be no more difficult to solve than in coal mining. However, in this regard it may be well to quote from an Australian publication¹⁸ on the mining of some of the rich oil shales in the State of New South Wales, as similar conditions may perhaps be met in shale mining in the United States:

Both "longwall" and "pillar-and-stall" systems of extraction have been adopted; in the latter the pillars are subsequently removed. In the present working mines—Genowlan and New Hartley—the workings are on the "longwall" system. Here explosives are unnecessary for breaking down the seam; on the contrary, great danger is experienced from the continuous cracking and shooting of the kerosene shale as the roof settles on the face. The unyielding nature of the top holing, which consists of a band of hard cannel forming the top of the shale seam, affords no relief to the vertical pressure. Where soft bituminous coal forms the roof, it gives or crushes under stress, and thus to some extent relieves the enormous pressure of the superincumbent Coal Measures and overlying Hawkesbury series on the tough, semilelastic kerosene shale. Where lateral expansion is rendered possible by open spaces—

¹⁸ Carne, J. E., *The kerosene shale deposits of New South Wales: Memoir Geol. Survey New South Wales, 1903, pp. 84-85.*

i. e., working faces—the tension is so great that cracking and shooting are almost continuous as settling proceeds. Further release by holing is therefore attended with considerable risks to the miners, who are compelled to work behind breast boards and cover their eyes with wire-gauze guards as protection against flying fragments of keen-edged shale. The tendency of kerosene shale to conchoidal fracturing under stress at right angles to the stratification increases the development of sharp edges which cut like knives. Numerous accidents have resulted, some unfortunately attended with serious impairment of sight.

Similar phenomena have been noted by Mr. E. C. Andrews, geological surveyor, in connection with the “kicking” or “spitting” shale of the Baker’s Creek Proprietary Gold Mine, at Hillgrove, where an unfortunate miner was cut in two by a large flying fragment which had previously passed through a 3-inch by 2 inch scantling. The slightest disturbance of the apparent equilibrium between compression and expansion, caused by drill or hammer, in this instance also caused sharp-edged conchoidal fragments to fly off with extreme violence.

At Joadja, where the overburden is greatly less, and where also in the early workings soft bituminous coal formed both top and bottom holing, the tension of the face was so slight that no trouble was experienced from “shooting.” In the present workings under heavier cover, with the pressure unrelieved by a yielding coal medium on roof or floor, a tendency to “shoot” is being manifested. Explosives (gelignite) are used for breaking down, as it is also used at Genowlan, together with compressed powder, for “brushing” or deepening the roadways, the latter being used in the New Hartley workings.

None of the kerosene shale workings have been troubled with fire damp, naked light always being used, in which paraffin wax or heavy oil is burned. Scotch shale mining is similarly free from natural gases.

Ventilation in all cases is by air shafts and furnaces.

MINE GASES AND THE EXPLOSIBILITY OF OIL-SHALE DUST.

In an earlier section (p. 61) the author has pointed out that the Scottish miners have not been seriously troubled with explosive or poisonous gases in the shale workings, and that the dust produced in shale mining in Scotland is not inflammable or explosive. The preceding section also indicates that dangerous gases were not encountered in mining the rich Australian shales. These facts must not be taken positively to indicate that similar conditions will prevail in American shale mines.¹⁴

As to explosive gases, rich American oil shales give off a strong odor like petroleum when freshly broken, and the writer has been advised of one shale mine where, after each shot, gas smelling much like natural gas was given off freely and could be detected at a short distance from the mine entry for some time after a shot.¹⁵

¹⁴ Scottish shale mines are not altogether free from inflammable gases. In December, 1922, while two miners were making a regular weekly inspection of waste in a section of one of the mines in the Scottish shale fields where the pillars were being removed, gas, in some way not yet explained, became ignited, and though the explosion was not severe, the afterdamp resulting caused the death of the men whose lights had been extinguished, and who were therefore unable to find their way out. At the time the miners were carrying safety lamps which were afterwards inspected and found to be in perfect condition.

¹⁵ Jenson, J. B., Personal communication, 1920.

There is a possibility that oil-shale dust may present a source of danger in American shale mines, particularly where rich shale is being worked. At the Pittsburgh experiment station of the Bureau of Mines, tests of a Colorado oil shale, estimated by the writer to yield about 35 gallons of oil to the ton, led to conclusions¹⁶ that the oil-shale dust can not be considered highly inflammable. "The dust will probably not be ignited by the flame from an ordinary blown-out shot, but is fully capable of propagating an explosion once started." Similar tests of shales from this and other districts are now being made to determine whether the richness and mineral composition of an oil shale have important bearing on its explosibility.

It is recommended that miners working in rich oil shale take the same precautions that they would when working a gassy or dusty coal seam, until such time as they are certain that no dangerous conditions in the way of gas and explosive dusts are to be encountered.

The State of Colorado has already provided that the mining of oil shale will come under the jurisdiction of the State bureau of mines. At first, at least, it would seem that the State bureau will require the same precautions to be taken in shale mining as are observed in mining coal. The following¹⁷ indicates the attitude of the Colorado bureau, and also gives further information as to the possible hazards to be encountered in oil-shale mining:

MINING REGULATIONS.

The attention of all shale mine operators in Colorado is called to the fact that, under the existing laws a shale mine, like all mines and quarries, except coal mines, comes under the jurisdiction of the [State] bureau of mines; also a shale-retorting plant is a metallurgical plant and is under the same jurisdiction. It is the duty of the commissioner of mines to make such rules and regulations as are necessary, in addition to the statutes, to reduce the hazards of mining and metallurgical operations as far as circumstances permit and to safeguard in every possible way the lives and health of the miners and other workers. The mine inspectors are to see that the laws, rules, and regulations are observed and to make such recommendations as may be necessary to carry out the spirit of the law.

Any person or corporation starting operations is required by law to notify the bureau of mines so that the inspectors may not overlook any operating properties.

At this time it appears probable that the first shale mining on a commercial scale will be underground, using the same methods as in mining coal. Consequently the same hazards will be encountered and the same precautions must be observed as in coal mining. Open-cut mining or quarrying must be conducted under the same regulations as are observed in other quarries.

¹⁶ Leighton, Alan, and Lents, H. E., The explosibility of oil shale: Report of the Pittsburgh experiment station of the Bureau of Mines, 1920.

¹⁷ Lunt, H. F., Dalrymple, J., and Duce, J., The oil shales of northwestern Colorado: Colorado State Bureau of Mines Bull. No. 8, 1919, pp. 45-46.

In underground shale mines there is a possibility that inflammable gas will be encountered. There does not appear to be enough gas to be dangerous in the shale itself. There is, however, a considerable amount of inflammable gas in the underlying strata, as shown by numerous gas wells at De Beque and elsewhere. It is quite possible that this gas may find its way into the shale through cracks or minute fissures in the underlying rocks. If, in the course of mining, one of these fissures is tapped, the gas, being under pressure, will escape into the mine, forming an explosive mixture with the air, and if it comes in contact with an open flame or spark an explosion will result. To guard against this it will be necessary, after shots are fired, to have the mine inspected by a qualified fire boss before the employees are allowed to enter it.

Another source of danger, and one that is certain to be present, is the dust. Mining operations of any sort are conducive to the formation of large quantities of fine dust, which collects on the floors and irregularities of the walls of the workings. Shale dust is highly inflammable² and like coal dust, flour dust, or the dust of any other combustible substance, will, under certain conditions, form a dangerously explosive mixture with air. This explosive mixture may be ignited by the open flame of a miner's lamp or by the blasts of the explosives used to break down the shale. Coal dust is rendered innocuous by humidity, which renders it plastic and prevents it being held in suspension in the mine atmosphere. The necessary moisture is supplied either by the direct use of water, applied with a sprinkler, or by steam. In the latter case, in cold weather, the steam is used to raise the cold air entering the mine to mine temperature by means of radiators, and is then turned into the air to give it the desired humidity.

Where it is not practicable to use steam or water, coal dust is mixed with stone or adobe dust so that there is at least 65 per cent of the latter present in the mine dust, under which conditions it will not form explosive mixtures with the air. It seems probable that the latter method will have to be used in shale mining, as indications are that shale dust does not easily combine with water.

It will require larger quantities of explosive to break the shale than are used in coal mining, and the blasting will raise its temperature materially. It is very possible that the heat generated in blasting will be sufficient to cause a slight distillation of the lighter and more dangerous inflammable gases from the hydrocarbons in the shale. To remove such gases, as well as the smoke and gases from the blasting, will require an adequate and reliable supply of air, properly conducted to the working faces.

With the above-described conditions to be met, it will be necessary in order to secure reasonable safety, to have all blasting done by a properly qualified shot firer after the other employees have left the mine, to use only permissible explosives, to use only electric lamps underground, and to have a mine foreman who holds a first-class certificate from the [State] coal mining department.

CRUSHING OIL SHALE.

Oil shale must be broken or crushed before it is retorted, and the type of crusher required will naturally depend largely on the degree of fineness to which the shale is to be reduced. This in turn may depend largely on the type of retort used. As yet there is little in-

² The inflammability of shale dust may be shown by letting a handful of it trickle through a hot flame. The particles will ignite and give the effect of a miniature roman candle.

formation available on the most suitable size of shale for retorting, and the size may vary with different shales and with different retorts. (See p. 178.)

For crushing, or rather breaking, toothed rolls do the work satisfactorily in Scotland, but it will be remembered that pieces of shale, often as large as a building brick, are put in a Scottish retort, and that the average size of shale retorted is fairly large. Toothed rolls should be suitable in the United States if the shale is not to be reduced to very fine pieces, but most of the proposed retorting processes intend to use shale varying in size from half inch to 80 mesh. Reducing rich American shales to such small sizes is difficult, as the shale is tough and elastic and in the crusher tends to yield without shattering.

There is little real information available regarding the crushing of American oil shales, but so far, the hammer mill, ring pulverizer, and gyratory crusher seem to be favored, with toothed rolls or ordinary jaw crushers for preliminary crushing to sizes varying from 1 to 1½ inch. Ore-milling practice should be of assistance in solving this problem.

INTERRELATION OF RETORTING AND REFINING.

No special effort is made in this bulletin to discuss separately the problems of retorting and refining, as it can not be too strongly emphasized that these phases of shale-oil production are so intimately connected that one must be considered with the other.

Certain retorting conditions produce a high yield of oil, but the oil may be of relatively poor quality. Other conditions may give a smaller amount of crude oil, but of so much better quality that it will yield a larger quantity of refined products. Certain methods of retorting will leave most of the nitrogen in the spent shale, from which it can be recovered as ammonia or ammonium compounds if desired. Other conditions will cause more nitrogen compounds to appear in the oil, from which they must be removed in refining. Certain oils of poor quality can be greatly improved by preliminary refining. Relatively slight changes in conditions of retorting sometimes have important influences on the nature of the oil produced, and the result of a change may differ in degree with different shales.

The broad technologic problem to be considered here probably may best be stated as the determination of those conditions of retorting and methods of refining that will yield the greatest profit. Generally, this means those conditions and methods that produce at lowest cost the maximum quantity of products for which there is the greatest demand or for which the greatest demand can be created. Thus, the problem is partly economic and its solution may be different for each individual plant.

RETORTING PROBLEMS.

When retorting is considered broadly, the first question to be answered is whether Scottish practice can be adopted entirely for American shales and conditions; must it be modified, and how; or must it be abandoned entirely. The same considerations hold with respect to refining the shale oils. Scottish practice has never been given a fair trial on American shales, and there is much that would lead one to expect that the Scottish retort, probably somewhat modified, may be well adapted to treat many of our shales. These points have been discussed previously, so it is unnecessary to cover the ground again.

Some rich shales tend to intumesce in the retort when heated. Even Scottish shales tend to bridge and clog in the retort. The modification of existing retorts or the design of new types to overcome this difficulty is a matter that must be carefully considered.

An important field for study is to determine the best types of material for retort construction. This applies to refractories. When coke by-product plants were first introduced into this country from Europe, ordinary fire brick was used as refractory material in their construction, but American operators were not satisfied with the heat-conducting capacities of such brick and other materials were tried. Other refractories were utilized, some of which greatly increased the rate of heat transmission from the furnace to the charge in the retort, and the new refractories resisted abrasion better than ordinary fire brick. Oil-shale operators may expect to obtain much valuable information on this point from those who have applied such refractories to coke-oven construction, and the possibilities of using silica brick, carborundum brick, and the like in shale retorts seem to be favorable. As for the highly heated metal parts of the retort, consideration should be given to the possibility of using recently developed alloys and protected castings, capable of standing sustained high temperatures without deterioration.

The shale-plant engineer will desire information on various physical and chemical constants of the oil shale that his plant is to treat. Some of these constants are specific gravity, weight per unit volume when crushed to different sizes, specific heat, heat of conductivity, heat of combustion, and proximate and ultimate analyses. These factors naturally differ for different shales, but the Bureau of Mines, cooperating with the State of Colorado, has determined them for an average Colorado shale, and they are presented in the Appendix (p. 152). Knowledge of these factors is necessary, if best engineering principles are to be applied to oil-shale retorts and retorting practice. Likewise specific and latent heats of the crude oils and their fractions should be determined. Knowledge of the coefficients of expansion of the different oils will be essential.

Certain individual factors that influence retorting, and thereby the quantity and quality of products obtained, deserve careful study. Some of these factors, which are now being studied by the Bureau of Mines, cooperating with the States of Colorado and Utah,¹⁹ are—

(1) Rate of rise of temperature of shale; (2) maximum temperature to which the shale is subjected; (3) amount and temperature of steam or other vapors and gases, such as shale gas, carbon monoxide, and the like, used in connection with retorting; (4) fineness of the shale being retorted; (5) pressure under which retorting is conducted; and (6) time of contact of vapors with retort walls.

A summary of the results of the experimental work mentioned above is presented in the Appendix (p. 174).

After the best conditions have been determined experimentally, they must be applied to large-scale work, in so far as commercial and engineering considerations permit. In commercial practice, of course, absolute optimum conditions, experimentally determined, are not always applied; but they are approached as closely as economic limitations and mechanical considerations allow.

That the retorting of oil shales and the refining of shale oils are intimately interrelated has already been emphasized. The production of crude oil from oil shale is a simple operation, merely requiring the heating of the shale with exclusion of air and subsequent condensation of vapors; but the economic production of maximum quantities of the oils most valuable from the refiner's standpoint is not a simple but a complex and difficult problem; therefore the study of production of oil and other products, carried out under different conditions as outlined above, will not be complete until the products so obtained have been subjected to thorough refinery and chemical study.

It is quite conceivable, from a consideration of the principles of destructive distillation, that certain conditions of retorting will produce an oil which will be practically valueless from the refiner's standpoint. Many inventors who are seeking to develop retorts with extremely large unit throughout, apparently do not realize that the quality of an oil is as important as, and in some cases more important than, its absolute quantity. The quantity of crude oil produced is not nearly as important as the quality and quantity of refined products that can be made from the crude. Not only must the crude oil be of good quality, but it must be of uniformly good quality day after day. (See Appendix, p. 182.)

¹⁹ Gavin, M. J., and Sharp, L. H., A study of the fundamentals of oil shale retorting: Reports of Investigations, Bureau of Mines, Serial No. 2141, June, 1920, 4 pp.

PRODUCTS DESIRED FROM OIL SHALES.

The success of the Scottish oil-shale industry has depended primarily on the production of paraffin wax and ammonium sulphate. Conditions relating to ammonia production in the United States are discussed on page 120, and as to wax, the present condition of the wax market in this country is such that if any large amount were being produced from a new source, unless it were of exceptional quality, there might be difficulty in selling it at a profit. The demand for wax is limited, and American petroleum, even though its production gradually declines, should largely take care of that demand for many years without much difficulty. It must be realized that much of our petroleum contains considerable quantities of paraffin wax that are not now extracted simply because the demand for wax is insufficient to warrant increased production. If, however, as is almost generally expected, waxes of higher melting point will be obtained cheaply from American oil shales, there will undoubtedly be a demand for them.

During the past several years the petroleum products in greatest demand have been motor fuels, illuminating oils, lubricants, and fuel oils. There is also a growing demand for fuel distillates for gas making, for certain metallurgical operations, and particularly for heavy-oil engines, especially of the semi-Diesel and Diesel types. The Diesel engine has the highest thermal efficiency of all known internal-combustion engines, and as the more volatile motor fuels become scarcer and more costly it is natural to expect that the Diesel, semi-Diesel, and other types of engines that utilize fuels most efficiently will be used more and more.

The demand for illuminating oils is increasing very slowly, and no immediate appreciable increase can reasonably be expected. Lubricants will probably be supplied from petroleum for a great many years, since much of the petroleum and petroleum residuums now burned as fuel can be worked into lubricants when the demand for lubricants increases. Oil will continue to be used as fuel until its price considerably exceeds that of an equivalent amount of coal. The demand for motor fuels is, however, likely to grow at an increasing rate. Hence it seems likely that shale oil will be used first of all as a source of motor fuels and secondly as a source of fuel oil. Other products will probably be of secondary importance.

American shale oils will probably yield a relatively good percentage of volatile motor fuel, once refining methods have been developed; and seemingly a large percentage of the less volatile distillates from shale oils will make satisfactory gas oils and heavy-oil engine fuels. It is doubtful whether satisfactory domestic kerosenes for lamp use can be made from shale oil, except at a cost that will be

practically prohibitive for years, but probably the cheaper grades of illuminating and burning oils can be made at relatively low cost and without great difficulty. In recent years the tendency has been to decrease the volatility of fuels for automotive engines, and the design of engines is constantly being changed so they can use the lower grade fuels satisfactorily. If this tendency continues, as it probably will with higher prices for oils, automotive fuels made from shale oils may contain a large amount of what would now be considered an illuminating-oil distillate. This distillate probably could not be cheaply refined into a satisfactory illuminating oil, but it might be used as a motor fuel.

As is shown on page 182, the first step in the refining of shale oil will probably be to run the crude oil down to coke. For this reason little or no residual fuel oil will be produced from most shale oils. In subsequent distillations of partly refined products, however, residues may be produced that will be satisfactory fuel oils if they do not contain too much paraffin wax. Many of the heavy distillates that may be used as heavy-oil engine fuels or fuel oils may contain so much wax that it will have to be removed or the oil heated to a temperature of complete fluidity before transportation or use.

Since motor fuels and fuel oils will probably be the products in most demand and most easily made, the greatest effort in retorting and refining should be made to obtain them, particularly motor fuels. The market for them will undoubtedly be stable and the demand can reasonably be expected to increase steadily. Later on, as the price of petroleum becomes higher and more is known regarding the methods of refining shale oils, it may be possible to make a more complete line of products.

As yet it is not definitely known whether lubricating oils of high viscosity and durability can be made from American shale oils, but investigations by the Bureau of Mines are yielding hopeful results. (See Appendix, p. 156.) Although the demand for lubricating oils and lubricants generally is, and for many years probably will be, taken care of by petroleum, the possibility of producing satisfactory lubricants from shale oil should be given serious consideration, as sooner or later the petroleum supply may fail to such an extent that it can not supply demands for lubricants.

NITROGEN RECOVERY.

Oil-shale operations in Scotland have been successful largely because of the high recovery of ammonium sulphate, which has a ready sale as a fertilizer. Table 5 (p. 30) indicates that many American shales have a nitrogen content equal to or larger than that of the average Scottish shales; thus the possibility of producing ammonium sulphate or other ammonia salts from domestic shales is indicated.

Little information is available as to the form in which nitrogen exists in American oil shales, but results of investigations by the Bureau of Mines indicate that the nitrogen exists in such condition that it may be readily converted into ammonia by the action of steam on the shales at high temperature after the oil has been produced. Conditions elsewhere found favorable for producing ammonia will probably be favorable for producing ammonia from American oil shales; but conditions for producing maximum economic yields of this material from our shales are, of course, not yet definitely known, and can not be known until commercial shale plants are in operation.

Experimental work of the Bureau of Mines indicates that conditions of producing the oil from shale largely determine the amount of nitrogen that remains in the oil-spent shale to be acted on by the steam at high temperature. Rapid production from the shale—rapid heating and rapid removal of vapors—yields an oil containing a relatively high percentage of nitrogen and a spent shale with a relatively small percentage of nitrogen. Slow production of oil reverses this effect.

There is a possibility of recovering the nitrogen as nitrogen bases, such as pyridine, pyrrol, and the like, which may be of considerable value as preservatives, germicides, and insecticides, although there is little market for them at present; because of their evil odor they must, in any event, be removed from the refined-oil products before the latter are marketed. Dilute (10 per cent) sulphuric acid removes about 10 per cent of the volume of crude naphtha from Colorado shale oil produced by dry distillation, and nitrogen bases constitute practically all of this loss. It may be found possible to recover such nitrogen bases in a form suitable for commercial use.

If a means of retorting and refining can be devised that will prevent in retorting the formation of oils containing nitrogen, or will cheaply and easily eliminate them from the crude and its oil products, it may prove advisable to disregard the production of ammonia altogether, since this product is being made by new and cheap methods that may render its recovery from oil shale generally unprofitable. This may especially hold if, by neglecting ammonia production, retorting costs can be reduced or, in other words, net profits can be enlarged, a larger throughput of shale per retort obtained, and the oil-producing capacity of the retort increased. A possible solution of this problem is discussed in the Appendix (p. 179).

This consideration also suggests the possibility of treating the oil-spent shale in separate gas producers, independent of the retort, in which increase of ammonia recovery almost to the total theoretical amount should be possible, as well as production of considerable quantities of gas for fuel. The lower part of the Scottish type of retort is, of course, a gas and ammonia producer, but many engineers

are of the opinion that gas and ammonia production may be effected more efficiently and to better advantage by means of a separate apparatus entirely independent of the oil-producing retort. Some suggest that by this means as much or little of the oil-spent shale as desired can be treated for the production of fuel gas and ammonia. In any event, full consideration should be given to the possible effect of any of these operations on the quality of the shale oil produced.

EFFECT OF SULPHUR AND SULPHUR COMPOUNDS.

Oil shales, as a rule, contain small quantities of sulphur, and it is important that the crude shale oil contain a minimum of this element or its compounds. Sulphur compounds in the oil make refining more difficult and costly and may even lead to the rejection of the crude altogether for refining or fuel purposes. It will be necessary to study the distribution of the sulphur in products made from different shales, and if sulphur compounds are present in objectionable quantities, either in the oil or gas, efforts should be made to prevent their occurrence in these products or to insure their presence in forms easily removable. Practically all shales yield hydrogen sulphide when they are retorted; some produce large quantities. This gas is a dangerous poison, if present in large enough concentrations, and it may sometimes be necessary to guard against the possible effects on plant workers.

DUST IN THE OIL.

Whenever oil shale is retorted, a certain amount of dust is formed and carried over into the condensers with the gases and vapors. The amount of dust produced varies with the shale, its condition, and the method of retorting. The dust accumulates in the condensers and varying quantities are always found in the oil. If much is present in the oil, it must be removed before the oil is refined.

A study of retorting and the design of retorts will always take account of possible dust troubles, and provision should always be made for easy removal of dust accumulations in vapor mains and crude-oil condensers.

REFINING PROBLEMS.

As to the refining of shale oils, it has been noted that shale oils are usually highly unsaturated; that is, they contain certain unsaturated hydrocarbons together with nitrogen bases which must be removed in refining before the oil products can be marketed. The amount present will depend somewhat on the method by which the oil was produced. Certain of these compounds, probably the more highly unsaturated, cause the color of the products to darken slowly and the deposition of gums and resins renders the oils unsuitable for fuel and

illuminating purposes. This behavior is noted particularly in the lighter fractions of the oil and, as stated, is probably due to polymerization and oxidation of certain of the compounds. The removal of these objectionable bodies requires several acid and alkali treatments and redistillations in refining, and largely accounts for the high refining loss in Scottish shale-oil refineries.

In Scotland about 25 per cent of the crude oil, including still coke, is lost in refining, as against a maximum refining loss of about 7 per cent in completely refining petroleum in the United States. Petroleum, of course, contains a much smaller quantity of unsaturated hydrocarbons than shale oil; many petroleum products contain none or only a trace of them. In spite of the long experience and economical methods used by the Scottish operators in refining shale oils, the high refining loss can not be reduced in commercial practice. If ordinary American petroleum refining methods were used the refining loss would undoubtedly be considerably higher. It is to be noted that when there is a loss of this nature in refining oils, not only is there a loss of a certain amount of the oil, but this loss must be paid for, as it results from expensive chemical treatment, made necessary by the presence of objectionable compounds. If the crude oil originally contained less of these substances, a saving is made, both in absolute recovery of refined products and in lessened costs of treatment.

From the above discussion certain definite refining problems may be outlined:

1. Determination of the color and resin-forming compounds in the oil and methods of removing them without removing other harmless unsaturates.

In Scottish practice not all the unsaturated compounds are removed in refining. As a general rule, the presence up to a certain percentage of olefins in motor fuels is even desirable, and therefore some of the olefins may be permitted to remain in certain of the products, while the more objectionable compounds should be eliminated.

2. Recovery of acid and alkali from sludges obtained in treating the oils.

Recovery of acid from sludges is an established practice in many American petroleum refineries, but a practical means of recovering the more expensive alkalis has not been perfected.

3. Improved and cheapened methods of refining.

Refining of shale oil is at best an expensive operation. The feasibility of using continuous distilling and treating processes and the possibility of using less expensive treating reagents should be considered.

4. Use of sludges for purposes other than for fuel.

In Scottish practice the sludges produced in treating the oils amount to about 16 per cent of the crude treated. After the recovery of the acid from them, the treated acid and alkali sludges are burned under the stills as fuel. In most American petroleum refineries the sludges are also burned as fuel. It would seem that there are certain possibilities in connection with the reworking of these sludges to obtain other commercial products.

5. Hydrogenation or saturation of unsaturated compounds.

If the unsaturated compounds of the oils could be cheaply saturated, there would be a smaller refining loss and a production of more and higher grade finished products. Commercially, it has not been possible to hydrogenate mineral oils successfully, but a further study is advisable, especially in the case of oils so highly unsaturated as shale oils.

6. The effect of coking distillation on crude shale oils is discussed in the Appendix (p. 179). It may be noted here that the effect of a coking distillation on an oil produced by rapid retorting is to yield a distillate much like the crude oil made by slow retorting. Even an oil made by slow heating can be improved by a coking distillation. A thorough study of this effect will indicate to what extent it can be used in refining and is certain to have a decided influence on the methods used in producing the oil.

ECONOMIC CONSIDERATIONS.

Economic considerations must govern all oil-shale operations. As in most industries, it is not possible to obtain absolute optimum results in all the phases of retorting and refining. Operations will, naturally, be so conducted that the greatest net profits will be made on the whole operation of producing finished products. In each locality, with every shale and with each type of retort used, the most economic conditions for the production of the greatest profit should be determined. For example: At the present time ammonium sulphate commands a good price and nitrogen bases have no market whatever—that is to say, no market has been developed for them; hence, under present conditions, the practice might be to make as much ammonium sulphate as possible from the shales, if nitrogen recovery is considered at all. Under reversed conditions, it would be advisable to make as much of the nitrogen bases as possible. Again, the most valuable product obtained from Scottish shale oil is paraffin wax, but conditions of supply and demand might be such as to make advisable the leaving of as much as possible of the wax in the gas and fuel oils and the lubricating oils without unduly raising their setting points or cloud tests. Market conditions must be studied, and the total net value of products made and marketed

should be such that the shale and oil treating operations, as a whole, will yield the greatest possible profit.

CONSERVATION.

Many problems, especially some of those just discussed, are not peculiar to the oil-shale industry alone. A successful solution of some of these problems by oil-shale operators often may be applied to the refining of petroleum. It may be remarked that when petroleum is cheap much of it is wasted, and refining processes were not developed with the primary purpose of utilizing a valuable resource in as efficient a way as possible; but when petroleum becomes costly, and the Nation has realized that its natural petroleum resources are not inexhaustible, more attention will be paid to conservation and prevention of waste. It would have been better when the industry began to have regarded petroleum as a valuable and wasting national resource, to be utilized as efficiently as possible; but when a commodity is plentiful and cheap, little thought is given to the fact that in time it may not be so.

Our oil-shale deposits are large, but they are not inexhaustible. When the oil-shale industry first begins it may not be always feasible, as well as profitable, to introduce those refinements into the treatment of the shale and its products which would mean conservation of this natural resource, but the shale should not be wasted, and in considering the future the country should demand that it be efficiently utilized. To bring about this ideal condition much research and investigative work will be necessary to determine the products that can be obtained and the methods of producing, refining, and using them. By the proper application of work of this nature it should be possible to conserve and efficiently utilize the Nation's oil-shale resources very early in the history of the industry.

BY-PRODUCTS.

Discussion of conservation and of efficient utilization leads naturally to consideration of the possible by-products from oil shale. The term "by-products" used in connection with the oil-shale industry is commonly understood to mean products other than the ordinary hydrocarbon oils and waxes similar to those now obtained from petroleum.

It might be desirable, of course, to make by-products, if they are of sufficient commercial value. Much is heard regarding by-products, and it seems to be the general impression that because of them the industry will succeed. All kinds of substances have been included in the list of oil-shale by-products, such as vanillin, salicylic acid,

pepsin, rubber, perfumes, medicinal preparations, drugs, and dye-stuffs. For most of these "paper" derivatives there is a limited demand and consequently a limited production at present. It should be realized that if these products could be made from oil shale on a commercial scale, the possible output might far exceed consumption and demand; present high prices might decrease so much that further production would be unprofitable, unless a new market could be created for them.

In a large way, by-products are rather unstable things to serve as a foundation for a large industry, and the oil-shale industry, if it is to succeed at all as a national industry, must be of great magnitude. Putting it more plainly: A few small individual plants might possibly be operated at a profit because they made rare by-products, but it is a reasonable assumption that, when hundreds of large oil-shale works begin to produce, the once rare by-products will become plentiful and the market for them will be demoralized. This is only another view of the law of supply and demand which must be considered in connection with the oil-shale industry as with any other. It may be true that many rare products have been produced in an experimental way from oil shale, but there is a vast difference between the laboratory preparation of a substance and its profitable commercial production. Thus, it might be possible to produce sugar from garbage in the laboratory, but it is difficult to imagine that such a process would be a paying one commercially.

Oil shales can hardly be expected to produce products other than those that can be made from coal tar and petroleum, and it is not yet known whether they will yield appreciable quantities of products usually obtained from coal tar. Although the use of coal tar as fuel oil in the Eastern States, started during the war, continues to the present, there is a serious shortage of certain coal-tar products, such as creosote, and if materials of this nature can be recovered as a product or by-product in the oil-shale industry their production might be profitable. Traces of compounds such as phenols, cresols, etc., have been found in shale oil, but it is highly probable that to make from oil shales larger amounts of these products the method of retorting must be different from that for obtaining oils similar to petroleum. In other words, the conditions favorable for the production of substances commonly obtained from coal tar (if they can be made from oil shales at all) will not be those favorable for making products similar to those derived from petroleum, for which there will undoubtedly be a greater demand and a more stable market.

Oil-shale operators will do well to plan on making products for which there is a demand and a stable market. Such products are

likely to be those similar to the refined petroleum oils of commerce and possibly some of the by-products of the petroleum industry for which a market is well established. These products and by-products, mainly hydrocarbon oils similar to those produced from petroleum, are those which oil shales are expected to supply. If it fails to do this, the oil-shale industry can not hope to become one of much importance or magnitude.

Certain exceptions are possible in regard to the above statements. Nitrogen products may, and possibly will, be derived from American oil shales. In Scotland ammonium sulphate is not considered a by-product. Moreover, it is probable that by-products may be obtained from oil shale for which a stable demand can be created by creating new uses, or in the case of some, for which a small demand now exists, cheaper production or the expansion of demand might make their production profitable. Notwithstanding these considerations it seems that much of the by-product investigation and discussion is ill-timed, particularly as the industry is not fully informed as to methods of producing satisfactory crude oils and making marketable hydrocarbon oils from them. On these latter products the success or failure of the oil-shale industry will depend.

SCIENTIFIC RESEARCH.

In connection with the study of oil shale many investigations of a scientific nature should be made. Some of these are: The determination of the nature of the oil-producing material of the shales; the manner in which it decomposes in forming oil; the relation between mode of origin, geologic age, and character of products; and the chemical nature of the oil and gas produced from different shales under different retorting conditions. Such studies are often disparaged because of the failure to appreciate that pure research of this type often yields results of highest commercial importance.

ECONOMIC PROBLEMS.

The oil-shale industry, to be commercially successful, must make a return on the capital invested in it; and to attract capital must offer reasonable evidence to the prospective investor that it will give him a fair return on his investment. Some of the factors to be considered in this regard are technical and have been discussed in other sections. Other factors are economic but, from a broad viewpoint, the technical and economic considerations can not be wholly separated. It is not a function of the Bureau of Mines to investigate the economic side of the oil-shale industry, but a brief discussion of some of these phases may call attention to matters hitherto not always carefully considered.

LABOR SUPPLY.

The question of obtaining, housing, and feeding the labor necessary in oil-shale districts, when the industry begins to assume some magnitude, is one to which attention must be paid. In the Eastern States the problem of obtaining labor may not always be particularly difficult, but in the Rocky Mountain States it is a real problem, which is liable to be serious rather early in the development of the industry.

Local labor will undoubtedly be available at the start, but, as the industry grows, men will have to be hired, transported to the shale fields, and housed in a region not now well supplied with transportation facilities and for the most part sparsely settled. When the industry begins to reach the magnitude of the present petroleum industry, the required number of miners alone will be comparable with that employed in our coal mines (average of 750,000 men). The problem is not serious now, nor is it likely to be for some time, but the ultimate probability of drawing hundreds of thousands of men from other industries without serious industrial disturbance and organizing an effective working force from them is so fraught with economic problems as to deserve attention even now.

SUPPLIES.

Those who will manage oil-shale plants will have to consider the problem of procuring supplies necessary in the development and conduct of the industry. Iron and steel, refractories, chemicals used in refining, timber and explosives for mining, fuel and the like must be available or made available. These may necessitate the development of subsidiary industries as near as possible to the shale fields.

TRANSPORTATION.

Attention will have to be paid to transportation facilities and the transport of products from the shale works and of supplies to them. There are few railroads in the shale regions of the Rocky Mountain States, and transportation facilities as they are at present will serve for the development of only a small part of the shale resources. Much of the rich oil shale is many miles distant from railroads. The great Uinta Basin of Colorado and Utah, for example, is now served by a single narrow-gauge road, with Shay locomotives, that runs more than 60 miles to a transcontinental main-line connection. The grade on this line in places reaches a maximum of 9 per cent.

It is probable that most retorting plants will be erected as near the shale mines as possible, and that where railroad facilities at present are not too far from the deposits, spurs and branches will be built

to connect with the various plants. For other deposits new roads must be built. Those companies now operating or erecting small shale plants should consider the future of the industry and consider the securing of adequate transportation service for the time when the industry will become commercial.

One phase of transportation involves a technical problem: The chief product of an oil-shale retorting plant is crude oil, and if this oil can be carried by pipe lines from the retorting plants to central refineries on the railroads, one of the chief transportation problems will be solved. On account of the high proportion of solid paraffins in most shale oils, as a rule, they will not flow well below temperatures ranging from 85° to 100° F. However, if heating stations are provided at intervals along the pipe line, the line buried to proper depth and well insulated, and the oil forced through at proper velocity, it is probable that the oil can easily be pumped, even during the winter. Most shale oils are quite fluid at temperatures only a few degrees above their setting points.

It is recommended that the possibility of piping shale oil from groups of retorting plants at the mines to a large refinery on the railroad be given serious consideration, not only on account of the low cost of such transportation compared with shipment in tank cars, but because a large refinery can operate more efficiently and with smaller overhead and fixed charges per barrel than a small one. Also, such a refinery would be in an advantageous position for procuring supplies and marketing products. It is believed that a 20,000-barrel refinery, situated on a railroad and receiving oil by pipe line from 20 retorting plants, each producing 1,000 barrels of oil daily, could operate at a much lower cost per barrel and with greater efficiency than twenty 1,000-barrel refineries, each situated close to and serving a retorting plant producing a like amount of oil daily, and depending on railroad spurs and tank cars to get its products to the main line or even sending its refined oils through pipe lines to storage on the railroad.

The oil-shale districts of the Eastern States are more adequately served by railroad systems than those of the West, but in all of them due consideration will have to be given to the problem of getting marketable products to the consumers.

MARKETING OIL-SHALE PRODUCTS.

Under the best possible conditions, it will be many years, because of the capital and construction work required, before an oil-shale industry begins to assume the magnitude of the present petroleum industry in this country, and therefore many years before shale products are extensively marketed. In the next section (p. 135) it will

be noted that the writer does not believe that shale-oil products will ever be serious competitors of petroleum products. Shale oil may be expected gradually to replace petroleum as the petroleum supply fails. It seems likely, therefore, that shale-oil products, when they are marketed in considerable quantity, will be marketed through the present systems and under the conditions involved in marketing petroleum.

SOME FACTORS INFLUENCING PLANT LOCATION.

Some of the factors that should determine the situation of an oil-shale plant have been discussed in the previous section, but good engineering practice will involve consideration of other problems, such as position of plant with respect to mine, availability of dumping space for spent shale, water supply, layout and future expansion of plant, housing of employees, and sanitary requirements.

POSITION OF PLANT WITH RESPECT TO MINE.

The relative positions of the plant and the mine govern the conveyance of the raw shale to the retorting plant and will depend to a considerable extent on the topographic features of the ground on which the plant is to be constructed. In most of the western shale regions it is probable that for a long time the shale will be mined from outcrops on cliffs, high above the valley floors, and the plants will be erected in the valley, or between mine and valley floor, to facilitate gravity flow of waste. In this case chutes or surface trams undoubtedly will be used in conveying shale to the plant. Aerial trams may also be used. Types of all these systems are being employed in the Rocky Mountain region by the small plants now operating intermittently for experimental or promotion purposes. Surface trams or chutes will probably be most favored, as they are cheaper to install and operate than other systems. Later on, shale may be mined by shafts from the tops of the mesas, and the plants may also be situated there. (See p. 112.) In other parts of the country it may be necessary to haul the shale cars from the mine entry to the crusher bins, particularly where there is little natural slope to the ground. At such places electric locomotives or cable haulage may be used.

DUMPING SPACE FOR SPENT SHALE.

Scottish oil shale expands while being retorted, though most of the richer American shales contract somewhat. Therefore, the volume of waste to be handled will be nearly, if not quite as great as that of the shale put into the retorts. Some use for the spent shale

may be found but with large-scale operations it is not likely that a market can be established for spent-shale products sufficient to take care of any large proportion of spent shale.

On the assumption that a ton of crushed raw shale occupies 35 cubic feet and the volume does not change in retorting, a plant of 1,000 tons daily capacity will discharge about 476,000 cubic yards of spent shale in a year, equal to a cone about 580 feet in diameter and 145 feet high. (See Pl. XII for size of dumps at one of the Scottish works.) Provision must be made for disposal of this large quantity of waste at the lowest possible cost. In many districts the plant can be so placed as to discharge directly upon a steep hillside, but in others it will be necessary to provide conveying or haulage systems to move the spent shale to dumps. (See Pl. XIV.) This is an item which must be considered in selecting plant sites and calculating operating costs.

WEATHERING OF SPENT SHALE.

Many American spent shales when exposed to the weather rapidly disintegrate and settle down, probably because of their high content of calcium and magnesium oxides, and form a black sticky mud that would probably be washed away and deposited as silt during rainstorms. Shale operators should note this point, as it may call for special attention by them. The writer believes that it may be necessary later for those States in which oil-shale deposits occur to enact laws to enforce the disposal of spent shales in such manner that property at a lower elevation may not be damaged. J. J. Jakosky, of the Bureau of Mines, to whom the writer is indebted for data on experiments on the weathering of spent shale, is convinced that the weathering of such shale may present serious problems to oil-shale operators.

WATER SUPPLY.

At many places the problem of procuring adequate and reliable supplies of water for oil-shale retorts and refineries will be serious. The problem may not often be important in the Eastern States, but the shale fields of the West are in districts where rainfall is usually sparse and watercourses few and rather widely scattered. In some places it is probable that water can be developed by sinking wells. Some operators plan to bring water from the larger streams to the shale works, but at many places this will require extensive piping and pumping systems. In many districts a large part of the stream flow has already been appropriated for agricultural purposes, and further irrigation projects are planned, or can reasonably be anticipated, in potential farming areas near the shale fields or below them on the same river systems.

Retorting and refining operations will probably require from 150 to 250 gallons of water per ton of shale treated. Water may be used as steam in the retorts; probably water-cooled condensers will be used, especially in regions where seasonal temperatures range from 100° F. in summer to below zero in winter; boiler plants require water; refinery condensers must certainly be water cooled; water may be required in mines; and a good supply of potable water will be needed for domestic purposes by the large number of employees and their families. It seems certain that in many plants the water will have to be carefully conserved; exhaust steam will be condensed; condenser water will be cooled in towers or ponds and reused; and heat exchangers will be employed partly to take the burden of cooling from the condenser water. It will probably be advisable at most plants, at least in the Western States, to use electric power transmitted from hydroelectric plants on the larger rivers to the shale works and mines. Possibly excess combustible gases from the retorts, or producer gas made from raw or spent shale, can be used in internal-combustion engines to generate power at the plants. There is also a chance that at least part of the water required in the works can be obtained from the shales themselves. Many shales when heated yield 30 gallons and more of water to the ton, and if this water is pure enough it may be used for boilers and condensers.²⁰

PLANT LAYOUT AND FUTURE EXPANSION OF PLANT.

A shale retorting plant and oil refinery should be arranged for economy of space and operation. Provision should be made for expansion, and the frequent error of petroleum-refinery engineers in failing to allow for plant enlargement should be avoided from the start. Naturally, the first shale plants will be comparatively small, but the assumption is that they will grow, and extensions should not be haphazard. A convenient, logical, and economic flow sheet should be designed at the very beginning and allowances made so that enlargements of the plant will permit conformation with the same flow sheet. Many engineers do not realize the extent production costs are lessened by convenient internal arrangements of a plant which afford logical and orderly flow of raw materials to the works, flow of products in the course of manufacture from stage to stage, and output of finished products and waste materials. Work of this type calls for high engineering skill. An industry that can hope for only a relatively small margin of profit can not pay too much attention to this point.

²⁰ Jakosky, J. J., Uses of water in the oil-shale industry, with particular reference to engineering and sanitary requirements: Tech. Paper 324, Bureau of Mines, 1923, 57 pp.

This matter makes necessary the consideration of the position of the retorting plant with respect to mine, railroad, refining plant, and storage facilities. At many places the ideal condition of having mine above retorts, retorts above refinery, and refinery above storage can be realized. This will allow gravity flow of raw, finished, and waste material, and construction can be planned so that subsequent enlargements will not disturb the efficiently planned flow sheet.

HOUSING AND SANITATION.

It seems proper here to discuss the problems of housing for employees and of camp sanitation.

Industry as a whole is rapidly appreciating that a contented worker means an efficient worker. Nothing assists more in bringing contentment than comfortable and clean living quarters and sanitary and wholesome living conditions. The point has already been made that labor will have to be brought into many of the shale fields from remote points and into a sparsely settled district. Conditions must be made attractive to the workers if it is hoped to retain their services for any extended period. Living quarters should therefore be made comfortable, clean, and attractive; well-constructed and inviting homes should be available for the married workers; wholesome and substantial food should be provided; and mess houses should be well lighted and clean. Opportunities should be afforded for wholesome recreation and amusement.

Camp sanitation should receive careful attention. Water supplies may often be scanty, and at such times particularly proper sanitation will require utmost watchfulness. Disposal of waste and sewage must be carefully considered not only for the benefit of the individual camp but for the protection of others at a lower elevation along the same stream or lower down the same valley.²¹

THE FUTURE OF THE OIL-SHALE INDUSTRY IN THE UNITED STATES.

The future of the oil-shale industry in this country depends primarily on the relative supply of and demand for petroleum products, particularly in the regions remote from seaboard. As already indicated, there is good reason for believing that within the next several years the domestic production of petroleum will decrease, the demand for its products will increase, and oil shales can well make up for the deficiency of crude petroleum as a source of refined mineral-oil products.

²¹ The subject of sanitation in oil-shale camps is fully discussed by Murray, Arthur L., *Sanitation in planning and developing oil-shale camps: Reports of Investigations, Bureau of Mines, Serial No. 2265, July, 1921, 7 pp.*

No one expects the oil-shale industry to spring into being overnight as a full-grown industry of national importance and comparable with the present petroleum industry. Even under the most favorable conditions its development must be slow, although this can be hastened by the employment of highly trained specialists, the proper kind of experimental work, and sincere cooperation and mutual helpfulness among the oil-shale operators.

There are certain practical limitations to the rapid development of the oil-shale industry that should be mentioned. M. L. Requa, former director of the oil division of the United States Fuel Administration, has said ²² that the oil-shale industry developed to the scale of the present petroleum industry "would require a mining activity comparable in size to the coal-mining industry." George Otis Smith, Director of the United States Geological Survey, speaking of the possibilities of the oil-shale industry, makes the point that "plainly our country can not afford to support another such army of workers [as that of the coal-mining industry], until we reach another stage in our industrial development." ²³

Large sums of money will have to be invested before the oil-shale industry becomes one of important commercial consideration. It is probable that the investment necessary for an oil-shale retorting and refining plant will approximate \$3,000 per barrel of shale-oil daily capacity. The present annual domestic output of petroleum is over 700,000,000 barrels. To replace this production with shale oil would require over 1,900 shale retorting plants, each putting through, every day in the year, 1,000 tons of shale yielding 42 gallons of oil per ton. The total quantity of shale mined would be over 700,000,000 tons, which is more than the coal production of the United States in 1920. The investment for retorts and refineries alone for an industry of this magnitude would be over \$5,000,000,000, and that figure does not include estimated cost of lands, opening up and developing mines, or transportation and marketing facilities; neither does it include the cost of developing subsidiary industries, without which a shale-oil industry could not exist.

COMPARISON OF THE OIL-SHALE WITH THE PETROLEUM INDUSTRY.

The investment necessary to carry on an oil-shale industry with an output sufficient to replace the present domestic output of petroleum would probably equal the capital already invested in the petroleum industry, at present a more attractive field if total costs

²² Requa, M. L., *The petroleum problem: Bulletin of the Union Petroleum Co., Philadelphia*, 1920, p. 26.

²³ Smith, G. C., *Where the world gets its oil: Nat. Geog. Mag.*, vol. 27, February, 1920, p. 197.

of oil production in both industries are compared. Profits are often large and rapidly acquired in the petroleum industry, whereas only a conservative profit can be expected from oil-shale operations, at least for a great number of years. Large interests will not invest heavily until convinced that oil-shale operations will be attractively profitable. On the other hand, once a shale-oil industry becomes profitable, profits are assured, subject to nothing more than ordinary business risks. There will be little of the spectacular in the oil-shale business, but ultimately it will be a conservative industry, in which investments will yield moderate though stable returns. Statements in this nature have been disputed by many of the present-day oil-shale enthusiasts, but the writer has seen no reliable estimates or figures that would cause him to alter his views.

It must be understood, however, that the industry will undoubtedly grow slowly, and while growing it will become more successful. One plant will begin operating at a fair profit, after which capital can be more readily attracted. One well-situated plant, with favorable markets and properly financed and managed, may begin to make money in a relatively narrow marketing field; whereas others, not so well favored, may have to wait years for profits.

The oil-shale industry is not strictly comparable with the petroleum industry; it is still embryonic in this country and calls for much new training and experience. Obtaining oil from shale is not comparable to obtaining petroleum from a well. In drilling an oil well the element of chance has to be taken into consideration, but there is always the possibility of large immediate reward on a relatively small investment. There should be no such element of chance in producing shale oil under proper management. In the first place, a few well-placed drill holes on the land selected will make possible the estimation of the thickness and quality of the deposits. After that, the production of crude shale oil becomes a technical problem, that of producing a crude product from a low-grade ore. The production of oil from shale requires one more operation than does the production of petroleum. When an oil well is driven into the oil sand, the crude product is recovered with but relatively little cost; but oil shale must be mined and afterwards treated to produce the crude oil. The quality of a petroleum is what it happens to be when the driller strikes the sand; the quality of shale oil is largely dependent on the process used and the conditions of manufacture. After the crude shale oil has been produced, its refining is more complex and more costly than the equivalent refining of petroleum.

Ultimately the oil-shale industry will be developed on as safe and sane a basis as other great manufacturing and mining industries. It is clear, however, from experience in other countries, and from the

very nature of the mineral and its crude products, that the industry can come to commercial importance only as a large-scale mining and manufacturing industry handling a low-grade raw material, and that the profits derived from it will be of the order of those derived from other industries of the same general type.

It is hardly believed that shale oil, considered in a large way, will be a competitor of petroleum; it is more likely to be a slowly developed successor to petroleum. A general impression has arisen that the larger oil companies are seeking to retard development of the oil-shale industry because they fear the possible competition of shale oil. There can be little truth in this belief. The writer believes that the petroleum industry, as a whole, welcomes the advent of shale oil, as its leaders are fully aware of the serious situation the American petroleum industry is facing. The larger, and many of the smaller, companies are already considering the time when they will be producing shale oil or handling its products. Many have already acquired oil-shale lands. Competition of shale oil with petroleum can not be regarded seriously, since, because of the many technical problems that must be solved in connection with the former and the large amount of capital that will be required before the industry can hope to compare in production with the present petroleum industry, shale oil will probably be needed long before it is produced in quantities sufficient to relieve appreciably the impending shortage of petroleum. It will be much longer, as above noted, before the industry can go far in supplying the present demand for petroleum, which normally will grow at a much greater rate than domestic petroleum supplies can support. M. L. Requa, former director of the oil division of the United States Fuel Administration, at the November, 1920, meeting of the American Petroleum Institute, in an address, made a statement²⁴ which is worth quoting in this regard:

The oil-shale industry, the coal-refining industry, the power-alcohol industry, with their potentialities and their limitations, deserve our close consideration. While they may superficially appear as our competitors, they are fundamentally our allies. When the time is ripe, I believe these supplemental sources of supply can be developed by the petroleum industry more advantageously than by any other agency.

Shale oil appears to be the most natural and logical substitute for petroleum. The supplies of shale are so great as to dwarf by comparison the quantity of petroleum already produced and still available for production in the United States. The writer believes that the oil-shale industry will ultimately be an industry of great magnitude and commercial importance in this country, but many years and much money will be required before it reaches this status. In its last

²⁴ Requa, M. L., Conservation: Am. Pet. Inst. Bull. 132, Dec. 10, 1920, p. 58.

analysis, it is an industry comparable to the low-grade ore mining industries of the Western States. Like them, it will require the services of the highest types of business, executive, and technical skill, backed by a large capital; and the investors must be able and prepared to wait a considerable time for a conservative return on the investment.

The development of the oil-shale industry should be promoted legitimately and honestly. It can be best developed and the right kind of backing encouraged by presenting facts and not half-proved theories; by deliberately recognizing and facing the limitations and problems of the industry, instead of glossing these over with a coating of vague estimates and statements of enormous possibilities. Development can be aided also by the proper kind of investigation and the publication of the results of investigations by governmental and private agencies, and by the proper cooperation of governmental agencies, petroleum organizations, and shale-oil companies. The embryonic oil-shale industry should, for the sake of rapid growth and its future, clear itself of the swarm of fraudulent promoters and promotion companies which is so large at present that the true position and merits of the oil-shale industry are difficult for many to see.

ESTIMATED COSTS AND PROFITS IN THE OIL-SHALE INDUSTRY.

Those who have had experience in estimating or calculating costs and profits in any large-scale manufacturing industry do not need to be cautioned as to making similar estimates for the oil-shale industry. However, much promotion literature has appeared, giving estimated costs and profits of this or that oil-shale company, operating or contemplating operations, and similar estimates have appeared from time to time in the technical and nontechnical press. All but very few of these estimates have neglected to include many important items of cost which must be considered if an estimate of even fair accuracy is to be obtained.

In the first place, the present status of our oil-shale industry is such that the making of reliable estimates of costs is impossible. Practically all retorting and refining installations intended to treat oil shale and shale oil are so small that they indicate little as to their operation under commercial conditions or whether they will work at all under such conditions. No one really knows just what products can or will be made from oil shale, what price these products will bring in a competitive market, or with what degree of satisfaction they can be used. Rates of depreciation of equipment can not be determined accurately from small and noncommercial plants for application to large plants working under commercial conditions.

There are other items entering into the profit and loss statement which can not be determined with even a rough degree of accuracy at present. It must be realized that economic and competitive conditions will be different in different districts, and that it is, therefore, impossible to make a blanket estimate to cover operations in any or all districts, as has been frequently attempted.

For these reasons the writer does not venture to estimate probable costs and profits for the future American oil-shale industry. Some factors influencing these items will be the type of process used; nature and extent of shale deposits; completeness of refining the crude products; scale of operations; situation of plant with respect to supplies, water, and markets; local cost of labor; and transportation costs and facilities. Such factors may be very different for different companies and must be considered for each plant.

It may be worth while to point out some items entering into cost estimates for the oil-shale industry, many of which have been frequently overlooked or ignored by those who have publicly issued statements of estimates.

SOME ITEMS TO BE CONSIDERED IN ESTIMATING COSTS.*

CAPITAL INVESTMENT.

The main items of capital investment will consist of the following:

(a) Cost of land, in which may be included legal expense in connection with clearing of title; cost of surveying; water rights; permanent improvements, such as road building; and cost of sampling and preliminary assays.

(b) Buildings, including those necessary for mining, crushing, storage, and retorting of the shale, and for refining the products. This item will also include machine shop, laboratories, garage, office, buildings for housing labor, and the like.

(c) Machinery and equipment, including that necessary in mining, retorting and refining operations; power-plant equipment; laboratory equipment; garage equipment (machinery and automotive equipment); machine-shop equipment; furniture and fixtures for office, laboratory, and camp; and transportation equipment (tanks, barrels, drums, tank cars, pipe lines, trucks, and the like).

MANUFACTURING EXPENSE.

Under manufacturing expense are grouped many items of expense frequently overlooked in making estimates. Manufacturing expense includes the following:

* The material in this section is based on a report, "Some Items of Investment, expense, and profit in oil-shale operations": Reports of Investigations, Bureau of Mines, Serial No. 2214, February, 1921, prepared by L. H. Sharp and A. T. Strunk, of the State of Colorado Cooperative Oil-Shale Laboratory, Boulder, Colo., with the assistance of the writer.

(a) *Material*.—Cost of raw material includes either the purchase price of the land or the rentals and royalties on leased land. If the land is purchased it constitutes a wasting asset and depreciates in value for every ton of shale mined. Under this heading may also be included costs of supplies, chemicals, fuel, water, etc., used in mining, retorting, and refining, and cost of their delivery to the plant. Sulphuric acid and caustic soda will probably be used in refining, and these chemicals constitute an important item of cost.²⁴ There may also be included under this heading the cost of packing and preparing products for market, although this may better be charged to sales expense or to plant overhead.

(b) *Direct labor*.—Labor entering into the direct production of shale oil and its products. It should be distributed properly for each stage of mining and manufacture.

(c) *Overhead expense*.—Overhead includes indirect labor; superintendence; depreciation of plant and equipment; heat, light, and power; taxes; insurance; repairs and maintenance; interest on value of products in storage; miscellaneous supplies; laboratory maintenance; and miscellaneous expense. Overhead should also be distributed properly for each stage of mining and manufacture.

SELLING AND GENERAL EXPENSE.

The main items of selling and general expense would be somewhat as follows: Salaries of general office force and sales force; advertising; commissions; insurance on office buildings, fixtures, and sales equipment; office supplies and maintenance; taxes on office building and sales equipment; heat, light, and power for office building; depreciation of office building, furniture, fixtures, and distributing and sales equipment; repairs on and maintenance of delivery and sales equipment; interest and expense on unpaid obligations; traveling expenses; cost of legal services and accident insurance or compensation for employees. Shrinkage in quantity of products between refining and selling point should be charged to selling expense.

The above statement of items of investment and expense for the industry does not pretend to be complete, but it may serve to indicate that many of the hitherto published cost estimates for oil-shale operations are not as comprehensive as they should be.

SOURCES OF PROFIT.

The probable sources of profit in the oil-shale industry are crude oil or its products which ultimately are expected to be motor fuels,

²⁴ Losses in refining shale oil will be greater than losses incurred in the equivalent refining of petroleum. Such losses must be charged to the particular stage of manufacture in which the loss occurs and the amount of such loss based on the value of the refined, not the crude, product.

burning oils, gas and fuel oils, lubricants, paraffin wax, nitrogen compounds derived from oil, shale gas (which may be an indirect source of profit by reducing fuel costs, as also may refinery sludges if burned under stills), and possibly ammonium sulphate or similar ammonia salts. There may also be important by-products and specialties, but as yet it is not safe to calculate on them.

The total income from these sources must be sufficient to cover manufacturing, marketing, and general expense, and leave a margin for depletion reserves and for profit on capital invested. Income must be calculated on the basis of current refinery prices, not retail prices. The difference between these two, less transportation and sales expenses, will be the profit of the retail sales division, if the organization has its own retail sales agency and if there is any profit to be made in retailing.

APPENDIX.

SAMPLING AND ASSAYING OIL SHALES.

SAMPLING.

Before a deposit of oil shale is opened, the operator will wish to determine the probable quantity of oil and other products obtainable from the shale, and also to decide on what part of the deposit to work. After operations have started, it will be advisable to make assays of samples, taken from the mine and from new workings, to serve as a check on commercial retorting operations and to insure that the shale sent to the retort does not go below a certain minimum richness.

Probably the most important factor entering into the value of an oil-shale assay is the method used in taking and preparing the sample. The accuracy and reliability of any analysis depends primarily on the fairness of the sample taken, that is, whether it accurately represents the deposit sampled. Samples must be so taken as to represent accurately, in proper proportions, the entire deposit under consideration. It is not particularly easy or simple to obtain a representative sample from any shale deposit.

In sampling shale it is important to take the samples from beyond the zone of surface weathering, and for this reason it is recommended that they be taken by core drilling. However, especially in massive deposits, surface weathering apparently does not affect the shale for more than a few inches from the outcrop, and in many places, therefore, it should be satisfactory to cut a trench perpendicular to the bedding plane of the seam and across the entire face of the seam. This trench should be of uniform width and depth, and material taken from the first 6 or 8 inches from the outcrop should not be included in the sample. In this case it is recommended that a sample of at least 100 pounds be taken, crushed, and quartered down to 25 pounds. This quantity should be crushed to maximum size of one-half inch and again quartered to 5 pounds, which should be crushed to maximum size of one-fourth inch, thoroughly mixed, and used for assaying.

Sampling at a working face could be done in the same manner, though it would not be necessary to reject the surface material, and

at many faces a head sample of 25 pounds should be sufficient. For sampling a working face it is recommended that a trench 4 by 4 inches be cut, clean and uniform, perpendicular to the bedding plane of the seam. In sampling a trench care should be taken not to break off very large pieces of shale, as the richer material has a tendency to break in larger pieces than that of lower grade.

Generally the methods commonly used²⁷ for mine sampling, as described in any standard textbook on mining, should be carefully followed. The accuracy of the assay depends on the accuracy of the sample.

ASSAYING OIL SHALES FOR OIL YIELD.

Many field and laboratory methods have been proposed for determining the oil yield of samples of oil shale. In a previous publication²⁸ of the Bureau of Mines on oil shale the assay method used in Scotland was described and tentatively recommended to American oil-shale investigators. However, the bureau has for some time been studying methods for testing and assaying oil shales and has developed a method and apparatus which is more convenient to use than that developed in Scotland and is of apparent greater accuracy, especially for the richer American shales. The yields obtained by the new apparatus from Scottish shales are about 8 gallons higher per ton than those obtained by the Scottish method on the same shale, but the quality of oil produced is about the same with each method. The new method was devised by L. C. Karrick, of the Bureau of Mines, after trying out practically all of the proposed types of assay apparatus. For the purpose of this bulletin it will be sufficient to quote from a report²⁹ describing the apparatus and the method of use.

Figure 4 illustrates the assay retort now used by the bureau. The parts necessary to set up the apparatus are indicated and may be purchased from any good chemical or assay supply house. By observance of the directions below, the results of duplicate determinations can be expected to agree within 1 to 1½ per cent in the field and within less than 1 per cent if the apparatus is used in the laboratory.

²⁷ The following publications of the Bureau of Mines describe recommended methods of sampling coal:

Pope, G. S., Sampling coal deliveries and types of Government specifications for the purchase of coal: Bull. 63, 1913, 63 pp.; Methods of sampling delivered coal, and specifications for the purchase of coal for the Government: Bull. 116, 1916, 64 pp.; Directions for sampling coal for shipment or delivery: Tech. Paper 133, 1917, 15 pp.

Holmes, J. A., The sampling of coal in the mine: Tech. Paper 1, 1911, 18 pp.

Feldner, A. C., Notes on the sampling and analysis of coal: Tech. Paper 76, 1914, 59 pp.

²⁸ Gavin, M. J., Hill, H. H., and Perdew, W. E., Notes on the oil-shale industry with particular reference to the Rocky Mountain district: Reports of Investigations, Bureau of Mines, Serial No. 2250, April, 1921, p. 19.

²⁹ Karrick, L. C., A convenient and reliable retort for assaying oil shales for oil yield: Reports of Investigations, Bureau of Mines, Serial No. 2220, March, 1921, revised April, 1922, 6 pp.

METHOD OF OPERATING ASSAY RETORT.

SEALING OF RETORT IMPORTANT.

In assembling the outfit, all pipe threads should be turned up tightly with a mixture of pulverized litharge and glycerin. The connections (12) with the reflux condenser (11) and with the graduate (10) should also be well sealed with the same cement unless the highest grade corks are used.

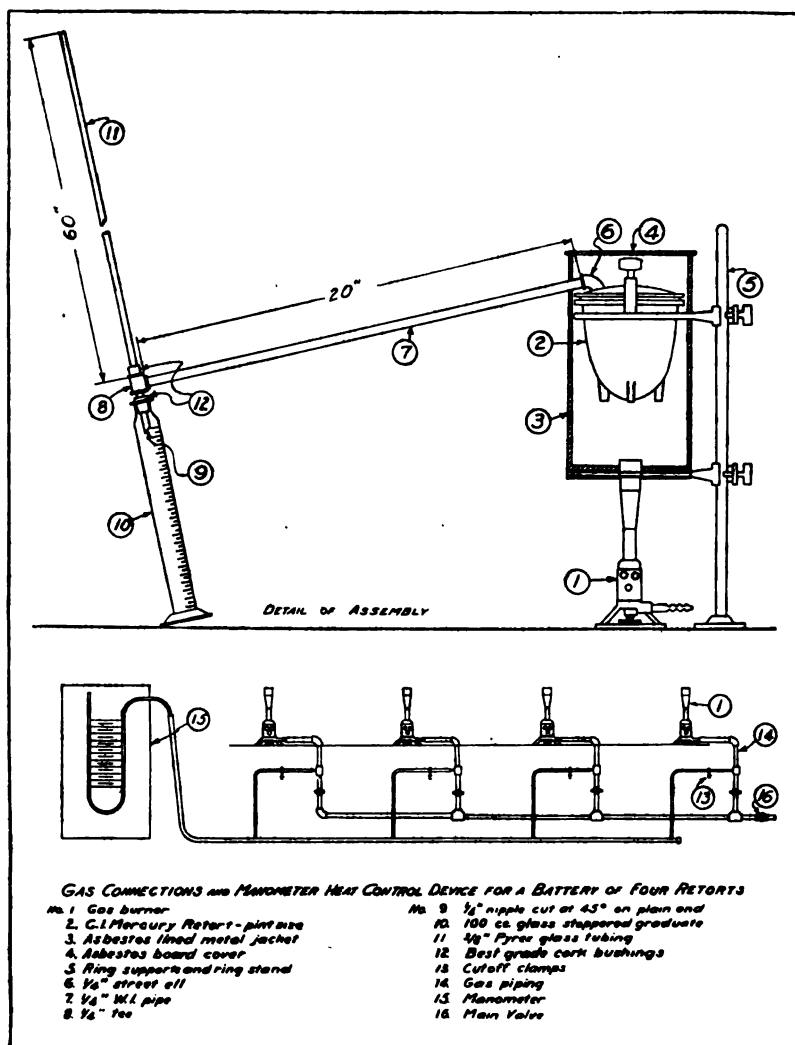


FIGURE 4.—Oil-shale assay retort.

Extreme care should be exercised in the preparation and maintenance of the contact surfaces of the lid and base of the retort. These surfaces will not form a gas-tight fit when new, and this defect must be remedied by grinding the two parts together by hand, using No. 220 carborundum and water, as in grinding cylinder valves. The grinding is facilitated by clamping the lid in an inverted

position on a table and pressing down lightly on the base while rotating it. After grinding, clean the parts thoroughly with clean water, clamp the lid in place, then test the improved contact by immersing the retort in water, and forcing air into the delivery tube, with the lungs. If small air bubbles appear, the leakage must be repaired. Try readjusting the lid, and if this does not stop the leakage, then repeat the grinding. The first preparation of the contact surfaces may require one-half to an hour's time, but once an air-tight contact is produced, a slight grinding before each run will keep the surfaces in perfect condition. The improved surfaces will form a gas-tight contact if kept perfectly clean and aligned while tightening the clamp.

It is unnecessary to use a cement between the contact surfaces when they are properly prepared, and it is recommended that no cement be used. All the cements tested for this purpose have been found to be unsuitable, either because they fail to seal the joint or adhere so tenaciously that difficulty is experienced in cleaning the surfaces for subsequent runs, or become more or less porous on heating, causing loss of oil by capillarity. A clean, well-prepared emery-ground joint is convenient and reliable, and when tested as above described, assurance can be had that no oil has been lost during the run through leakage of vapors. After a little experience has been gained, each retort can be ground and made ready for a run in five minutes.

It is important to use a jacket around the retort while heating, permitting about one-quarter of an inch clearance at the sides and high enough to clear the clamp screw on top. Holes must be cut in the jacket for ring support and delivery tube. A cover of asbestos board is necessary to reflect heat down against the retort lid, otherwise much oil will condense on the inner side of the lid, and drip down on the shale in the retort, thus causing losses by secondary distillation.

OPERATION OF THE RETORT.

Crush the shale so that all will pass a one-quarter inch screen or smaller mesh. Mix the samples thoroughly and then weigh out just enough to fill the retort (2) level full. Tamping the charge into the retort is permissible and will not affect the results; this, however, is seldom necessary, and experience has demonstrated that it is better to permit the charge to rest loosely in the retort, as some rich shales intumesce or coke together, and removal of the residue in such cases is usually difficult. After the lid is fastened carefully in place and tested for leakage as directed above, place the retort in ring support (5), adjust jacket (3) and cover (4), start the burner (1), then fasten the graduated cylinder (10) and reflux condenser (11) tightly in place. Use a very small flame at first, so that no oil will appear within one hour after starting. One hour is about the minimum time in which the shale charge can be brought up to the initial distillation temperature and yet possess a fairly even temperature throughout. Some shales begin to yield vapors at approximately 250° C., whereas others show no signs of oil vapors until 375° C. is reached. Before distillation makes much progress, the interior or coolest shale should be at least as hot as the vaporizing temperature of the oil evolved, otherwise vapors will migrate to the central or coolest part and condense there, causing losses of oil by cracking on redistillation of the condensed oil.

When about 1 c. c. of oil has accumulated, the flame should be increased. If the maximum quantity of oil is desired, it is imperative to distill rapidly as soon as oil appears in the receiver. If the flame has been of such size that oil first appears in about 60 minutes, it will then be safe to double the size of

the flame, and thereafter every 20 to 40 minutes to increase the heat by an equal amount. The oil should accumulate in the graduate at a uniform rate, and the analyst will soon become adept at pushing the rate of retorting without exceeding the condenser capacity. It should be remembered that the quantity of oil recovered will vary for the same shale sample if the rate of retorting varies, the greater quantity being obtained with rapid retorting and, conversely, a much less amount with a long distillation period. With the average oil shale the recovery when the actual oil-producing period is approximately 2 hours will be 5 per cent greater than when a 6-hour distillation period is used. When distillation is completed in less than 2 hours, there is danger of inaccuracy from exceeding the condenser capacity and from decomposition of oil vapors by excessive temperatures. When heat is being supplied very rapidly the temperature surrounding the distilling shale particles may be much higher than is required for further oil production. This condition is brought about by the low thermal conductivity of the shale and by the heat-consuming reactions that take place as destructive distillation progresses inwardly from the surface of each shale particle toward its center. The surface of the shale particle will, therefore, be the hottest part, and it is through this hot zone that the vapors must pass in finely separated capillary streams in escaping from the porous spent shale. This condition will cause some cracking and loss of oil. Distillation should be rapid, however, when the maximum yield of oil is desired, in order to prevent stagnation of oil vapors within the hot retort or connection, and in all cases should progress at a uniform rate. If oil droplets appear near the top of the reflux condenser (11), the rate of heating must be retarded slightly. The condenser has been made small in order to limit to a safe maximum the rate of retorting that will assure the greatest recovery of oil of high quality from the average oil shale.

If distillation has progressed uniformly, the rate of oil accumulation will decline very suddenly as the last oil is being distilled off, and, when this cessation becomes apparent, the flame should once more be increased the usual amount, which will be sufficient to finish the distillation. White vapors may appear at the end of the run, but these have been found to be noninflammable and to carry no oil vapors. The delivery tube should be kept warm throughout the run in order to prevent the oil from congealing therein and to save delay in draining at the end of the run. When it is observed that the rate of accumulation of oil has suddenly declined and the bottom of the retort has become a very dull red, it can be safely assumed that the distillation of oil is completed, as this temperature is well above the final oil-yielding temperature for all oil shales. The heating should continue for a half hour longer to permit complete drainage of the delivery tube and reflux condenser.

MEASUREMENT OF THE OIL RECOVERED.

VOLUMETRIC METHOD.

Inexperience in measuring the accumulated oil will cause a much greater error in results than will a lack of refinement in retorting technique or imperfections in the retort.

The stoppered graduate and contents must be warmed until the oil is in a very fluid condition. Allow the oil to settle well; some water is given off near the last stages of distillation and will remain suspended in the partially congealed oil. In order to read accurately the volume of the oil and water, it is necessary to assist the formation of a perfect meniscus by releasing any

oil and water clinging to the sides of the graduate. This is best accomplished by revolving the graduate, while still warm, between the palms of the hand. Read the upper level of the oil and then the lower level, if it is well defined and is not rendered obscure by emulsion, sediment, or clinging oil and water. The following procedure will facilitate accurate reading of the lower meniscus and also provide against the possibility that water and sediment may not have settled out completely. With a pipette draw off all but a few cubic centimeters of the oil while still warm; dilute the contents of the graduate with 10 to 20 c. c. of clean gasoline, and agitate gently till the emulsion disappears. Allow the solution of shale oil and gasoline to settle, then draw off and discard it; add gasoline as before. Repeat as often as required, until a clear meniscus results. From the number of cubic centimeters of oil collected calculate the gallons of oil per ton of shale.

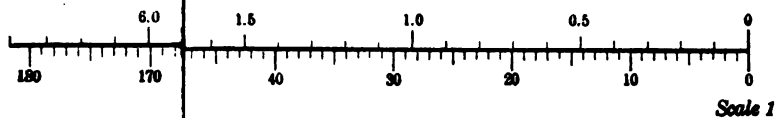
GRAVIMETRIC METHOD.

To determine more accurately the amount of oil recovered, weigh the graduate and contents and allow sufficient time to settle well, while warm. (The weight should be expressed in grams.) Remove the oil and determine its weight by the method described below. Draw off with a large pipette all the oil except 4 or 5 c. c., being extremely careful that no water or sediment is drawn into the pipette. Set this sample aside for specific gravity and other determinations as it will be a sufficiently representative sample of the total oil recovered. Dilute the remaining few cubic centimeters of oil with 20 c. c. of petroleum ether, and while adding the ether wash down the inner sides of the graduate with the solvent. When all the water and solids have settled and the shale oil is well mixed with the petroleum ether, draw off the clear liquid nearly to the level of the water and discard it. Repeat this washing process several times until a colorless solution of ether remains on top of the water and no shale oil is left clinging to the walls of the graduate. Remove the ether to approximately the level of the water and eliminate the balance by evaporation. The last of the ether is very quickly removed by tipping the graduate as far as possible to one side without spilling the contents, then revolving the graduate while blowing into it; the ether is picked up on the side walls of the graduate during rotation, thus providing a large evaporating surface. As soon as the ether disappears, which will require scarcely a minute's time, weigh the graduate, now containing the water and sediment only, and subtract this weight from the original weight; the difference will be the total weight of the oil obtained from the shale. Determine the specific gravity of the oil first removed, with the Barrett, Regnault, or similar type of specific gravity bottle, and divide the total weight of oil obtained by the specific gravity. The result will be the true volume of oil in cubic centimeters, at the temperature of the specific gravity determination. (Ordinarily 60° F.) This is converted to gallons of oil per ton of shale by use of the formula:

$$\text{Gallons of oil per ton of shale} = \frac{\text{Cubic centimeters of oil} \times 240}{\text{Grams of shale used in retort}}$$

Duplicate volumetric determinations should agree within 2 per cent and gravimetric determinations within 1 per cent.

The table of factors given in Table 24 below will facilitate calculations. For any given weight of shale used (column 1 or 2), select the corresponding



VOLUME

Join cubic centimeters
with grams of shale
yield on Scale 5.

U. S. BUREAU OF MINES

PETROLEUM DIVISION

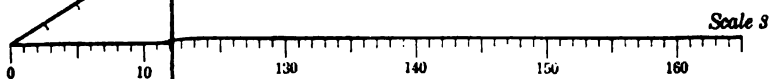
OIL SHALE SECTION

DESIGNED BY

M. J. GAVIN

L. C. KARRICK

J. J. JAKOWSKY



actor in column 3. Divide by this factor the volume of oil collected (in cubic centimeters) in order to convert into gallons of oil per ton of shale. For shale charges whose weights in grams are not even multiples of 10 it will be necessary to interpolate to obtain the proper factor.

TABLE 24.—*Factors facilitating calculations of oil-shale assays.*

| Weight of retort charge. | | | Weight of retort charge. | | |
|--------------------------|--------------|--------------|--------------------------|--------------|--------------|
| Grams.
1 | Ounces.
2 | Factor.
3 | Grams.
1 | Ounces.
2 | Factor.
3 |
| 10 | 0.35 | 0.042 | 310 | 10.94 | 1.294 |
| 20 | .71 | .083 | 320 | 11.30 | 1.335 |
| 30 | 1.06 | .125 | 330 | 11.65 | 1.377 |
| 40 | 1.41 | .167 | 340 | 12.00 | 1.419 |
| 50 | 1.76 | .209 | 350 | 12.36 | 1.460 |
| 60 | 2.12 | .250 | 360 | 12.71 | 1.502 |
| 70 | 2.47 | .292 | 370 | 13.06 | 1.544 |
| 80 | 2.82 | .334 | 380 | 13.41 | 1.586 |
| 90 | 3.18 | .376 | 390 | 13.77 | 1.627 |
| 100 | 3.53 | .417 | 400 | 14.12 | 1.669 |
| 110 | 3.88 | .459 | 410 | 14.47 | 1.711 |
| 120 | 4.24 | .501 | 420 | 14.83 | 1.753 |
| 130 | 4.59 | .542 | 430 | 15.18 | 1.794 |
| 140 | 4.94 | .584 | 440 | 15.53 | 1.836 |
| 150 | 5.29 | .626 | 450 | 15.88 | 1.878 |
| 160 | 5.65 | .668 | 460 | 16.24 | 1.919 |
| 170 | 6.00 | .709 | 470 | 16.59 | 1.961 |
| 180 | 6.35 | .751 | 480 | 16.94 | 2.003 |
| 190 | 6.71 | .793 | 490 | 17.30 | 2.045 |
| 200 | 7.06 | .835 | 500 | 17.65 | 2.086 |
| 210 | 7.41 | .876 | 510 | 18.00 | 2.128 |
| 220 | 7.77 | .918 | 520 | 18.36 | 2.170 |
| 230 | 8.12 | .960 | 530 | 18.71 | 2.212 |
| 240 | 8.47 | 1.001 | 540 | 19.06 | 2.255 |
| 250 | 8.82 | 1.043 | 550 | 19.41 | 2.295 |
| 260 | 9.18 | 1.085 | 560 | 19.77 | 2.337 |
| 270 | 9.53 | 1.127 | 570 | 20.12 | 2.379 |
| 280 | 9.88 | 1.168 | 580 | 20.47 | 2.420 |
| 290 | 10.24 | 1.201 | 590 | 20.83 | 2.462 |
| 300 | 10.59 | 1.252 | 600 | 21.18 | 2.504 |

Plate XVIII is an alignment chart prepared to facilitate calculations of results with the assay retorts. Larger-scale copies of this chart can be obtained from the Director of the Bureau of Mines, Washington, D. C.

Figure 4 also illustrates a method of connecting up a battery of four assay retorts to the same gas supply, so as to permit simultaneous distillations with all retorts. The manometer system of connections provides a means of observing instantly the gas pressure at each burner, and therefore each burner can be regulated to give any desired heat. This system is in use in the oil-shale laboratories of the Bureau of Mines and provides accurate heat control when the four retorts are operated simultaneously, whether at one rate or separate rates.

If an oil or an activated charcoal scrubber is used with the assay retort, scrubber naphtha can be recovered from the average shale at the rate of 1 gallon or more per ton. This naphtha is a valuable product and should be taken into account if accurate results are desired. If an oil scrubber is used, it is connected to the upper end

of the reflux condenser by a short piece of rubber tubing, and the gas is drawn through the oil by an aspirator or water suction pump connected to the outlet of the scrubber bottle. A manometer must be placed in the line between condenser and scrubber to make certain that a slight negative pressure is maintained. If the gas produced with the oil is being collected in a gasometer, the discharge of the gas-scrubbing bottle can be connected to the gasometer and the counterweight of the latter adjusted to create a slight negative pressure in the system. A pinch clamp on the rubber tubing between scrubber and gasometer permits close regulation of pressure in the retort condenser system. With this arrangement all connections must, of course, be tight.

Any good type of gas-washing bottle can be used. About 200 c. c. of scrubber (mineral seal) oil should be used. Usually more accurate results will be obtained if two retorts are operated simultaneously and the gas from both passed through the scrubber. After distillation is completed the scrubber oil is placed in a 300 c. c. still of the Engler type and the naphtha slowly distilled off. The condenser should be cooled with a mixture of ice and water and the distillate collected in a cooled, graduated receiver. When almost all the naphtha has been distilled from the oil, a ring of oil will be seen gradually creeping up the neck of the distilling flask. When this ring has come within an inch of the side outlet, or delivery tube, distillation should be stopped. The yield of naphtha should be calculated according to the formula:

$$\text{Gallons of naphtha per ton of shale} \left\{ \frac{\text{Cubic centimeters of naphtha recovered} \times 240}{\text{Grams of shale used in retort}} \right.$$

If two retorts have been connected to one scrubber, the total weight of shale in both should, of course, be used in the formula.

Fresh scrubber oil should be used in each determination; it should have an initial boiling point not lower than 450° F.

In the laboratories of the Bureau of Mines activated charcoal is used as an absorbing medium. A glass or metal cylinder of about 100 c. c. capacity, and having inlet and discharge connections at opposite ends, is filled with activated charcoal specially prepared for gas scrubbing. Plugs of cotton are placed at each end of the cylinder to keep the charcoal from passing into the inlet and discharge tubes. All connections should be tight. The absorber is connected to the end of the reflux condenser of the assay retort with a short piece of rubber tubing. (Two retorts can be connected to one absorber.) With this kind of absorber it is not necessary to use a pump or aspirator to draw the gas through.

After the run has been completed the absorber is disconnected and superheated steam at about 570° F. is passed into one end, while the other is connected to a metal condenser cooled with ice water. The steam should be passed through slowly, and at the end the whole body of charcoal should reach the temperature mentioned above. The steam and naphtha will condense together and separate cleanly in the graduated receiving cylinder, which should be immersed in a beaker filled with shaved ice and water. After accumulation of naphtha has ended steam should be passed through for 10 minutes more, when the charcoal will be in condition for reuse. The yield of naphtha is calculated by means of the formula given above.

ESTIMATION OF AMMONIA YIELDS FROM OIL SHALE.

It is a relatively simple matter to determine the total nitrogen content of an oil shale and from this to calculate the theoretically maximum yield of ammonium products—usually ammonium sulphate. It is not definitely known as yet if it will be desirable to try to recover nitrogen—as ammonia or other products—from American oil shales, but there is a fair probability that nitrogen will be recovered in some form. In view of the lack of knowledge of the processes that will be used in domestic oil-shale operations, and how adaptable these processes will be for making nitrogen products, or what particular nitrogen products will be made, it is the opinion of the writer that a determination of the total nitrogen in an oil shale should furnish a satisfactory basis for estimating possible yields of nitrogen products.

Although it is highly improbable that in commercial practice all the nitrogen in an oil shale can be converted into marketable products, yet knowledge of the total nitrogen will give a good indication of the value of shales as sources of nitrogen compounds, whatever they may be. Until more work has been done on the problem of recovering nitrogen from oil shales, the writer does not believe it advisable to recommend any analytical or testing method for oil shales that will give results expressed in terms of commercial yields of any particular nitrogen product, for the reason stated above. If total nitrogen is determined, it can be converted into terms of ammonium sulphate (or any other expected product), and for the time being, 60 per cent of the theoretically maximum recovery can be taken as perhaps representing the possible commercial production. In Scotland 60 per cent of the nitrogen of the shale is recovered as ammonium sulphate.

The common methods used for determining total nitrogen in coals, oil shales, and the like are usually modifications of the Kjeldahl method, well known to organic chemists. The particular modi-

fication of this method used with good results by the writer is the Kjeldahl-Gunning method; as applied to oil shales it is as follows:

METHOD FOR DETERMINATION OF TOTAL NITROGEN.²⁰

Samples are carefully quartered, and one-quarter is ground to pass a 60-mesh sieve. One gram of this fine material is digested with 30 c. c. of concentrated sulphuric acid, 5 grams of potassium sulphate, and 0.5 gram of mercury, in a 500 c. c. Kjeldahl flask for 4 hours, or until the shale is perfectly white. It is advisable to place a small funnel in the neck of the digestion flask to prevent spattering of the liquid during the digestion. After the flask has cooled for 10 minutes, add a few crystals of potassium permanganate to insure complete oxidation.

After thorough cooling, the solution is diluted to 200 c. c. with distilled water, and 25 c. c. of potassium sulphide solution (40 grams potassium sulphide per liter) is then added to precipitate the mercury. One gram of powdered zinc and a small piece of paraffin are next added to prevent bumping and foaming during the distillation. Then enough saturated sodium hydroxide solution to make the contents of the flask distinctly alkaline is added cautiously. Danger of loss of ammonia while the alkali is being added can be minimized by holding the flask in an inclined position and allowing the sodium hydroxide solution to form a layer below the lighter acid solution. The mixture is not shaken until the flask is connected to the condenser.

Next, the flask is connected to a regular Kjeldahl condenser, with a condenser tube of block tin and with a Kjeldahl connecting tube between flask and condenser. The delivery tube of the condenser is connected by a piece of rubber tubing to a short length of glass tubing, which extends below the surface of an acid solution in a 500 c. c. Erlenmeyer flask. The acid solution consists of 10 c. c. of 0.25 normal sulphuric acid solution, diluted to 40 c. c. with distilled water.

The glass delivery tube is barely allowed to touch the surface of the acid in the Erlenmeyer flask until after the distillation flask has been shaken, to prevent the acid being drawn back into the condenser and distillation flask immediately after the shaking. After the distillation flask has been shaken to mix its contents thoroughly, the gas burner beneath is lighted; as soon as the contents begin to boil, the glass delivery tube is lowered to its full extent in the Erlenmeyer flask and should then dip one-fourth to one-half inch below the surface of the acid solution therein.

²⁰ This method is essentially that used by the Bureau of Mines for determining nitrogen in coal, as described by Stanton, F. M., and Fieldner, A. C., *Methods of analyzing coal and coke*: Tech. Paper 8, Bureau of Mines, 1913; pp. 24-25.

The rate of distillation is so regulated that 200 c. c. is distilled into the Erlenmeyer flask in about two hours; then the Kjeldahl flask is disconnected and the glass delivery tube thoroughly washed into the Erlenmeyer flask with distilled water. The excess acid in the Erlenmeyer flask is then titrated back with standard 0.1 normal ammonium hydroxide solution, with methyl orange as an indicator. The strength of the standard ammonia is carefully checked against the acid solution each day.

Blanks are run at frequent intervals during the course of analyses with 1 gram of sugar in place of the shale and the same proportions of all other reagents, the same time of digestion, and the like, as when an analysis is made. The correction indicated by these blanks is made in all subsequent shale analyses. It is particularly necessary to run blanks after a new solution of any of the reagents used is made up. Boiling the sodium hydroxide solution for three or four hours, after the solution has been made up and before it has been used in the analysis, will usually make the blank correction very small.

Results can be calculated to percentage of nitrogen or directly to pounds of ammonium sulphate per ton of shale. Ordinarily it is convenient to calculate to percentage of nitrogen. The percentage of nitrogen multiplied by the factor 94.3 gives the pounds of ammonium sulphate per ton of shale. On the basis of this method of analysis, or of any method that determines total nitrogen, such calculated yield of ammonium sulphate must, of course, be considered as the theoretical maximum recovery, probably never attainable in commercial practice. To calculate percentage of nitrogen to commercial yields of ammonium sulphate, based on Scottish working practice, multiply the percentage of nitrogen, as above determined, by 94.3 and then by 0.60.

DIRECT METHODS OF DETERMINING AMMONIUM SULPHATE YIELDS IN COMMERCIAL PRACTICE.

An earlier publication²¹ of the Bureau of Mines has described the method used in Scotland for determining probable commercial yields of ammonium sulphate from oil shale. The bureau, in cooperation with the department of metallurgical research of the University of Utah, has for the past two years been studying this and other methods of testing oil shales for ammonia recovery. With the apparatus under discussion, by changing conditions only a little, it has been possible to obtain very different results with the same shale. For this method to be of much value, operating conditions must be

²¹ Gavin, M. J., Hill, H. H., and Perdew, W. E. Notes on the oil-shale industry, with particular reference to the Rocky Mountain district: Reports of Investigations, Bureau of Mines, Serial No. 2256, April, 1921, p. 21.

definitely fixed and rigidly maintained, and these conditions will probably differ for different shales.

As an example of the necessity of fixing definite conditions for operating the Scottish tube method for determining ammonium sulphate yield, the directions given for operating the tube include the statement, " * * * the iron tube is brought to a bright red heat as rapidly as possible." The diversity of opinion among operators as to what shade of color represents a bright red is enough to make a difference of over 10 per cent in the yield of ammonia from a given shale by this method. However, the Scottish tube method and its modifications are proving of value in determining conditions most favorable for nitrogen recovery from oil shales, and the Bureau of Mines is therefore continuing their use. Properly equipped with pyrometer control the tube may be useful for assay purposes later on, when more is known of the commercial methods that will be used for treating American oil shales.

Complete descriptions of various methods of testing oil shales for ammonia recovery and of conditions favorable for maximum ammonia recovery will be given in a future publication.

PHYSICAL AND CHEMICAL DATA ON COLORADO OIL SHALE.*

The data presented below are for a sample of massive oil shale taken from the Green River formation near De Beque, Colo., and are probably representative of the average shale of that formation. Certain factors, of course, will not hold well for Green River shales that are much richer or leaner than this particular sample, and it is not probable that these data will apply to oil shales from different districts or geologic horizons.

Physical and chemical characteristics of oil shale from De Beque, Colo.

1. Weight per unit volume:

| Size of particles. | Weight per cubic foot, pounds. |
|--------------------------------|--------------------------------|
| Uncrushed (run of mine) | 53. 775 |
| Plus 1 inch | 54. 775 |
| Minus 1 inch | 56. 015 |
| Minus $\frac{1}{2}$ inch | 58. 200 |

2. Apparent specific gravity: 2.00 to 2.10.

3. Specific heat:

| | |
|---------------------------------------|--------|
| Raw shale | 0. 265 |
| Shale spent by dry distillation | 0. 223 |

4. Heat of combustion:

| | Calories per gram. |
|---------------------------------------|--------------------|
| Raw shale | 2, 460 |
| Shale spent by dry distillation | 600 |

5. Thermal conductivity: Mean, 25 to 75° C..... 0. 00382

* Gavin, M. J., and Sharp, L. H., Some physical and chemical data on Colorado oil shale: Reports of Investigations, Bureau of Mines, Serial No. 2152, August, 1920, 8 pp.

| | | |
|--|--|------------------|
| 6. Proximate analysis of raw shale: | | Per cent. |
| Loss at 110° C. (moisture)..... | | 0.60 |
| Loss on ignition..... | | 40.00 |
| Ash..... | | 59.40 |
| | | <hr/> |
| | | 100.00 |
| 7. Analysis of ash: | | Per cent. |
| Silica (SiO_2)..... | | 44.70 |
| Iron and alumina (Fe_2O_3 and Al_2O_3)..... | | 25.60 |
| Lime (CaO)..... | | 17.65 |
| Magnesia (MgO)..... | | 5.28 |
| Undetermined..... | | 6.77 |
| | | <hr/> |
| | | 100.00 |
| 8. Oil yield, 42.7 gallons per ton; specific gravity of oil, 0.905. | | |
| 9. Heat of combustion of oil from this shale, 10,215 calories per gram (18,387 B. t. u. per pound). | | |

ANALYTICAL DISTILLATION OF SHALE OILS.

Experimental study of oil-shale retorting involves the production of oil from different shales under different controlled sets of conditions, and determination of the quantity and quality of the products recovered. For this work the Bureau of Mines uses several types of retorts and with each type systematically alters the retorting conditions in the direction indicating best results as to oil production, with regard to both quantity and quality of the crude oil. The problem, briefly, is to determine those conditions that from any shale will give the highest yield of the best oil. In the investigations, shales typical of the deposits in different States are being tested.

At the completion of each experiment the crude products are examined. The examination involves analyses of gas, water, spent shale, scrubber products, and crude oil. Of these products, the crude oil is of course the most important.

It is ordinarily a simple matter to determine the amount of oil yielded by a shale, but determination of the relative quality is more difficult. Thus far the most satisfactory method found for determining the quality of the shale oils is an analytical fractional distillation of the crude oil and examination of the fractions of the distillation. This method, with minor modifications, is the one used by the Bureau of Mines for examining petroleum;²² it is briefly described below:

NATURE OF THE LABORATORY DATA.

The methods employed in testing and analyzing the samples are to be described in detail in a bulletin that is now in process of preparation. For the

²² Dean, E. W., Properties of typical crude oils from the Eastern producing fields of the United States: Reports of Investigations, Bureau of Mines, Serial No. 2202, Jan., 1921, p. 2.

present, discussion is confined to a brief statement of the nature of the methods and to the significance of the figures obtained. The following data are recorded for each sample:

1. The specific gravity at 60° F. and the corresponding Baumé gravity (based on the modulus 140).²⁴ Actual measurements were in most cases made by the use of a specific gravity balance.

2. The percentage of sulphur. This was determined by burning a weighed quantity of oil in a calorimeter bomb, precipitating the sulphate formed as barium sulphate, filtering, igniting, and weighing.

3. The percentage of water. If any water was present it was determined by the distillation method.

4. Distillation at atmospheric pressure with data for the percentages distilling within 25° C. (45° F.) limits up to 275° C. (527° F.), and the specific and Baumé gravities of the fractions. Distillations were made by the Bureau of Mines Hempel distillation method, using charges of 300 c. c. of oil and distilling through a 6.5-inch fractionating column.

5. Vacuum distillation at a reduced pressure of 40 mm. "absolute" of the residuum from the "air distillation," with data for the percentage distilling between 25° C. (45° F.) limits up to 300° C. (572° F.) and for the specific and Baumé gravities. Saybolt universal viscosities (at 100° F.) and Fahrenheit cloud tests of the distillation fractions.

6. Conradson carbon-residue percentages of the residuum from the combined "air" and "vacuum" distillations.

ADDITIONAL TESTS ON SHALE OILS.

Up to the time data for this bulletin were assembled the Bureau of Mines had not always determined percentages of sulphur in the shale oils. Therefore Table 25 (p. 160) does not give the percentage of sulphur in every oil. At present, however, all crude shale oils are analyzed for sulphur content. It has been found that nearly all shale oils contain rather large percentages of sulphur; the average content of those thus far examined is about 0.6 per cent. There is, however, considerable variation from this figure, some oils running somewhat lower and a number much higher.

Also, until recently, Conradson carbon-residue tests have not always been made on shale-oil residuums; consequently these are not given in all the analyses in Table 25.

On account of the high percentage of solid paraffin wax in many crude shale oils and shale-oil fractions, it has been found somewhat inconvenient always to determine viscosities at 100° or 130° F. For this reason some of the viscosities reported were determined at 140° F. (60° C). Viscosities of the crude oils are also reported.

Specific gravities of the fractions solid or semisolid at 15.56° C. (60° F.) are determined with a Regnault specific gravity bottle.

The dark color of many of the vacuum fractions and of the crude oils themselves makes the taking of the cloud test impractical. Con-

²⁴ The bureau now expresses gravities in terms of the American Petroleum Institute scale ("A. P. I."), which is based on the modulus 141.5.

sequently on those fractions, and on the crude, a "setting-point" test is made. This method also is of value in that it follows the method used in Scotland and serves there as an index of the amount of solid paraffin wax recoverable from the oil. The procedure is to freeze a drop of the oil, of rather definite size, on the tip of the cooled bulb of a thermometer, then to invert the thermometer, and while rotating it about a vertical axis, to allow the temperature to rise at a definite rate until the drop melts and flows down the bulb. Unless the oil is very tarry or asphaltic, the setting point can be sharply and accurately determined.

In addition to these tests, shale-oil fractions resulting from the distillation at atmospheric pressure are combined and an unsaturation test of the combined fractions is made. The method used and the value of the data thus obtained are discussed under the heading "Evaluation of shale oils" (p. 156).

INTERPRETATION OF DISTILLATION DATA.

A discussion by Dean²² of the interpretation of the data obtained by a distillation is given below:

The methods employed by the Bureau of Mines for the distillation analysis of crude petroleum have not been developed with the idea of obtaining figures that parallel the results of actual refinery practice. As refinery practice has never been standardized, it has been necessary to select a fundamentally reproducible basis of comparison, rather than attempt to work in terms of yields and properties of commercial products.

The chief value of the present report lies in the fact that it permits a reasonably adequate comparison of different crude oils on the basis of fundamental physical and chemical properties. * * *

It is believed while the most satisfactory use of the figures involves a comparison, there is also a need for some sort of "rough and ready" interpretation in terms of commercial products. Therefore, the author has employed the following classification which, even if it has no other justification, is convenient because it permits discussion in terms of "given names."

1. The sum of all fractions distilling at atmospheric pressure below 200° C. (392° F.) is reported as *gasoline and naphtha*.
2. The sum of all fractions distilling at atmospheric pressure between 200° C. (392° F.) and 275° C. (527° F.) is reported as *kerosene*.
3. The sum of all vacuum distillation fractions having Saybolt viscosities (at 100° F.) of less than 50 seconds is reported as *gas oil*.
4. The sum of all vacuum distillation fractions having Saybolt viscosities (at 100° F.) between the inclusive limits of 50 and 99 seconds is reported as *light lubricating distillate*.
5. The sum of all vacuum distillation fractions having Saybolt viscosities (at 100° F.) between 100 and 199 seconds, inclusive, is reported as *medium lubricating distillate*.
6. The sum of all vacuum distillation fractions having Saybolt viscosities (at 100° F.) of 200 seconds or more is reported as *viscous lubricating distillate*.

²² Dean, E. W., Properties of typical crude oils from the Eastern producing fields of the United States: Reports of Investigations, Bureau of Mines, Serial No. 2202, January, 1921, p. 8.

The suitability of a given type of crude for producing cylinder stock has been assumed to be approximately in inverse proportion to the percentage of carbon obtained by the Conradson carbon-residue test made on the residuum from the vacuum distillation. This carbon-residue test undoubtedly does not tell the whole story, but its general value is indicated by the fact that Pennsylvania and West Virginia crudes show carbon residue of residuum figures between 1 per cent and 3 per cent while the corresponding figures for Kentucky crudes are from 5 to 8 per cent and for California crude from 13 to 20 per cent. Pennsylvania and West Virginia crudes are, of course, regarded as the most suitable material for producing cylinder stock, whereas it is made from oil of the type of Kentucky crude only through the use of special processes, and from average California crudes not at all.

Caution should be used in interpreting data obtained by vacuum distillation of shale oils. Many of the vacuum fractions appear to have properties satisfactory for the production of lubricating oils. Little is known, however, as to how these fractions would stand refining or how satisfactory for commercial use the refined lubricating oils would be. Many shale oils yield fractions with lubricating properties said to be much superior to those indicated by their viscosities, but the whole question of producing lubricating oils from shale oils is yet in an indeterminate state.

EVALUATION OF SHALE OILS.

Attempts to evaluate shale oils, even for comparative purposes in the laboratory, involve both technical and economic considerations. With respect to technical problems, one must consider and determine the possible products that can reasonably be expected from shale oils. With respect to economic questions, one must consider which of these possible products will be in greatest demand and how cheaply they can be placed on the market. Possibly in some parts of the country certain products will be in greater demand than in others, and a discussion of how cheaply these products can be produced and marketed leads to a consideration of plant location, transportation, labor supply, and similar economic factors which have been discussed in a previous section but present problems that each plant must individually solve.

Inasmuch as the chief problem in the oil-shale work of the Bureau of Mines is to determine the conditions most favorable for the production from different shales of highest yields of best products, the bureau must first consider what these products are likely to be; second, determine the relative quantities of these products in the oils under consideration; and, finally, determine the relative amounts of these products from different shales. The question has been given careful thought and the matter taken up with many producers and refiners of petroleum in the United States.

In a previous section (p. 119) the opinion has been expressed that shale oils will be of importance primarily as a source of motor fuels, and, secondarily, as a source of fuel oils. Other products will probably be of less importance, and it has been finally decided that shale oils may reasonably be evaluated on the basis of the amount and quality of motor fuel they will yield.

An analytical distillation can indicate the relative amount of motor fuel of a given volatility to be obtained from different oils, but chemists have to be somewhat arbitrary in selecting the range of volatility of the motor fuel, or, in other words, its "end point." In making analytical distillations of petroleum and shale oil it is the practice of the Bureau of Mines to stop the "air" or atmospheric pressure distillation when the vapor temperature in the neck of the fractionating column reaches 275° C. The combined fractions distilling up to this temperature are considered, for purposes of comparison, as representing all the gasoline, crude naphtha, and crude kerosene. The general tendency during the past few years has been to increase the boiling range, or, in other words, decrease the volatility of motor fuels. This tendency has been checked for the present, but if there should be a marked increase in the demand for motor fuels it might be renewed in the future.

No definite prediction, therefore, can be made as to the future end point of motor fuel; accordingly it has been found most satisfactory for purposes of comparison to classify shale oils on the basis of relative percentages distilling up to 275° C. vapor temperature. This distillate is generally spoken of as "tops." Thus the value of a shale oil as a producer of motor fuel may tentatively be indicated by the percentage of tops that can be produced from it. It is to be noted that this is taken to indicate the quantity of *crude* motor fuel to be expected from a given shale oil.

Another point to which attention must be directed is the suitability of this crude motor-fuel fraction for refining. Probably the point can be studied most conveniently by determining the percentage of "unsaturation" ²⁶ of this fraction.

For the purpose of this discussion the term "unsaturates" or "unsaturated hydrocarbons" is taken to mean that part of the oil soluble in concentrated sulphuric acid under the conditions of the test mentioned above. If used with this understanding, the term "unsaturates" probably includes the diolefins, and part, but not necessarily all, of the olefins. Some of the branched chain hydro-

²⁶ Specifications for petroleum products. Bureau of Mines Tech. Paper 323, 1923, p. 87, describes a method for determining unsaturation of petroleum products. The method for shale-oil distillates involves the use of a bottle with different dimensions and three volumes of acid for one of oil. The acid and oil must be kept cool while mixing.

carbons also may possibly be sulphonated under the conditions of the test. The acid also removes basic compounds, and many shale naphthas contain up to 10 per cent of nitrogen bases.

The term "saturated hydrocarbons" or "saturates" as used here signifies that part of the oil not soluble in sulphuric acid under the conditions of testing.

Just what part percentage of unsaturation will play in the refining of shale oil is not clear at present. Certainly under commercial practice the refining loss of any distillate will be much less than the absolute unsaturation percentage. However, it is assumed that the relative refining losses will be approximately proportional to the respective percentages of unsaturated hydrocarbons in the oils under consideration. For example, a shale oil containing 50 per cent of unsaturated hydrocarbons will on refining suffer a greater loss than will an oil containing only 25 per cent of unsaturates, but the loss may not be twice as great.

There is much to be learned regarding the refining losses of shale oils and the relationship between refining loss and unsaturation, but for the present and for purposes of comparison it seems most feasible to rate qualities of shale oils largely on the basis of their percentages of unsaturated hydrocarbons.

It is very difficult to determine accurately by the usual test with sulphuric acid the percentage of unsaturated hydrocarbons in a crude shale oil. The acid tars produced under this treatment are usually very viscous and difficult to separate from the oil. On the other hand, it is relatively simple to determine this unsaturation percentage in the tops and to obtain with tops closely agreeing results in successive unsaturation tests. Therefore, and again tentatively, the Bureau of Mines compares the qualities of shale oils on the basis of *relative percentages of unsaturates in the tops from the oils*.

Another sound reason for classifying oils on this basis is brought out by the analyses. A low percentage of unsaturated hydrocarbons in the tops generally indicates more desirable properties in the remaining fractions of the oil. In some oils, with the reduction of unsaturates the percentage of solid paraffin wax in the oil seems to decrease, but even in this case the crystalline structure of the wax often is altered in a direction that would apparently facilitate its recovery.

Knowing the amount of tops yielded by the crude oil and the quality of the tops as indicated by the percentage of unsaturates (also the percentage of saturated hydrocarbons, since this is obtained by subtracting percentage of unsaturates from 100), an index number tentatively evaluating the shale oil may be readily calculated. If *percentage of tops* from a shale oil be multiplied by the percentage

of *saturated hydrocarbons* in the tops, the product seems to be a good numerical index of the value of that oil. If the possible oil yield of the shale has been determined by a standard method, then the crude-oil yield of the shale, calculated to gallons per ton, multiplied by this index number, gives another number which may be used as an index of the value of the shale for the production of motor fuel.

It should be repeated that this system of classification does not take into consideration the value of the oil other than as a producer of motor fuel, although, as noted, a high yield of the motor-fuel fraction coupled with a low percentage of unsaturates in this fraction is reflected in the quality of the other fractions from the oil. Likewise, the system does not take into consideration the absolute volatility range of the motor-fuel fraction or the percentage of sulphur in the oil. These matters require further study, and perhaps the method will require revision when these factors are properly taken into account. Nevertheless, the method has been found extremely useful to the bureau's chemists in their oil-shale investigations, especially in studying a particular shale.

An application of the method described above may be given. Consider two shales, one yielding 50 and the other 35 gallons of crude oil to the ton. The oil from the first yields 38 per cent of tops containing 55 per cent of unsaturates (45 per cent saturated), and the other 35 per cent of tops containing 40 per cent of unsaturates (60 per cent saturated). First, the oils are evaluated:

Relative value of oil from shale No. 1 = $38 \times 0.45 = 17.10$.

Relative value of oil from shale No. 2 = $35 \times 0.60 = 21.00$

If the shales themselves are to be evaluated, the numbers obtained as above are multiplied by the respective crude-oil yields of the two shales:

Relative value of shale No. 1 = $17.10 \times 50 = 855$.

Relative value of shale No. 2 = $21.00 \times 35 = 735$.

Investigations now under way may develop a method for evaluating the heavier fractions of shale oils. The Conradson carbon-residue percentages of the vacuum residuums may possibly be used for this purpose, once the relationship between this and the quality of the heavier fractions has been determined.

Distillation analyses of typical shale oils are presented in Table 25, following. Sample 101 is oil made from Scottish shale by Bureau of Mines assay retorts and sample 102 is oil produced by commercial retorts in Scotland. The analysis of a typical Pennsylvania crude oil (sample 1107) is included for purposes of comparison between typical shale oils and a typical high-grade petroleum. The other analyses are discussed and some conclusions drawn from them in the pages following the table.

TABLE 25.—*Distillation analyses of shale oil.*

Sample No. 305a.

Shale from Grand Valley, Colo.
 Specific gravity of oil, 0.891.
 Viscosity at 130° F., 45.
 Percentage of tops, 53.2.
 Percentage of water, trace.
 Retort: Horizontal rotary. First frac-
 tion of oil.*

Gravity, ° A. P. I., 27.3.
 Setting point, 18° C. (65° F.).
 Unsaturation of tops, per cent, 40.7.
 Index No. of oil, 31.55.

Distillation, Bureau of Mines, Hempel method.

Air distillation: Barometer, 636 mm. First drop, 47° C. (117° F.).

| Temperature, ° C. | Temperature, ° F. | Per cent
cut. | Sum per
cent. | Specific
gravity
of cut. | ° A. P. I.
of cut. | Viscosity
at 130° F. | Setting
point, ° F. |
|-------------------|-------------------|------------------|------------------|--------------------------------|-----------------------|-------------------------|------------------------|
| Up to 50..... | Up to 122..... | Tr. | ----- | | | | |
| 50-75..... | 122-167..... | 1.2 | 1.2 | 0.728 | 62.9 | | |
| 75-100..... | 167-212..... | 1.9 | 3.1 | | | | |
| 100-125..... | 212-257..... | 4.0 | 7.1 | .758 | 55.2 | | |
| 125-150..... | 257-302..... | 6.1 | 13.2 | .782 | 49.6 | | |
| 150-175..... | 302-347..... | 7.9 | 21.1 | .810 | 43.2 | | |
| 175-200..... | 347-392..... | 6.3 | 27.4 | .834 | 38.2 | | |
| 200-225..... | 392-437..... | 8.1 | 35.5 | .855 | 34.0 | | |
| 225-250..... | 437-482..... | 8.5 | 44.0 | .874 | 30.4 | | |
| 250-275..... | 482-527..... | 9.2 | 53.2 | .894 | 28.6 | | |

Vacuum distillation at 40 mm.

| Temperature, ° C. | Temperature, ° F. | Per cent
cut. | Sum per
cent. | Specific
gravity
of cut. | ° A. P. I.
of cut. | Viscosity
at 130° F. | Setting
point, ° F. |
|-------------------|-------------------|------------------|------------------|--------------------------------|-----------------------|-------------------------|------------------------|
| Up to 200..... | Up to 392..... | 3.6 | 3.6 | 0.891 | 27.3 | 39 | Below 31 |
| 200-225..... | 392-437..... | 6.1 | 9.7 | .907 | 24.5 | 46 | 39 |
| 225-250..... | 437-482..... | 6.0 | 15.7 | .922 | 22.0 | 57 | 60 |
| 250-275..... | 482-527..... | 4.7 | 20.4 | .940 | 19.0 | 82 | 78 |
| 275-300..... | 527-572..... | 7.6 | 28.0 | .958 | 16.2 | 241 | 94 |

Residuum: Specific gravity 1.002; carbon residue, per cent, 9.4; residuum asphaltic.

* This is the first fraction of oil from the retort taken during a run in which a total of 3,572 c. c. of crude oil was produced. This fraction consisted of 1,033 c. c., or 28.9 per cent of the total oil produced.

TABLE 25.—*Distillation analyses of shale oil*—Continued.

Sample No. 305b.

Shale from Grand Valley, Colo.

Specific gravity of oil, 0.904.

Viscosity at 130° F., 45.

Percentage of tops, 41.9.

Percentage of water, trace.

Retort: Horizontal rotary. Second fraction of oil.*

Gravity, ° A. P. I., 25.0.

Setting point, 22° C. (72° F.).

Index No. of oil, 22.58.

Unsaturation of tops, per cent, 48.1.

Distillation, Bureau of Mines, Hempel method.

Air distillation: Barometer 628 mm. First drop 48° C. (118° F.).

| Temperature, ° C. | Temperature, ° F. | Per cent cut. | Sum per cent. | Specific gravity of cut. | ° A. P. I. of cut. | Viscosity at 130° F. | Setting point, ° F. |
|-------------------|-------------------|---------------|---------------|--------------------------|--------------------|----------------------|---------------------|
| Up to 50..... | Up to 122..... | Tr. | ----- | 0.716 | 66.1 | ----- | ----- |
| 50-75..... | 122-167..... | 1.1 | 1.1 | | | | |
| 75-100..... | 167-212..... | 1.7 | 2.8 | | | | |
| 100-125..... | 212-257..... | 2.5 | 5.3 | | | | |
| 125-150..... | 257-302..... | 4.7 | 11.0 | | | | |
| 150-175..... | 302-347..... | 4.8 | 15.8 | | | | |
| 175-200..... | 347-392..... | 5.3 | 21.1 | | | | |
| 200-225..... | 392-437..... | 6.0 | 27.1 | .858 | 33.4 | ----- | ----- |
| 225-250..... | 437-482..... | 6.6 | 33.7 | .879 | 29.5 | ----- | ----- |
| 250-275..... | 482-527..... | 8.2 | 41.9 | .898 | 26.1 | ----- | ----- |

Vacuum distillation at 40 mm.

| | | | | | | | |
|----------------|----------------|------|------|-------|------|-----|----------|
| Up to 200..... | Up to 392..... | 2.4 | 2.4 | 0.909 | 24.2 | 40 | Below 32 |
| 200-225..... | 392-437..... | 7.1 | 9.5 | .916 | 23.0 | 48 | 41 |
| 225-250..... | 437-482..... | 8.3 | 17.8 | .928 | 21.0 | 65 | 60 |
| 250-275..... | 482-527..... | 11.0 | 28.8 | .941 | 18.9 | 107 | 76 |
| 275-300..... | 527-572..... | 13.2 | 42.0 | .953 | 17.0 | 232 | 91 |

Residuum: Specific gravity 0.993; carbon residue, per cent, 8.1; residuum asphaltic.

* Second fraction from retort, 1,384 c. c. or 38.7 per cent of total oil produced in run. See footnote to sample 305a.

TABLE 25.—*Distillation analyses of shale oil—Continued.*

Sample No. 305c.

Shale from Grand Valley, Colo.

Specific gravity of oil, 0.9025.

Viscosity at 130° F., 40.

Percentage of tops, 48.0.

Percentage of water, trace.

Retort: Horizontal rotary. Third
fraction of oil.^a

Gravity, ° A. P. I., 25.0.

Setting point, 22° C. (71.6° F.).

Unsaturation of tops, per cent, 45.4.

Index No. of oil, 26.21.

Distillation, Bureau of Mines, Hempel method.

Air distillation: Barometer, 629 mm. First drop, 40° C. (104° F.).

| Temperature, ° C. | Temperature, ° F. | Per cent cut. | Sum per cent. | Specific gravity of cut. | ° A. P. I. of cut. | Viscosity at 130° F. | Setting point, ° F. |
|-------------------|-------------------|---------------|---------------|--------------------------|--------------------|----------------------|---------------------|
| Up to 50..... | Up to 122..... | 0.7 | 0.7 | 0.708 | 68.4 | | |
| 50-75..... | 122-167..... | 2.2 | 2.9 | | 58.7 | | |
| 75-100..... | 167-212..... | 2.2 | 5.1 | | 54.9 | | |
| 100-125..... | 212-257..... | 4.7 | 9.8 | | 47.8 | | |
| 125-150..... | 257-302..... | 5.9 | 15.7 | | 42.1 | | |
| 150-175..... | 302-347..... | 5.9 | 21.6 | | 37.6 | | |
| 175-200..... | 347-392..... | 5.6 | 27.2 | | 33.6 | | |
| 200-225..... | 392-437..... | 6.3 | 33.5 | .857 | 28.0 | | |
| 225-250..... | 437-482..... | 6.5 | 40.0 | .887 | 26.3 | | |
| 250-275..... | 482-527..... | 8.0 | 48.0 | .697 | | | |

Vacuum distillation at 40 mm.

| | | | | | | | |
|----------------|----------------|------|------|-------|------|-----|----------|
| Up to 200..... | Up to 392..... | 2.7 | 2.7 | 0.910 | 24.0 | 40 | Below 23 |
| 200-225..... | 392-437..... | 6.7 | 9.4 | .924 | 21.6 | 50 | 41 |
| 225-250..... | 437-482..... | 8.1 | 17.5 | .937 | 19.5 | 63 | 59 |
| 250-275..... | 482-527..... | 8.9 | 26.4 | .951 | 17.3 | 108 | 78 |
| 275-300..... | 527-572..... | 10.4 | 36.8 | .962 | 15.6 | 242 | 92 |

Residuum: Specific gravity, 1.003; carbon residue, per cent, 14.1; residuum asphaltic.

^a Third fraction from retort, 432 c. c. or 12.2 per cent of total oil produced in run. See footnote to sample 305a.

TABLE 25.—*Distillation analyses of shale oil*—Continued.

Sample No. 305d.

Shale from Grand Valley, Colo.

Specific gravity of oil, 0.922.

Viscosity at 130° F., 46.

Percentage of tops, 42.2.

Percentage of water, trace.

Retort: Horizontal rotary. Fourth

and last fraction of oil.^a

Gravity, ° A. P. I., 22.0.

Setting point, 24° C. (75° F.).

Unsaturation of tops, per cent, 46.3.

Index No. of oil, 22.66.

Distillation, Bureau of Mines, Hempel method.

Air distillation: Barometer, 633 mm. First drop, 49° C. (120° F.).

| Temperature, ° C. | Temperature, ° F. | Per cent cut. | Sum per cent. | Specific gravity of cut. | ° A. P. I. of cut. | Viscosity at 130° F. | Setting point, ° F. |
|-------------------|-------------------|---------------|---------------|--------------------------|--------------------|----------------------|---------------------|
| Up to 50..... | Up to 122..... | | | | | | |
| 50-75..... | 122-167..... | 1.4 | 1.4 | 0.731 | 62.1 | | |
| 75-100..... | 167-212..... | 1.7 | 3.1 | | | | |
| 100-125..... | 212-257..... | 4.1 | 7.2 | .770 | 52.3 | | |
| 125-150..... | 257-302..... | 5.1 | 12.3 | .792 | 47.2 | | |
| 150-175..... | 302-347..... | 5.5 | 17.8 | .819 | 41.3 | | |
| 175-200..... | 347-392..... | 5.5 | 23.3 | .844 | 36.2 | | |
| 200-225..... | 392-437..... | 6.1 | 29.4 | .865 | 32.1 | | |
| 225-250..... | 437-482..... | 6.0 | 35.4 | .886 | 28.2 | | |
| 250-275..... | 482-527..... | 6.8 | 42.2 | .905 | 24.9 | | |

Vacuum distillation at 40 mm.

| | | | | | | | |
|----------------|----------------|------|------|-------|------|-----|----------|
| Up to 200..... | Up to 392..... | 3.0 | 3.0 | 0.916 | 23.0 | 40 | Below 32 |
| 200-225..... | 392-437..... | 7.8 | 10.8 | .935 | 19.8 | 49 | 41 |
| 225-250..... | 437-482..... | 8.5 | 19.3 | .944 | 18.4 | 63 | 55 |
| 250-275..... | 482-527..... | 10.8 | 30.1 | .966 | 15.0 | 121 | 82 |
| 275-300..... | 527-572..... | 9.0 | 39.1 | .980 | 12.9 | 239 | 93 |

Residuum: Specific gravity, 1.021; carbon residue, per cent, 13.5; residuum asphaltic.

^a Fourth fraction from retort, 723 c. c., or 20.2 per cent of total oil produced in run. See footnote to sample 305a.

TABLE 25.—*Distillation analyses of shale oil—Continued.*

Sample No. 306.

Shale from Grand Valley, Colo.
 Specific gravity of oil, 0.913.
 Viscosity at 130° F., 53.
 Percentage of tops, 34.8.
 Percentage of water, trace.
 Percentage of sulphur in oil, 0.60.
 Retort: Assay. Rate of oil production, 1 c. c. per minute.*

Gravity, ° A. P. I., 28.5.
 Setting point, 22° C. (72° F.).
 Unsaturation of tops, per cent, 44.4.
 Index No. of oil, 19.85.
 Percentage of nitrogen in oil, 2.07.

Distillation, Bureau of Mines, Hempel method.

Air distillation: Barometer, 628 mm. First drop, 57° C. (135° F.).

| Temperature, ° C. | Temperature, ° F. | Per cent cut. | Sum per cent. | Specific gravity of cut. | ° A. P. I. of cut. | Viscosity at 130° F. | Setting point, ° F. |
|-------------------|-------------------|---------------|---------------|--------------------------|--------------------|----------------------|---------------------|
| Up to 50..... | Up to 122..... | | | | | | |
| 50-75..... | 122-167..... | 0.3 | 0.3 | 0.745 | 58.4 | | |
| 75-100..... | 167-212..... | 1.0 | 1.3 | | | | |
| 100-125..... | 212-257..... | 2.2 | 3.5 | | | | |
| 125-150..... | 257-302..... | 4.1 | 7.6 | .782 | 49.5 | | |
| 150-175..... | 302-347..... | 4.6 | 12.2 | .810 | 43.2 | | |
| 175-200..... | 347-392..... | 4.6 | 16.8 | .837 | 37.6 | | |
| 200-225..... | 392-437..... | 5.4 | 22.2 | .850 | 35.0 | | |
| 225-250..... | 437-482..... | 5.6 | 27.8 | .870 | 31.1 | | |
| 250-275..... | 482-527..... | 7.0 | 34.8 | .890 | 27.5 | | |

Vacuum distillation at 40 mm.

| Up to 200..... | Up to 392..... | 4.3 | 4.3 | 0.902 | 25.4 | 39 | Below 32 |
|----------------|----------------|------|------|-------|------|-----|----------|
| 200-225..... | 392-437..... | 8.2 | 12.5 | .920 | 22.3 | 48 | 45 |
| 225-250..... | 437-482..... | 10.7 | 23.2 | .938 | 19.4 | 82 | 73 |
| 250-275..... | 482-527..... | 8.2 | 31.4 | .953 | 17.0 | 148 | 90 |
| 275-300..... | 527-572..... | 11.3 | 42.7 | .962 | 15.6 | 241 | 99 |

Residuum: Specific gravity, 0.992; carbon residue, per cent, 11.6; residuum asphaltic—softens at 35° C. (95° F.).

* Yield of crude oil from shale, 35.73 gallons per ton; scrubber naphtha, 1.17 gallons per ton; specific gravity scrubber naphtha, 0.714 (66.7° A. P. I.), unsaturation, 31.8 per cent. Scrubber naphtha not included in calculating index number.

TABLE 25.—*Distillation analyses of shale oil*—Continued.

Sample No. 307.

Shale from Grand Valley, Colo.
 Specific gravity of oil, 0.901.
 Viscosity at 130° F., 50.
 Percentage of tops, 39.0.
 Percentage of water, trace.
 Percentage of sulphur in oil, 0.53.
 Retort: Assay. Rate of oil production, 0.2 c. c. per minute.^a

Gravity, ° A. P. I., 25.6.
 Setting point, 28° C. (79° F.).
 Unsaturation of tops, per cent, 39.1.
 Index No. of oil, 23.75.
 Percentage of nitrogen in oil, 1.79.

Distillation, Bureau of Mines, Hempel method.

Air distillation: Barometer, 629 mm. First drop, 60° C. (122° F.).

| Temperature, ° C. | Temperature, ° F. | Per cent cut. | Sum per cent. | Specific gravity of cut. | ° A. P. I. of cut. | Viscosity at 130° F. | Setting point, ° F. |
|-------------------|-------------------|---------------|---------------|--------------------------|--------------------|----------------------|---------------------|
| Up to 50..... | Up to 122..... | | | | | | |
| 50-75..... | 122-167..... | 1.2 | 1.2 | 0.712 | 67.2 | | |
| 75-100..... | 167-212..... | 1.1 | 2.3 | | | | |
| 100-125..... | 212-257..... | 2.8 | 5.1 | .750 | 57.2 | | |
| 125-150..... | 257-302..... | 4.4 | 9.5 | .779 | 50.1 | | |
| 150-175..... | 302-347..... | 4.8 | 14.3 | .803 | 44.7 | | |
| 175-200..... | 347-392..... | 5.3 | 19.6 | .820 | 41.1 | | |
| 200-225..... | 392-437..... | 5.6 | 25.2 | .846 | 35.8 | | |
| 225-250..... | 437-482..... | 6.4 | 31.6 | .868 | 31.5 | | |
| 250-275..... | 482-527..... | 7.4 | 39.0 | .885 | 28.4 | | |

Vacuum distillation at 40 mm.

| Up to 200..... | Up to 392..... | 4.3 | 4.3 | 0.895 | 26.6 | 41 | Below 32 |
|----------------|----------------|------|------|-------|------|-----|----------|
| 200-225..... | 392-437..... | 9.4 | 13.7 | .910 | 24.0 | 50 | 43 |
| 225-250..... | 437-482..... | 9.1 | 22.8 | .928 | 21.0 | 70 | 65 |
| 250-275..... | 482-527..... | 10.4 | 33.2 | .944 | 18.4 | 110 | 81 |
| 275-300..... | 527-572..... | 11.0 | 44.2 | .955 | 16.7 | 190 | 96 |

Residuum: Specific gravity, 0.981; carbon residue, per cent, 6.1; residuum asphaltic—softens at about 40° C (104° F.).

^a Yield of crude oil from shale, 34.66 gallons per ton; scrubber naphtha, 0.80 gallon per ton. Not enough scrubber naphtha for unsaturation or gravity determinations. Scrubber naphtha not included in calculating index number.

TABLE 25.—*Distillation analyses of shale oil—Continued.*

Sample No. 208a.

Shale from Soldier Summit, Utah.
 Specific gravity of oil, 0.922.
 Viscosity at 140° F., 68.
 Percentage of tops, 24.79.
 Percentage of water, trace.
 Retort: Laboratory vertical. Steam
 used, ratio 1:1. Oil is composite of
 several runs.

Gravity, ° A. P. I., 22.0.
 Setting point, 87.5° C. (99° F.).
 Unsaturation of tops, per cent, 46.3.
 Index No. of oil, 13.80.

Distillation, Bureau of Mines, Hempel method.

Air distillation: Barometer, 637 mm. First drop, 55° C. (131° F.).

| Temperature, ° C. | Temperature, ° F. | Per cent cut. | Sum per cent. | Specific gravity of cut. | ° A. P. I. of cut. | Viscosity at 140° F. | Setting point, ° F. |
|-------------------|-------------------|---------------|---------------|--------------------------|--------------------|----------------------|---------------------|
| Up to 50..... | Up to 122..... | | | | | | |
| 50-75..... | 122-167..... | 0.67 | 0.67 | 0.791 | 47.4 | | |
| 75-100..... | 167-212..... | .67 | 1.34 | | | | |
| 100-125..... | 212-257..... | .84 | 2.18 | | | | |
| 125-150..... | 257-302..... | .84 | 3.02 | | | | |
| 150-175..... | 302-347..... | 2.08 | 5.70 | .814 | 42.3 | | |
| 175-200..... | 347-392..... | 2.06 | 8.38 | .830 | 39.0 | | |
| 200-225..... | 392-437..... | 3.96 | 12.31 | .840 | 37.0 | | |
| 225-250..... | 437-482..... | 5.53 | 17.84 | .858 | 35.4 | | |
| 250-275..... | 482-527..... | 6.95 | 24.79 | .899 | 31.3 | | |

Vacuum distillation at 40 mm.

| | | | | | | | |
|----------------|----------------|------|-------|-------|------|----|----|
| Up to 200..... | Up to 392..... | 1.16 | 1.16 | 0.882 | 28.9 | 41 | 70 |
| 200-225..... | 392-437..... | 4.80 | 5.96 | | | | |
| 225-250..... | 437-482..... | 5.84 | 11.80 | | | | |
| 250-275..... | 482-527..... | 8.96 | 20.76 | | | | |
| 275-300..... | 527-572..... | 5.77 | 26.53 | .936 | 19.7 | 80 | 99 |

Residuum: Specific gravity, 0.991.

• Asphaltic; no definite setting point.

TABLE 25.—*Distillation analyses of shale oil—Continued.*

Sample No. 203b.

Shale from Soldier Summit, Utah.
 Specific gravity of oil, 0.878.
 Viscosity at 140° F., 39.
 Percentage of tops, 39.95.
 Percentage of water, nil.
 Oil is part of sample 203a, distilled
 to coke once.

Gravity, ° A. P. I., 29.7.
 Setting point, 24° C. (75° F.).
 Unsaturation of tops, per cent, 43.5.
 Index No. of oil, 22.59.

Distillation, Bureau of Mines, Hempel method.

Air distillation: Barometer, 639 mm. First drop, 30° C. (86° F.).

| Temperature, ° C. | Temperature, ° F. | Per cent cut. | Sum per cent. | Specific gravity of cut. | ° A. P. I. of cut. | Viscosity at 140° F. | Setting point, ° F. |
|-------------------|-------------------|---------------|---------------|--------------------------|--------------------|----------------------|---------------------|
| Up to 50..... | Up to 122..... | 0.75 | 0.75 | 0.738 | 60.2 | | |
| 50-75..... | 122-167..... | .96 | 1.71 | | | | |
| 75-100..... | 167-212..... | .71 | 2.42 | | | | |
| 100-125..... | 212-257..... | 1.93 | 4.35 | | | | |
| 125-150..... | 257-302..... | 3.18 | 7.43 | | | | |
| 150-175..... | 302-347..... | 5.30 | 10.73 | .780 | 49.9 | | |
| 175-200..... | 347-392..... | 4.42 | 15.15 | .808 | 43.6 | | |
| 200-225..... | 392-437..... | 6.70 | 21.85 | .820 | 41.1 | | |
| 225-250..... | 437-482..... | 7.35 | 29.20 | .838 | 37.4 | | |
| 250-275..... | 482-527..... | 10.75 | 39.95 | .853 | 34.4 | | |
| | | | | .872 | 30.8 | | |

Vacuum distillation at 40 mm.

| | | | | | | | |
|----------------|----------------|-------|-------|-------|------|----|----|
| Up to 200..... | Up to 392..... | 4.60 | 4.60 | 0.881 | 29.1 | 39 | |
| 200-225..... | 392-437..... | 8.70 | 13.30 | .887 | 28.0 | 42 | |
| 225-250..... | 437-482..... | 8.75 | 22.05 | .897 | 26.3 | 48 | 64 |
| 250-275..... | 482-527..... | 9.50 | 31.55 | .926 | 21.3 | 84 | 84 |
| 275-300..... | 527-572..... | 10.35 | 41.90 | .936 | 19.7 | 70 | 96 |

Residuum: Specific gravity, 0.954; setting point, 43° C. (109° F.).

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TABLE 25.—*Distillation analyses of shale oil—Continued.*

Sample No. 208c.

Shale from Soldier Summit, Utah.
 Specific gravity of oil, 0.865.
 Viscosity at 140° F., 35.
 Percentage of tops, 46.42.
 Percentage of water, nil.
 Oil is part of sample 203a, distilled
 to coke twice.

Gravity, ° A. P. I., 82.1.
 Setting point, 20° C. (68° F.).
 Unsaturation of tops, per cent, 43.2.
 Index No. of oil, 26.37.

Distillation, Bureau of Mines, Hempel method.

Air distillation: Barometer, 637 mm. First drop 27° C. (99° F.)

| Temperature, ° C. | Temperature, ° F. | Per cent cut. | Sum per cent. | Specific gravity of cut. | ° A. P. I. of cut. | Viscosity at 140° F. | Setting point, ° F. |
|-------------------|-------------------|---------------|---------------|--------------------------|--------------------|----------------------|---------------------|
| Up to 50..... | Up to 122..... | 0.33 | 0.33 | 0.743 | 58.9 | | |
| 50-75..... | 122-167..... | 2.33 | 2.66 | | | | |
| 75-100..... | 167-212..... | 1.93 | 4.59 | | | | |
| 100-125..... | 212-257..... | 1.93 | 6.52 | | | | |
| 125-150..... | 257-302..... | 3.00 | 9.52 | .798 | 45.8 | | |
| 150-175..... | 302-337..... | 4.67 | 14.19 | .805 | 44.3 | | |
| 175-200..... | 337-372..... | 5.26 | 19.45 | .823 | 40.4 | | |
| 200-225..... | 372-407..... | 7.44 | 26.89 | .840 | 37.0 | | |
| 225-250..... | 407-442..... | 9.36 | 36.25 | .851 | 34.8 | | |
| 250-275..... | 442-527..... | 9.17 | 45.42 | .870 | 31.1 | | |

Vacuum distillation at 40 mm.

| | | | | | | | |
|----------------|----------------|-------|-------|-------|------|----|-----|
| Up to 200..... | Up to 392..... | 5.60 | 5.60 | 0.880 | 20.3 | 37 | |
| 200-225..... | 392-437..... | 9.40 | 15.00 | .896 | 28.2 | 39 | |
| 225-250..... | 437-482..... | 15.20 | 30.20 | .901 | 25.6 | 44 | 68 |
| 250-275..... | 482-527..... | 13.05 | 43.25 | .920 | 22.3 | 55 | 82 |
| 275-300..... | 527-572..... | 7.30 | 50.55 | .932 | 20.3 | 77 | 103 |

Residuum: Specific gravity, 0.968; asphaltic; no definite setting point.

TABLE 25.—*Distillation analyses of shale oil*—Continued.

Sample No. 214.

Shale from Soldier Summit, Utah.
 Specific gravity of oil, 0.908.
 Viscosity at 130° F., 45.
 Percentage of tops, 34.2.
 Percentage of water, trace.
 Percentage of sulphur, 1.89.

Retort: Assay. Rate of oil production, 1 c. c. per minute.*
 Gravity, ° A. P. I., 24.3.
 Setting point, 30° C. (86° F.).
 Unsaturation of tops, per cent, 40.5.
 Index No. of oil, 20.34.

Distillation, Bureau of Mines, Hempel method.

Air distillation: Barometer, 630 mm. First drop, 55° C. (131° F.).

| Temperature, °C. | Temperature, °F | Per cent cut. | Sum per cent. | Specific gravity of cut. | °A. P. I. of cut. | Viscosity at 130° F. | Setting point, °F. |
|------------------|-----------------|---------------|---------------|--------------------------|-------------------|----------------------|--------------------|
| Up to 50..... | Up to 122..... | | | | | | |
| 50-75..... | 122-167..... | 0.6 | 0.6 | 0.741 | 59.5 | | |
| 75-100..... | 167-212..... | 1.3 | 1.9 | | | | |
| 100-125..... | 212-257..... | 2.4 | 4.3 | .777 | 50.6 | | |
| 125-150..... | 257-302..... | 3.2 | 7.5 | .791 | 47.4 | | |
| 150-175..... | 302-347..... | 4.5 | 12.0 | .817 | 41.7 | | |
| 175-200..... | 347-392..... | 4.4 | 16.4 | .830 | 39.9 | | |
| 200-225..... | 392-437..... | 5.2 | 21.6 | .845 | 36.0 | | |
| 225-250..... | 437-482..... | 5.6 | 27.2 | .863 | 32.5 | | |
| 250-275..... | 482-527..... | 7.0 | 34.2 | .879 | 29.5 | | |

Vacuum distillation at 40 mm.

| | | | | | | | |
|----------------|----------------|------|------|-------|------|-----|-----|
| Up to 200..... | Up to 392..... | 3.6 | 3.6 | 0.880 | 29.3 | 38 | 83 |
| 200-225..... | 392-437..... | 9.1 | 12.7 | .900 | 25.7 | 42 | 84 |
| 225-250..... | 437-482..... | 8.5 | 21.2 | .920 | 22.3 | 48 | 73 |
| 250-275..... | 482-527..... | 8.5 | 29.7 | .938 | 19.4 | 62 | 86 |
| 275-300..... | 527-572..... | 12.9 | 42.6 | .956 | 16.5 | 113 | 102 |

Residuum: Specific gravity, 0.990; carbon residue, per cent, 9.0; residuum, waxy; setting point not determined.

* Yield of crude oil from shale, 49.5 gallons per ton; scrubber naphtha, 1.11 gallons per ton. Not enough scrubber naphtha for unsaturation and gravity tests. Scrubber naphtha not included in calculating index number.

TABLE 25.—*Distillation analyses of shale oil—Continued.*

Sample No. 215.

Shale from Soldier Summit, Utah.
 Specific gravity of oil, 0.890.
 Viscosity at 130° F., 42.
 Percentage of tops, 38.5.
 Percentage of water, none.
 Percentage of sulphur, 1.64.

Retort: Assay. Rate of oil production, 0.2 c. c. per minute.*
 Gravity, ° A. P. I., 27.5.
 Setting point, 26.5° C. (80° F.).
 Unsaturation of tops, per cent, 39.6.
 Index No. of oil, 23.25.

Distillation, Bureau of Mines, Hempel method.

Air distillation: Barometer, 634 mm. First drop, 53° C. (127° F.).

| Temperature, °C. | Temperature, °F. | Per cent cut. | Sum per cent. | Specific gravity of cut. | °A. P. I. of cut. | Viscosity at 130° F. | Setting point, °F. |
|------------------|------------------|---------------|---------------|--------------------------|-------------------|----------------------|--------------------|
| Up to 50..... | Up to 122..... | | | | | | |
| 50-75..... | 122-167..... | 0.9 | 0.9 | 0.735 | 61.0 | | |
| 75-100..... | 167-212..... | 1.4 | 2.3 | | | | |
| 100-125..... | 212-257..... | 3.3 | 5.6 | .774 | 51.3 | | |
| 125-150..... | 257-302..... | 4.3 | 9.9 | .792 | 47.2 | | |
| 150-175..... | 302-347..... | 4.4 | 14.3 | .809 | 43.4 | | |
| 175-200..... | 347-392..... | 4.7 | 19.0 | .830 | 39.0 | | |
| 200-225..... | 392-437..... | 5.4 | 24.4 | .845 | 36.0 | | |
| 225-250..... | 437-482..... | 6.6 | 31.0 | .862 | 32.7 | | |
| 250-275..... | 482-527..... | 7.5 | 38.5 | .878 | 29.7 | | |

Vacuum distillation at 40 mm.

| | | | | | | | |
|----------------|----------------|------|------|-------|------|-----|---------|
| Up to 200..... | Up to 392..... | 5.5 | 5.5 | 0.890 | 27.5 | 36 | Bottoms |
| 200-225..... | 392-437..... | 9.2 | 14.7 | .902 | 25.4 | 45 | |
| 225-250..... | 437-482..... | 9.9 | 24.6 | .920 | 22.3 | 51 | |
| 250-275..... | 482-527..... | 12.2 | 36.8 | .941 | 18.9 | 65 | |
| 275-300..... | 527-572..... | 10.2 | 47.0 | .952 | 17.1 | 104 | |

Residuum: Specific gravity, 0.977; carbon residue, per cent, 6.4; residuum, waxy; setting point, 61° (140° F.).

* Yield of crude oil from shale, 46.61 gallons per ton; scrubber naphtha, 1.32 gallons per ton. Scrubbers not included in calculating index number.

TABLE 25.—*Distillation analyses of shale oil—Continued.*

Sample No. 801.

Shale from Indiana.

Specific gravity of oil, 0.940.

Viscosity at 130° F., 48.

Percentage of tops, 47.5.

Percentage of water, trace.

Percentage of sulphur, 2.21.

Retort: Assay. Oil is a mixture from several miscellaneous samples; produced at rate of 1 c. c. per minute.

Gravity, ° A. P. I., 19.0.

Setting point, below 0° C. (32° F.).

Unsaturation of tops, per cent, 37.7.

Index No. of oil, 29.59.

Distillation, Bureau of Mines, Hempel method.

Air distillation: Barometer, 622 mm. First drop, 42° C. (108° F.).

| Temperature, °C. | Temperature, ° F. | Per cent cut. | Sum per cent. | Specific gravity of cut. | °A. P. I. of cut. | Viscosity at 130° F. | Setting point, °F. |
|------------------|-------------------|---------------|---------------|--------------------------|-------------------|----------------------|--------------------|
| Up to 50..... | Up to 122..... | 0.7 | 0.7 | 0.707 | 68.6 | ----- | ----- |
| 50-75..... | 122-167..... | 1.6 | 2.3 | | | | |
| 75-100..... | 167-212..... | 2.6 | 4.9 | .743 | 68.9 | ----- | ----- |
| 100-125..... | 212-257..... | 4.6 | 9.5 | .768 | 52.7 | ----- | ----- |
| 125-150..... | 257-302..... | 4.7 | 14.2 | .795 | 46.5 | ----- | ----- |
| 150-175..... | 302-347..... | 5.7 | 19.9 | .821 | 40.9 | ----- | ----- |
| 175-200..... | 347-392..... | 6.2 | 26.1 | .847 | 35.6 | ----- | ----- |
| 200-225..... | 392-437..... | 6.7 | 32.8 | .871 | 31.0 | ----- | ----- |
| 225-250..... | 437-482..... | 6.6 | 39.4 | .895 | 26.4 | ----- | ----- |
| 250-275..... | 482-527..... | 8.1 | 47.5 | .920 | 22.3 | ----- | ----- |

Vacuum distillation at 40 mm.

| | | | | | | | |
|----------------|----------------|-----|------|-------|-------|-----|----------|
| Up to 200..... | Up to 392..... | 2.1 | 2.1 | 0.982 | 20.3 | 40 | Below 32 |
| 200-225..... | 392-437..... | 9.2 | 11.3 | .980 | 15.9 | 52 | Below 32 |
| 225-250..... | 437-482..... | 5.4 | 16.7 | .982 | 12.6 | 95 | Below 32 |
| 250-275..... | 482-527..... | 8.3 | 25.0 | 1.004 | ----- | 280 | 46 |
| 275-300..... | 527-572..... | 8.6 | 33.6 | 1.019 | ----- | 962 | 67 |

Residuum: Specific gravity, 1.062; carbon residue, per cent, 22.0; residuum, asphaltic.

TABLE 25.—*Distillation analyses of shale oil—Continued.*

Sample No. 101.

Shale from Scotland.
 Specific gravity of oil, 0.864.
 Viscosity at 140° F., 41.
 Percentage of tops, 37.10.
 Percentage of water, 0.62.
 Percentage of sulphur, 0.84.

Retorted in Bureau of Mines.
 Assay retort, 2 hours retorting time.
 Gravity, ° A. P. I., 32.3.
 Setting point, 32° C. (90° F.).
 Unsaturation of tops, per cent, 32.0.
 Index No. of oil, 25.21.

Distillation, Bureau of Mines, Hempel method.

Air distillation: Barometer, 647 mm. First drop, 44° C. (111° F.).

| Temperature, °C. | Temperature, °F. | Per cent cut. | Sum per cent. | Specific gravity of cut. | *A. P. I. of cut. | Viscosity at 140° F. | Setting point, °F. |
|------------------|------------------|---------------|---------------|--------------------------|-------------------|----------------------|--------------------|
| Up to 50..... | Up to 122..... | 0.169 | 0.169 | 0.740 | 59.7 | ----- | ----- |
| 50-75..... | 122-167..... | .236 | .405 | | | | |
| 75-100..... | 167-212..... | 1.18 | 1.42 | | | | |
| 100-125..... | 212-257..... | 2.77 | 4.19 | | | | |
| 125-150..... | 257-302..... | 4.06 | 8.25 | | | | |
| 150-175..... | 302-347..... | 3.82 | 12.07 | .770 | 52.3 | ----- | ----- |
| 175-200..... | 347-392..... | 4.56 | 16.63 | .798 | 45.8 | ----- | ----- |
| 200-225..... | 392-437..... | 5.57 | 22.20 | .810 | 43.2 | ----- | ----- |
| 225-250..... | 437-482..... | 7.33 | 29.53 | .825 | 40.0 | ----- | ----- |
| 250-275..... | 482-527..... | 7.57 | 37.10 | .840 | 37.0 | ----- | ----- |
| | | | | .855 | 34.0 | ----- | ----- |

Vacuum distillation at 40 mm.

| | | | | | | | |
|----------------|----------------|------|-------|-------|------|----|-------|
| Up to 200..... | Up to 392..... | 2.10 | 2.10 | 0.868 | 31.5 | 38 | ----- |
| 200-225..... | 392-437..... | 4.38 | 6.48 | .872 | 30.8 | 41 | ----- |
| 225-250..... | 437-482..... | 5.88 | 12.36 | .874 | 30.4 | 43 | 70 |
| 250-275..... | 482-527..... | 7.30 | 19.66 | .894 | 26.8 | 50 | 84 |
| 275-300..... | 527-572..... | 9.45 | 29.11 | .898 | 26.1 | 60 | 99 |

Residuum: Specific gravity, 0.968; setting point, 45° C. (113° F.).

TABLE 25.—*Distillation analyses of shale oil*—Continued.

Sample No. 102.

Crude shale oil from Scotland.*
 Specific gravity of oil, 0.877.
 Viscosity at 140° F., 43.6.
 Percentage of tops, 35.44.
 Percentage of water, 0.13.

Gravity, ° A. P. I., 29.9.
 Setting point, 28° C. (82.4° F.).
 Unsaturation of tops, per cent, 31.9.
 Index No. of oil, 24.16.

Distillation, Bureau of Mines, Hempel method.

Air distillation: barometer, 644 mm. First drop, 49° C. (120° F.)

| Temperature, ° C. | Temperature, ° F. | Per cent cut. | Sum per cent. | Specific gravity of cut. | ° A. P. I. of cut. | Viscosity at 140° F. | Setting point, ° F. |
|-------------------|-------------------|---------------|---------------|--------------------------|--------------------|----------------------|---------------------|
| Up to 50..... | Up to 122..... | Tr. | | | | | |
| 50-75..... | 122-167..... | 0.13 | 0.13 | 0.760 | 54.7 | | |
| 75-100..... | 167-212..... | .32 | .45 | | | | |
| 100-125..... | 212-257..... | .99 | 1.44 | | | | |
| 125-150..... | 257-302..... | 1.66 | 3.10 | | | | |
| 150-175..... | 302-347..... | 3.00 | 6.10 | .785 | 48.8 | | |
| 175-200..... | 347-392..... | 6.12 | 12.22 | .807 | 43.8 | | |
| 200-225..... | 392-437..... | 8.10 | 20.32 | .826 | 39.8 | | |
| 225-250..... | 437-482..... | 6.65 | 26.97 | .842 | 36.6 | | |
| 250-275..... | 482-527..... | 8.47 | 35.44 | .857 | 33.6 | | |

Vacuum distillation at 40 mm.

| | | | | | | | |
|----------------|----------------|------|-------|-------|------|----|-------|
| Up to 200..... | Up to 392..... | 9.32 | 9.32 | 0.871 | 31.0 | 38 | |
| 200-225..... | 392-437..... | 5.27 | 14.59 | .881 | 29.1 | 41 | |
| 225-250..... | 437-482..... | 7.16 | 21.75 | .892 | 27.1 | 46 | 76 |
| 250-275..... | 482-527..... | 6.13 | 27.88 | .902 | 25.4 | 52 | 82 |
| 275-300..... | 527-572..... | 6.07 | 33.95 | .911 | 23.8 | 60 | 93 |

Residuum: Specific gravity, 0.957; setting point, 41° C. (106° F.).

* Represents approximately 90 per cent of the oil produced from the shale. The remainder, 10 per cent, is recovered as scrubber naphtha and is kept separate from the crude in refining. If the proper proportion of scrubber naphtha were added to the crude, the specific gravity of the mixture would be about 0.860 and the index number much higher.

TABLE 25.—*Distillation analysis of Pennsylvania crude petroleum*—Continued.

Sample No. 1107.

Specific gravity, 0.812.
Viscosity at 140° F., 38.8.
Percentage of tops, 52.78.
Percentage of water, trace.

Gravity, ° A. P. I., 42.8.
Setting point.*
Unsaturation of tops, per cent, 4.1.
Index No. of oil, 50.60.

Distillation, Bureau of Mines, Hempel method.

Air distillation: Barometer, 644 mm. First drop, 26° C. (79° F.).

| Temperature, ° C. | Temperature, ° F. | Per cent cut. | Sum per cent. | Specific gravity of cut. | ° A. P. I. of cut. | Viscosity at 140° F. | Setting point, ° F. |
|-------------------|-------------------|---------------|---------------|--------------------------|--------------------|----------------------|---------------------|
| Up to 50..... | Up to 122..... | 0.90 | 0.90 | 0.674 | 78.4 | | |
| 50-75..... | 122-167..... | 1.61 | 2.51 | | | | |
| 75-100..... | 167-212..... | 4.08 | 6.59 | .712 | 67.2 | | |
| 100-125..... | 212-257..... | 8.29 | 14.88 | .733 | 61.5 | | |
| 125-150..... | 257-302..... | 5.46 | 20.34 | .752 | 56.7 | | |
| 150-175..... | 302-347..... | 6.77 | 26.11 | .763 | 54.0 | | |
| 175-200..... | 347-392..... | 5.82 | 31.93 | .778 | 50.4 | | |
| 200-225..... | 392-437..... | 6.95 | 38.88 | .789 | 47.8 | | |
| 225-250..... | 437-482..... | 6.42 | 45.30 | .800 | 45.4 | | |
| 250-275..... | 482-527..... | 7.46 | 52.76 | .812 | 42.8 | | |

Vacuum distillation at 40 mm.

| | | | | | | | |
|----------------|----------------|------|-------|-------|------|----|----|
| Up to 200..... | Up to 392..... | 3.33 | 3.33 | 0.826 | 39.8 | 39 | |
| 200-225..... | 392-437..... | 7.75 | 11.08 | .832 | 38.6 | 40 | |
| 225-250..... | 437-482..... | 6.02 | 17.10 | .841 | 36.8 | 45 | 63 |
| 250-275..... | 482-527..... | 5.37 | 22.47 | .848 | 35.4 | 51 | 72 |
| 275-300..... | 527-572..... | 5.16 | 27.63 | .859 | 33.2 | 67 | 96 |

Residuum: Specific gravity, 0.882; setting point, 18° C. (65° F.).

* Not determined.

CONCLUSIONS BASED ON EXPERIMENTAL RETORTING OF OIL SHALES.

Below are summarized conclusions based on the experimental retorting of oil shales by the Bureau of Mines in its laboratories at Salt Lake City, Utah, in cooperation with the department of metallurgical research of the University of Utah, and at Boulder, Colo., in cooperation with the State of Colorado.

FRACTIONAL "EDUCTION."

When an oil shale is distilled it does not, as petroleum does, first yield a light oil, and as the temperature increases, heavier oil. Although there is a variation in the quality of oil produced during the course of retorting, it is not of the nature of fractional distillation, nor does it appear to be of such nature that it can be utilized in commercial work. There is no evidence to support the theory of fractional eduction. (See p. 46.) Analyses of samples 305a to 305d, Table 25, which were taken at different stages of retorting experiment, show clearly the nature of the variation referred to.

INFLUENCE OF RATE OF RETORTING.

Other conditions being constant, rapid retorting—rapid heating and rapid production of oil—favors high yields of oil; slow retorting gives a smaller yield, but the oil recovered is of better quality. This is shown in Table 26 below, and in greater detail by analyses of samples 214 and 215 and 306 and 307, Table 25.

TABLE 26.—*Summary of retorting experiments on oil shale from Soldier Summit, Utah—Assay retorts used.^a*

| Time of retorting, hours ^b | 2 | 4 | 6 | 8 | 10 | 20 |
|--|-------|-------|-------|-------|-------|-------|
| Oil yield, gallons per ton | 44.6 | 43.0 | 42.2 | 41.9 | 41.7 | 41.1 |
| Specific gravity of oil | 0.801 | 0.894 | 0.879 | 0.876 | 0.875 | 0.872 |
| Melting point of oil, °C. | 34.0 | 30.3 | 30.0 | 29.8 | 29.6 | 28.5 |
| Tops to 275° C., per cent. | 33.9 | 38.4 | 37.8 | 38.2 | 38.3 | 39.0 |
| Saturation of tops, per cent. | 61.0 | 63.8 | 65.0 | 65.3 | 65.7 | 67.3 |
| Index number ^c | 9.2 | 10.0 | 10.3 | 10.5 | 10.6 | 10.8 |
| Temperature to finish distillation, °C. | 526 | 467 | 458 | 454 | 452 | 449 |

^a Experimental work by L. C. Karrick, Bureau of Mines laboratory, Salt Lake City, Utah.

^b This is elapsed time, from beginning to end of oil production. In all tests the shales were completely retorted—they would yield no more oil if heated longer or to a higher temperature. Rate of oil production constant during each test.

^c This is second index discussed on p. 158. It is based on yield of crude oil, percentage of tops, and saturation of tops.

^d This is the temperature at which the last oil distilled from the shale

Particularly noteworthy in the above table are the data on temperature required to complete distillation. It is seen that when the shale is retorted rapidly, the production of oil is completed at a higher temperature than when the oil is produced slowly. These results were obtained with finely ground shale (minus one-fourth inch). If larger pieces were used, it is probable that even higher temperatures would be required to complete distillation in the same time.

Other results of rapid retorting, as compared with slow retorting, are: A slightly smaller percentage of spent shale, a smaller amount of fixed carbon in the spent shale, a larger amount of solid paraffin in the oil, and a larger amount of nitrogen in the oil and less in the spent shale.²⁷ As to the relative amounts of nitrogen in spent shale and of oil produced at different rates of retorting, the differences indicated as percentages in oil and spent shale are not particularly striking, but if calculated to actual weights of nitrogen in these two products, especially when it is considered that a greater weight of spent shale and smaller weight of oil is produced by slow retorting, the differences indicate that rate of retorting may have an important influence on the amount of ammonia producible from the spent shale and on the amount of basic nitrogen compounds

²⁷ This seemingly contradicts results obtained on Scottish shale by Bellby. See Bellby, G. T., Production of ammonia from the nitrogen of minerals: Jour. Soc. Chem. Ind., vol. 3, 1884, pp. 216–224; Jour. Roy. Soc. Arts, vol. 33, 1885, pp. 313–320. The writer and his associates, however, have reached the above conclusions as the results of work on representative American shales and a typical Scottish shale.

contained in the crude oil. Oils produced by slow retorting appear to contain less sulphur than those produced more rapidly, but more experimental work is required to establish definitely the effect of the rate of retorting on the distribution of sulphur.

It has been argued that in these experiments other factors than rate of heating or oil production were varying. This argument is sound, since when the oil is produced slowly, the velocity of vapors through and from the retort is lower, the average temperature to which the vapors are exposed is lower than when the rate of production is rapid and the degree of condensation and redistillation undoubtedly is different. However, these all depend on the rate of heating, and there appears to be no practical means of altering them in any given retort except by using steam or a scavenging gas. As is shown below, passing steam or nonoxidizing gases through the retort, at any given rate of oil production, causes the oil made to have the characteristics of one produced at a more rapid rate without the use of scavenging agents. The writer and his associates believe that slow distillation permits more complete conversion of the first-formed bitumen (see p. 44) into desirable oils, and when the shale is retorted slowly, that this conversion takes place at a lower temperature than under conditions of rapid retorting. They also believe that in the experiments performed, the possible differences in degree of condensation and redistillation, and the temperatures and the time during which the vapors were exposed to these temperatures after formation had relatively little to do with the amount and quality of oil produced. In larger retorts, however, where there would be longer contact of vapors with hot retort walls, longer exposure to possibly higher average temperatures, and probability of considerable refluxing condensation and redistillation on account of greater differences in temperature between different parts of the distilling mass of shale, these factors may have important influences. The above conclusions have resulted from work with several different types of small laboratory retorts.

INFLUENCE OF STEAM OR OTHER GASES.

Passage of steam or other nonoxidizing gases through the retort—all other conditions being constant—produces an effect similar to a more rapid rate of retorting without the use of these gases. That is, with any given retort and at any given rate of oil production the oil produced with steam will have more nearly the properties of an oil made at a more rapid rate without steam. The writer believes that in large retorts, particularly of the vertical type, for reasons given in the preceding section, the quality and amount of oil recovered would be relatively low unless the retorts were steamed.

IN RAPID DISTILLATION.

Steam in a retort adds volume to the vapors and thereby gives them a higher velocity, sweeping them from the retort so rapidly that reflux condensation is reduced to a minimum, and the period of exposure to high temperatures is very short. Mention has been made that the first effect of heat on the kerogen of the shale is to convert the kerogen into a bitumen. Undoubtedly this bitumen is fluid at the temperatures at which it is formed, and therefore it is probably held in the pores and on the surfaces of the shale lumps. If the temperature and the rate of distillation are high, the bitumen will rapidly decompose into oil vapors and gas. These will evolve rapidly from the shale and carry with them a certain amount of undecomposed bitumen, producing an oil containing a relatively large amount of undecomposed bitumen. This oil will be relatively large in quantity, poor in quality, and contain a relatively large percentage of nitrogen. The spent shale from such a distillation will contain relatively small amounts of fixed carbon and nitrogen. If, at any rate of retorting, steam or other scavenging gas is passed through the retort, the effect on the undecomposed bitumen will be that of a steam distillation, and more undecomposed bitumen will be distilled than if steam were not used. The steam will protect the oil vapors and undecomposed bitumen—in vapor form and mechanically carried—from further decomposition in passing through the retort.

IN SLOW DISTILLATION.

If, however, the shale is distilled slowly—at a slowly rising or even practically uniform temperature—conversion of kerogen into bitumen will take place slowly, and decomposition of the latter will go on slowly and at relatively low temperature; these conditions are favorable for the production of desirable oil hydrocarbons, though the absolute yield of crude oil will be smaller than when retorting is rapid. Thus, with slow retorting, decomposition of bitumen to desirable products will be more complete, and there will be less carrying over, or distillation, of undecomposed or partly decomposed bitumen. Possibly the bitumen produced at low temperatures will yield better products than those yielded by bitumens formed at somewhat higher temperatures under conditions of rapid oil production.

Thus the net result of slow distillation would be a somewhat smaller amount of oil than would be produced by rapid distillation, but the oil would be of better quality, except for amount of solid paraffins; the spent shale would contain a smaller amount of fixed carbon; and since, when the bitumen decomposes, it appears to

leave much of its nitrogen in the resultant coke, the oil would contain less, and the spent shale more, nitrogen.

In certain types of retorts, however, particularly those that are vertical, the vapors might be still more completely decomposed into gases and undesirable hydrocarbons, as mentioned before. Therefore the use of steam would be necessary, and if production and conversion of the bitumen were slow enough and took place at low enough temperature, the amount of bitumen steam distilled would probably not be enough materially to lower the quality of the resulting oil. In any event, when any definite amount of steam is used, the quality of the recovered oil improves as rate of distillation decreases, just as it does when no steam is used, although, as before mentioned, for any particular rate of retorting with the small experimental retorts thus far used, better oil is produced when steam is not used, and its quality decreases to some undetermined limit as the amount of steam is increased. At the same time the quantity recovered apparently increases in the same relative proportion.

The rate of distillation chosen and the amount of steam used will thus depend on the type and size of retort; the kind of oil desired whether nitrogen is wanted in the oil, or in the spent shale for ammonia production; the use to which the spent shale is to be put—if it is to be used in making water gas for fuel, the higher its content of fixed carbon the better—and how much the quality of the crude oil can be improved by subsequent refining. (See p. 182.) The proper combination will be determined largely by the cost sheet and will be that which will yield the largest profit per unit of shale treated.

INFLUENCE OF SIZE OF SHALE RETORTED.

Other conditions being constant, the size of the pieces of shale retorted has by itself little or no effect on the quantity and quality of the crude oil produced. If anything, the larger sizes yield a little less oil of somewhat better quality than do the smaller. However, when a piece of shale is broken or crushed, and various sizes are separated by screening, the larger sizes yield more and better oil than the smaller. For example, a Colorado shale was crushed, and separated into four sizes by screening. The largest size (plus 1 inch) yielded oil at the rate of about 38 gallons per ton, whereas the smallest (minus 10 mesh) yielded but 27 gallons. Though the rate of retorting was the same for both samples, the finer shale yielded oil of much lower quality.

If, however, large pieces of shale are crushed to small size, and all of the crushed material retorted, the yield and quality of oil will be practically the same as can be produced at the same rate or retorting from the larger pieces. In other words, when a shale is crushed, the

leaner parts break more readily than the richer. For this reason it may sometimes be advantageous to pay no attention to fines produced in mining or crushing, especially if the retort used is particularly adapted to treat shale in large lumps. Possibly the fine material will be used directly as fuel or gasified in producers.

INFLUENCE OF TYPE OF RETORT.

From the discussion on page 177, it is obvious that the design and the method of operation of the retort have an important effect on the amount and quality of the products made from any oil shale. A large amount of experimental work must be done before the effects of many factors of retort design and operation can be fully determined, but results already obtained show clearly that by slight changes of retort design or retorting conditions the character of the products obtained from a given shale can be materially changed. In large commercial retorts the character of the recovered products may largely be determined by what happens to the vapors after they are evolved from the shale. Exposure of vapors to excessive temperatures for any considerable time or contact with hot metallic surfaces is certain to have a harmful effect.

QUALITY OF OIL PRODUCED FROM DIFFERENT SHALES.

As noted on page 34, different shales retorted under identical conditions yield oils of different quality. This is more fully brought out by analyses of samples 214, 306, 101, and 801, Table 25. These oils were produced from different shales, as indicated in the table, but all were made under identical retorting conditions.

COKING DISTILLATION OF SHALE OILS.

In the experimental studies just discussed, it was noted that the highest yields of oils were produced from oil shales by rapid distillation, distillation with steam, or by a combination of both methods, but that the quality of the oils recovered in these distillations was lower than that of oils produced by slow distillation or without steam. The results obtained in the early stages of this investigation with the retorts then used led to the belief—since abandoned—that possibly the beneficial effect of the slow distillation was due to the condensation and gentle distillation to coke, within the retort itself, of the oil first produced from the shale. This suggested that the oil produced when highest recoveries were obtained might be redistilled to coke in an effort to improve the quality of the oil. Results of such redistillation are shown in Table 27, and the characteristics of the oils produced in one and two coking distillations are shown in Table 25 (samples 203a, 203b, and 203c).

TABLE 27.—*Effect of one and two coking distillations on crude shale oil.*

| Sample No. | OIL. | Specific gravity of oil. | Setting point, °F. | Per cent of tops. | Per cent of unsaturation tops. | Index No. of oil. | Specific gravity of residuum. | Nature of residuum. |
|------------|---|--------------------------|--------------------|-------------------|--------------------------------|-------------------|-------------------------------|---------------------|
| 203a | Utah crude shale oil produced with steam. | 0.922 | 99 | 24.79 | 46.3 | 13.30 | 0.991 | Asphaltic. |
| 203b | Same—once run to coke..... | .878 | 75 | 39.95 | 43.5 | * 22.59 | .964 | Waxy 8. P. 109° F. |
| 203c | Same—twice run to coke..... | .865 | 68 | 46.42 | 43.2 | * 26.37 | .968 | Asphaltic. |

* In comparing these oils consideration must be given to the quantity lost in the coking distillations. (See p. 182.)

It is interesting to note that the results of a single distillation to coke at atmospheric pressure were (1) the production of a large amount of still coke; (2) increase in the size and volatility of the motor fuel fraction; (3) a decrease in the amount of unsaturates in this fraction; and (4) an evident loss of solid paraffin wax, as indicated by the setting points of the vacuum fractions. A second redistillation produced only a small amount of coke and the changes in the recovered oil were not so great, but were in the same direction as those produced by the first distillation.

The first step in refining shale oil commercially in Scotland and Australia is to run the total crude from the retort condensers down to coke before fractionation and chemical treatments are begun. This procedure is said to facilitate the subsequent refining operations and to give finished products of better quality than those obtained from a crude not so handled.

There are several reasons why a coking distillation might improve the crude shale oils in the directions noted. The matter has been discussed by several investigators, many of whom are inclined to consider it as something new and mysterious. In the opinion of the writer there is little mysterious in the process, and certainly nothing novel, as it is common practice in Scotland.

When the oil is distilled to coke, many of the heavier unsaturated compounds in the oil probably break down, producing coke or carbon, and lighter unsaturated and saturated hydrocarbons. Consequently, the percentage of light oils in the recovered distillate would be increased, and the total percentage of unsaturates in that fraction would be increased, as the saturated oils originally in that fraction would be undecomposed, and to them would be added the light saturated and unsaturated oils produced by coking. Possibly, also, some of the heavier paraffins would break down to lighter paraffins and unsaturated bodies, and thus account for the decreased content of paraffin wax. Botkin²² believes that complex nitrogen

²² Botkin, C. W., A study of the saturated and unsaturated oils from shale: Chem. and Met. Eng., vol. 26, Mar. 1, 1922, pp. 398-401.

bodies in the oil, possibly undecomposed bitumen, are broken down by such redistillation and yield a distillate with a smaller percentage of nitrogen, most of the nitrogen remaining in the coke. As the percentage of nitrogen bases in the distillate would be smaller, and as the nitrogen bases on decomposing with the elimination of nitrogen might be expected to yield some saturated hydrocarbons, the loss on treating the distillate with sulphuric acid would be less than that of the original crude. The distillate would thus be more highly "saturated" (see p. 158) than the original crude. There is considerable experimental evidence to support this belief.

REDISTILLATION OF SHALE OILS FROM UTAH SHALE.

A composite sample of shale oil (sample 203a, Table 25) representing several runs under different conditions was used in the experimental work. The oil had been produced from oil shale from Soldier Summit, Utah, by retorting with steam and its quality was somewhat inferior to that usually produced from these shales by the bureau. Since then, many similar tests with similar results have been made on this and other oils.

This oil was sampled and an analytical distillation made. The balance of the sample was cut in two portions, the first of which was distilled to coke in a copper still at atmospheric pressure. The results of this distillation were as follows:

| | Per cent. |
|--------------------|-----------|
| Oil used..... | 100.0 |
| Oil recovered..... | 87.5 |
| Coke produced..... | 9.2 |
| Lost as gas..... | 3.3 |

The recovered distillate or "once-run crude" was subject to analytical distillation. (Sample 203b, Table 25.)

The second half of the original crude sample was then run to coke as above, with the following results:

| | Per cent. |
|--------------------|-----------|
| Oil used..... | 100.0 |
| Oil recovered..... | 87.0 |
| Coke produced..... | 9.1 |
| Lost as gas..... | 3.9 |

The oil recovered in this distillation, "once-run oil," was then placed in another copper-coking still, and immediately run to coke a second time, producing sample 203c, "twice-run oil."

The results of this second coking distillation are:

Oil used, on basis of "once-run oil," 100 per cent; on basis of original crude, 87 per cent.

Oil recovered, on basis of "once-run oil," 98.1 per cent; on basis of original crude, 85.3 per cent.

Coke produced on basis "once-run oil," 1.6 per cent.

Lost on gas on basis "once-run oil," 0.8 per cent.

It will be noted from Table 27 that the index numbers of the oils used in the redistillation experiments are as follows:

| | |
|--------------------|--------|
| Crude oil | 13. 30 |
| Once-run oil..... | 22. 59 |
| Twice-run oil..... | 28. 37 |

To obtain a true idea of the results of the coking distillations, these indices must be corrected for the losses incurred in the redistillations; 12.5 per cent was lost as gas and coke in the first distillation and 13 plus 1.9 in producing the twice-run oil.

If the index values given above are corrected for these losses, the numbers referring to the quality and quantity of the original crude become:

| | |
|--------------------|--------|
| Crude oil | 13. 30 |
| Once-run oil..... | 19. 77 |
| Twice-run oil..... | 22. 42 |

The effect of a coking distillation is of great importance in retort design and operation and deserves careful consideration. A coking distillation improves the quality of an oil for refining, and the oil will yield a larger net amount of desirable refined light products, not only because the preliminary coking distillation will increase the percentage of light distillates, but also because there will be a smaller loss in treating them. To offset this improvement is the loss through production of coke and gas. Another manner of obtaining an oil of good quality is to produce it more slowly from the shale, and this oil in turn can be further improved by coking it. The effects of redistillations or coking distillations can be regulated by the method of making the distillation—that is, by rate, degree of refluxing condensates, and use of steam, pressure, or vacuum.

In the opinion of the writer, the full application of the effects of coking distillations may result in the design of retorts to produce the oil from shale as rapidly as possible and with the use of steam, especially if no attempt is to be made to produce ammonia compounds. This will result in maximum yield of oil; its quality can be adjusted within fairly wide limits by suitable coking distillation. Slower retorting would produce results somewhat similar, but rather than reduce the capacity of expensive retorts for the sake of quality of oil it will probably be cheaper and give more economical final results to run the retorts at capacity and use relatively inexpensive coking stills for the production of quality. In effect this will be the same as reducing the investment in retorting equipment per barrel of oil produced. Stills will probably be cheaper than retorts and can be controlled more readily. An economic balance will probably be found in the rate of oil production and method of redistillation

which will lead to the highest economic returns. The by-product of coking distillations—coke—should be easily marketed.

SHALE GAS.

The fixed gases produced by distilling a given weight of different shales naturally differ in volume and composition. Likewise the amount and composition of the gas from a given shale differ with the conditions of retorting. Evolution of gas begins at a temperature below that of first oil production and continues during the oil-producing period. After all the oil has been produced, the evolution of gas drops off sharply and its heating value decreases, although if heating continues the shale will give off gas for a long time after oil production has ceased. The first gas evolved is often rich in hydrogen sulphide, but the evolution of this undesirable constituent drops off as the temperature of the shale increases. With some shales, however, as much as 20 per cent of the total gas produced by dry distillation is hydrogen sulphide, and all shale gases thus far examined have contained relatively large amounts of this substance.

Investigational work has not progressed far enough to permit reliable conclusions to be drawn regarding the amount of gas evolved from shales of different richness, or the change in gas composition through different retorting conditions. The Bureau of Mines has thus far worked only with gas produced by dry distillation, and has not yet examined gases from retorts that are steamed during distillation and in which some of the steam reacts with the carbon of the spent shale, producing a rather low-grade water gas. This gas mixes with the gases of destructive distillation that are evolved in another part of the retort. A large volume of gas of low heating value is produced from the relatively lean Scottish shales as a result of steaming in this manner. The composition and heating value of Scottish shale gas is shown on page 77. In Table 28, below, is shown the amount, composition, and heating value of the gases from two typical American shales. Sample No. 1 was produced from De Beque, Colo., shale during the retorting tests in which oil sample No. 306 (Table 25) was collected; and sample No. 2 was made from Soldier Summit, Utah shale when oil sample No. 214 was produced. The oil-production rate was constant and identical in each, and the retorts were heated to approximately the same final temperature for 30 minutes after oil production had ceased. Sample No. 3 also was produced from De Beque shale (the same shale used in producing sample No. 1), but the shale was retorted in a horizontal rotary retort used in the Boulder laboratory. The rate of oil production was comparable with that used when samples 1 and 2 were collected, and likewise the shale

was heated to about the same final temperature for 30 minutes after oil production stopped. The gas samples were scrubbed to remove hydrogen sulphide before analyses were made, and results are reported on hydrogen sulphide and air-free basis. The hydrogen sulphide percentages shown in parentheses were those of gas samples drawn directly from the gasometers before the gas was passed through scrubbers to remove hydrogen sulphide prior to complete analysis.

TABLE 28.—*Nature and composition of typical oil-shale gases produced by dry distillation.*

| Retort used. | Assay. | Assay. | Horizontal rotary. |
|--|------------------|------------------|--------------------|
| Gas sample No. | 1 | 2 | 3 |
| Shale | De Beque, lot 2. | Soldier Summit. | De Beque, lot 2. |
| Volume gas per ton of shale, cubic feet * | 787 | 1,219 | 1,120 |
| Gross heating value per cubic foot * | 807 | 765 | 1,079 |
| Analysis: | <i>Per cent.</i> | <i>Per cent.</i> | <i>Per cent.</i> |
| Carbon dioxide (CO ₂) | 17.1 | 11.0 | 13.8 |
| Hydrocarbons (C _n H _{2n}) | 6.4 | 3.1 | 11.6 |
| Carbon monoxide (CO) | 6.4 | 3.9 | 4.4 |
| Oxygen (O ₂) | 0.0 | 0.0 | 0.0 |
| Hydrogen (H ₂) | 35.8 | 47.5 | 30.2 |
| Methane (CH ₄) | 21.2 | 25.8 | 23.5 |
| Ethane (C ₂ H ₆) | 12.3 | 8.7 | 16.5 |
| Nitrogen (N ₂) | 0.8 | 0.0 | 0.0 |
| Total | 100.0 | 100.0 | 100.0 |
| Hydrogen sulphide | (1.4) | (6.1) | (1.3) |

* At 0° C., and 760 mm. pressure. Heating value determined with Junker's calorimeter.

The reader should note that the gasoline produced by the fractionation of most crude shale oil does not meet present-day specifications for motor fuels, because it is deficient in the more volatile hydrocarbons. It seems, however, that if the gases from shale distillation are scrubbed, the light hydrocarbons recovered may be blended with the gasoline from the crude oil to produce a mixture with a volatility range that will comply with specification requirements.

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Compiled by ELIZABETH H. BURROUGHS.

Selected by MARTIN J. GAVIN.

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The Bureau of Mines has published, as Reports of Investigations, Serial No. 2277, a selected bibliography of oil shales, prepared by E. H. Burroughs, and M. J. Gavin, and issued September, 1921. This bibliography lists the more noteworthy articles on oil shales that have appeared in the literature of the years 1915-1920, and gives references to standard publications that appeared previous to 1915. A total of 295 references is listed, and although the bibliography does not pretend to be complete, in the opinion of the writer, it contains references to the most important articles. From time to time this bibliography will be revised. Copies may be obtained by applying to the Director, Bureau of Mines, Washington, D. C., for the most recent revision of the Bibliography of Oil Shale.

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